24th World Mining Congress

MINING IN A WORLD OF INNOVATION

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

24th World Mining Congress **PROCEEDINGS**



INNOVATION IN MINING

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Vale Institute of Technology

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.

Luiz Mello

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.

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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ITV activities are focused across three key segments: Research, Education and Entrepreneurship, following the cycle through which scientific production evolves, creating practical application and generation of concrete benefits for the company. For further information visit: www.itv.org or send an e-mail: info.itv@itv.org.

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.

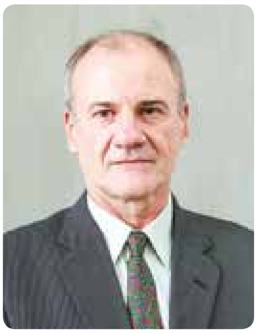
In addition to its positive impacts in the macroeconomic sphere in Brazil, mining has also become a driver of social development, particularly as it has a multiplier effect on other economic activities, contributing to the expansion of various production chains and consequently to the generation of jobs 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



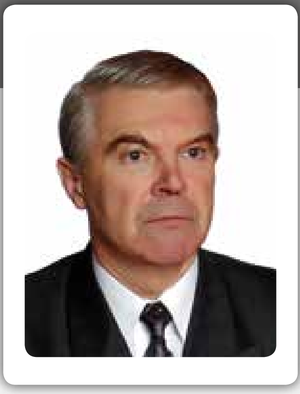
Mining has been extremely important to the World's economic growth and prosperity for centuries. The mining industry is currently facing an economic and social crises that can impact strongly the mineral production and productivity. In this scenario several challenges must be addressed, among them complex 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

A CASE STUDY INVOLVING LOADER TEETH, THERMAL VISION, BIG DATA, AND DEEP LEARNING

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A CASE STUDY INVOLVING LOADER TEETH, THERMAL VISION, BIG DATA, AND DEEP LEARNING

ABSTRACT

Missing loader teeth and adapters that go undetected can jam crushers or cause serious damage to downstream equipment. Retrieving a jammed tooth from a crusher is a very dangerous job. Furthermore, as loader teeth are usually narrower than shovel's, the undetected ones can pass through the crusher and cause serious damage to conveyor belts and screens. This poses serious financial and safety implications. A case study is presented here for an innovative solution for this nasty challenge. The solution combines rugged thermal vision with a lens cleaning system that offers a clear view of the hot teeth-line at all times. The acquired sequence of images are fed to an onboard artificial intelligence engine based on Deep Learning, which has been optimized using a large set of labeled bucket images (big data).

Using data gathered from mining operations around the world, a compelling case is outlined that determines the savings per missing tooth incident. The investigation analyzes data from both copper and gold mines taking into consideration the ore grade, metal prices, the mine's crushing capabilities, number of missing teeth, and average crusher downtime. It is found that an undetected broken tooth can cost the mine at least $350,000. Various image logs and missing tooth detection results from field trials are presented for the system.

KEYWORDS

Missing Tooth Detection, Deep Learning, LoaderMetrics, Big Data, Thermal Vision.

INTRODUCTION

Tooth breakage on wheel loaders presents ongoing challenges to both safety and productivity in surface mining operations. A broken tooth from a loader that goes undetected can end up in a crusher and cause a jam. Removing the tooth from the crusher requires manual labor which can be dangerous as the buildup of extremely high forces in the crusher may cause the tooth to fly out at very high speeds. On the other hand, as the bucket teeth on wheel loaders are generally smaller than those found on shovels, undetected broken teeth could also pass through crushers and cause serious damage to conveyor belts, screens, and other equipment in the downstream processing stages.

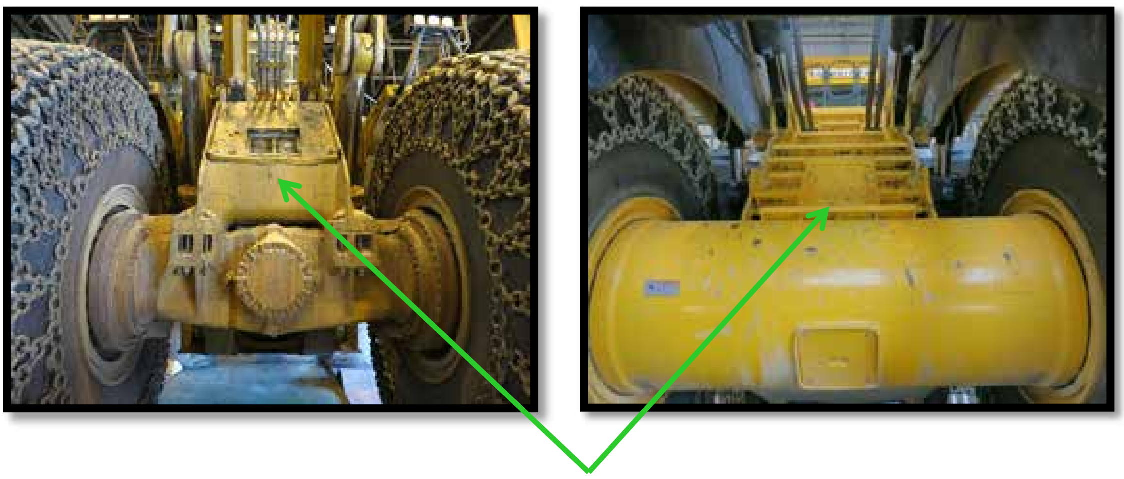
Tooth monitoring systems for mining shovels have been in the market for over ten years, but wheel loaders present unique challenges that require a different approach due to design and operational differences. On shovels, the vision system is mounted at a high vantage point on the boom or stick to capture images of the teeth. A loader does not offer such structures, so the vision system must be placed in a protected location that offers the best possible view of the teeth. In addition, a wheel loader's operating pattern differs significantly from a shovel's. These differences require different hardware and software platforms for missing tooth detection.

The proposed method for real-time missing tooth detection is to capture a video stream from a single IR camera. Each incoming frame is analyzed using artificial intelligence techniques in order to detect and extract the bucket teethline, and then examine the status of each individual tooth. The result will be shown in a graphical and visual format on a touch screen monitor. If any tooth is missing, a visual alarm will appear on the monitor and simultaneously, an audible alarm will sound to alert the machine operator.

Missing Tooth Detection

Missing tooth detection on wheel loaders presents unique challenges. The vision system must be placed such that the outline of the bucket teeth will be clearly visible in the image for tooth monitoring. As mentioned, the operational practices and physical structure of wheel loaders are different than shovels. The only position that a clear image of the bucket teethline can be viewed by a vision system is when it is placed between the machine arms and above its axle as shown in Figure 1 and Figure 2.

Figure 1- Vision System Mounting Location



Vision System Mounting Location

Figure 2- left: mounting location for Loader CAT994, right: mounting location for the Letourneau 1850/2350

From here, the teeth are visible when the bucket tilts downward, displaying the teeth as silhouettes against the background as shown in Figure 3. One of the challenges is that rocks occasionally fall on the camera in this location and could damage the camera. Additionally, water nozzles are required for cleaning debris from the camera lens to allow the tooth monitoring system to continue working under different working and weather conditions.

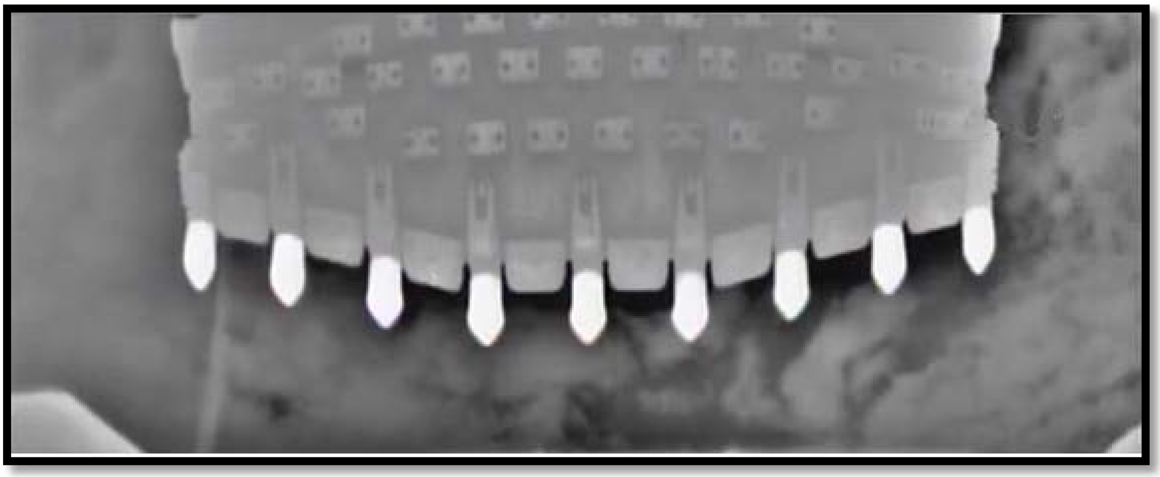


Figure 3 - Sample bucket camera view

In order to address the issues outlined above, a specially designed camera bracket with two different mounting points were created, one for LeTourneau 1850/2350 and one for CAT994, shown in Figure 4. The brackets are designed to secure infrared cameras and nozzles for lens cleaning, and also protect them from falling rocks.

Figure 4 - Camera bracket and mounting for LeTourneau 1850/2350

Blind Spot Reduction

In addition to the bucket camera, the system also includes a left, right, and rear-facing camera, which are connected to the same embedded computer located in the operator's cab. All of the camera views are displayed to the operator on a compact 7" touch screen, small enough to avoid disrupting the operator. The system interface is shown in Figure 5.

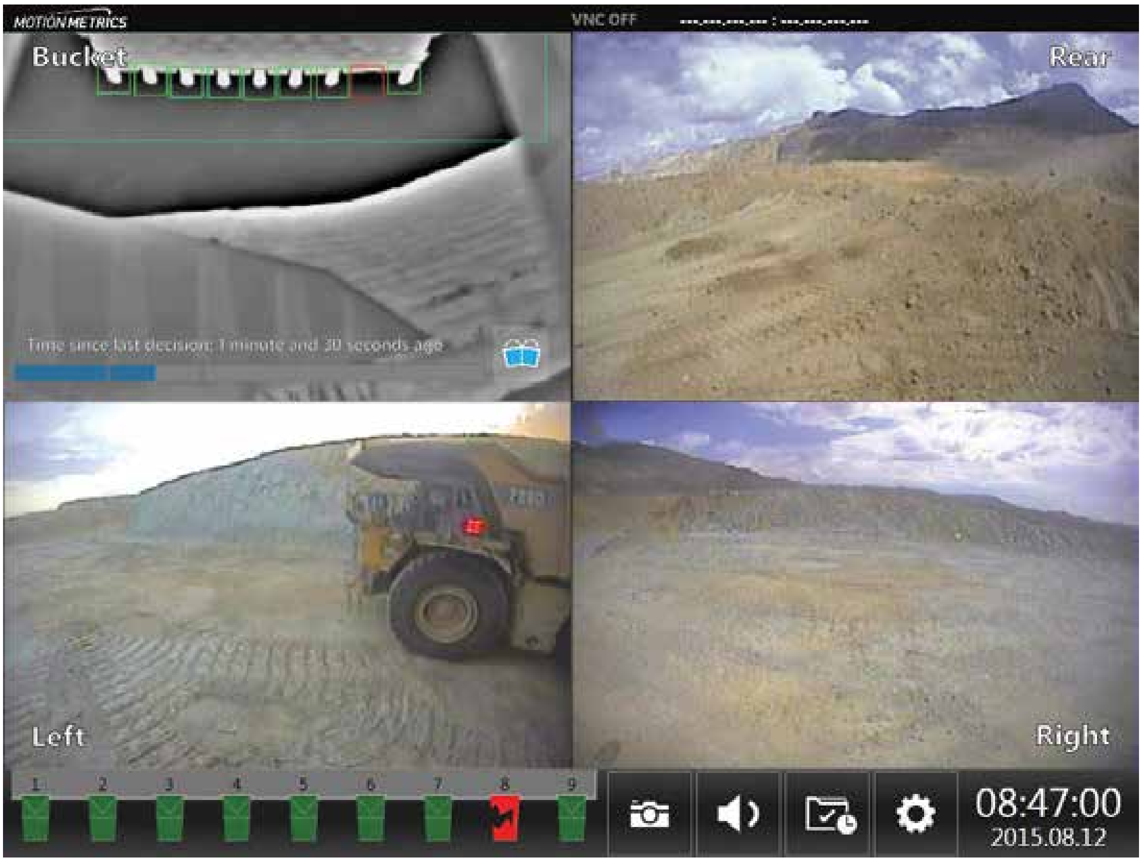


Figure 5 - Missing tooth detection interface

An embedded computer system continuously captures images from the video stream provided by the bucket camera, and the software analyzes the images by employing sophisticated image processing and artificial intelligence algorithms to determine if the bucket and teeth appear in the image. It will analyze each individual tooth and alert the operator if one is missing.

The optimal moment for capturing a clear image of the teeth is when the bucket is tilted down and completely empty as shown in the bucket view in Figure 5. As the bucket will only remain in this state for a short period of time, the video processing algorithms analyze the status of the individual teeth during this short window of opportunity.

When the missing tooth detection system successfully analyzes all the teeth, an image file is created and saved on the embedded system. Each log contains bucket camera views, as well as the status of each tooth and the time stamp. Figure 6 shows two consecutive log files produced by the missing tooth detection system. The top image shows that all teeth were present at 11:48:35, but one tooth is missing by 11:49:29. The system detected the missing tooth as outlined in the bottom image, sending an audible and visual alarm to the operator.

It is important to mention that the bucket teethline must be shown to the camera frequently, ideally after each dump; otherwise the system is unable to update the status of the teeth and cannot produce any alarms in the event of a missing tooth to alert the operator. The missing tooth system will notify the operator if the bucket teethline is not seen by the camera for a certain length of time.

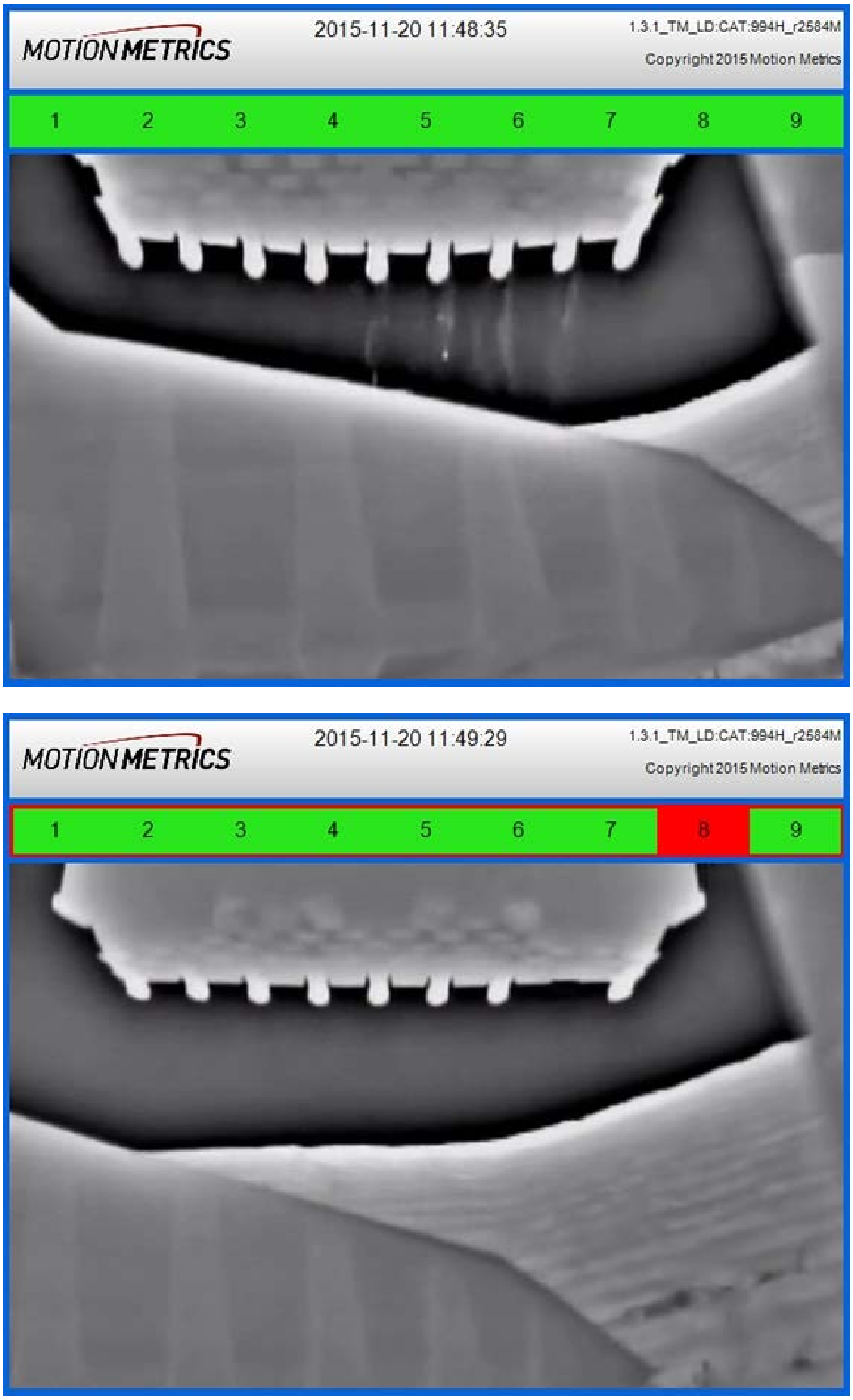


Figure 6 - Consecutive captured images in the event of missing tooth

Lens Cleaning System

An innovative, non-abrasive, non-contact Lens Cleaning System has been developed for keeping camera's clear of debris, reducing the likelihood of scratches and improving the longevity of the lens. The Lens Cleaning system is already integrated into the Motion Metrics embedded systems, see Figure 7.

Figure 7- Lens Cleaning System integrated into the GUI of LoaderMetrics™ software

The Lens Cleaning System uses anti-freeze wash fluid and pressurized air to remove dirt, mud, snow, road grime and mist. Spraying the wash fluid removes unwanted debris from the lens view, and is followed by pressurized air to dry the lens and prevent stains or streaks. The system can be operated in three ways:

- manually by pushing a dash-mounted button
- remotely through an Ethernet connection
- programmed automatically at specified intervals

In addition to a visual indicator on the device's exterior, the Lens Cleaning System is able to measure fluid level and send an alarm through the in-cab monitor when the reservoir is empty. Figure 8 shows the status of the Lens Cleaning System in the software.

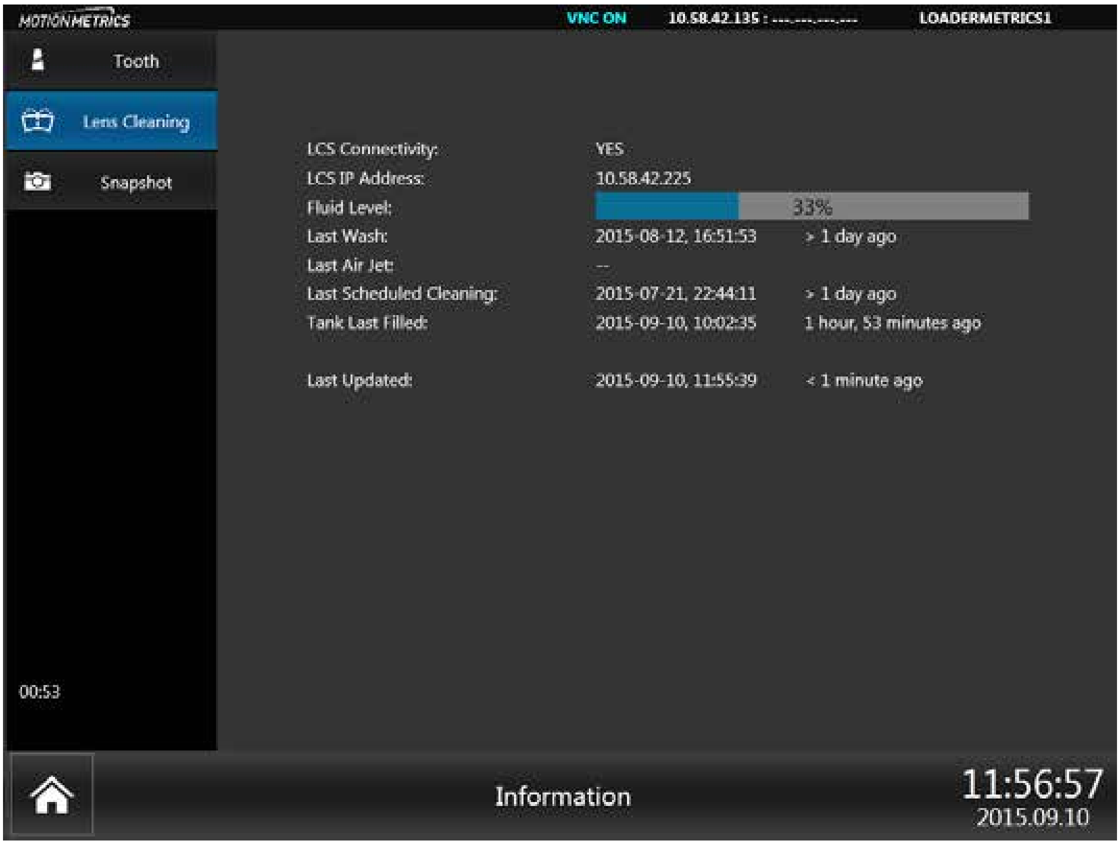


Figure 8 – The status of the Lens Cleaning System in the software

Centralized Data Server

A centralized data server solution has been designed to connect with all mining shovels and loaders that are equipped with the missing tooth detection system. The centralized data system utilizes a mine’s existing IT infrastructure to connect directly to any missing tooth detection systems, providing critical event notifications and information accessible throughout the mine. The simple web interface allows any devices on the mine network, whether it is a laptop or a tablet, to instantly access the health status for all loaders and shovels available on the mine’s network. Figure 9 shows the general structure diagram of the centralized data server system.

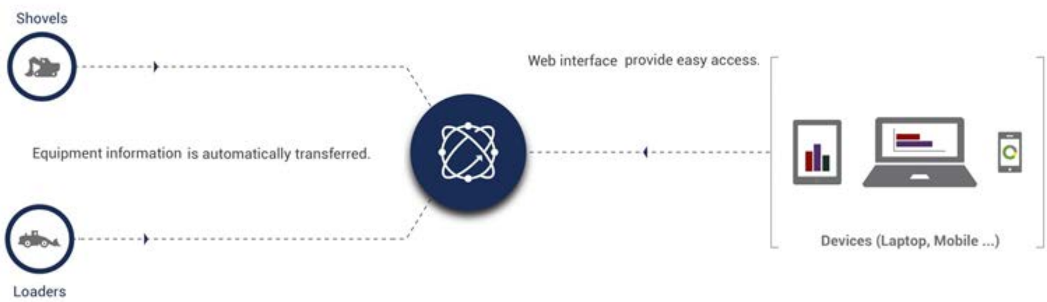


Figure 9 - The general structure of the centralized data server

In addition, all images from missing tooth detection systems can be easily downloaded, printed, or e-mailed. Figure 10 shows all log data for a desired period of time. The centralized data server generates a report for a defined period of time, which shows captured images of alarms and the progression of alarms on a timeline. Figure 11 shows an auto-generated report by the system. Also, customized alerts can be configured to notify the appropriate personnel for any critical events, as shown in Figure 12.

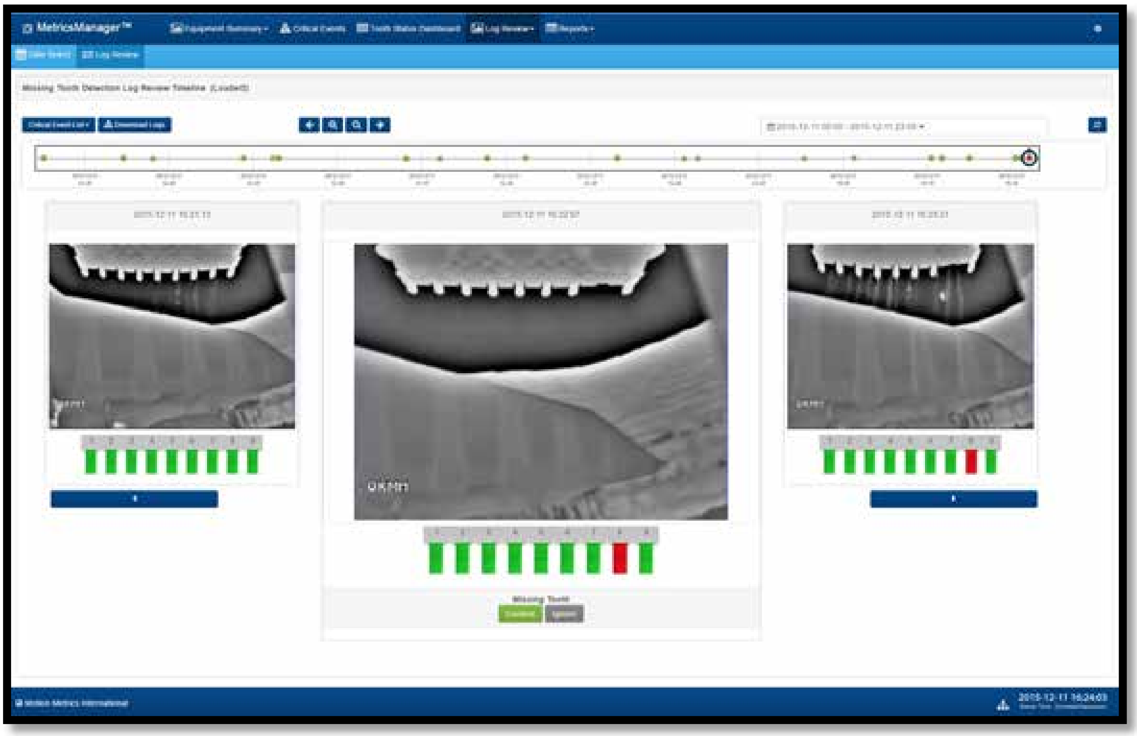


Figure 10 - Log viewer interface

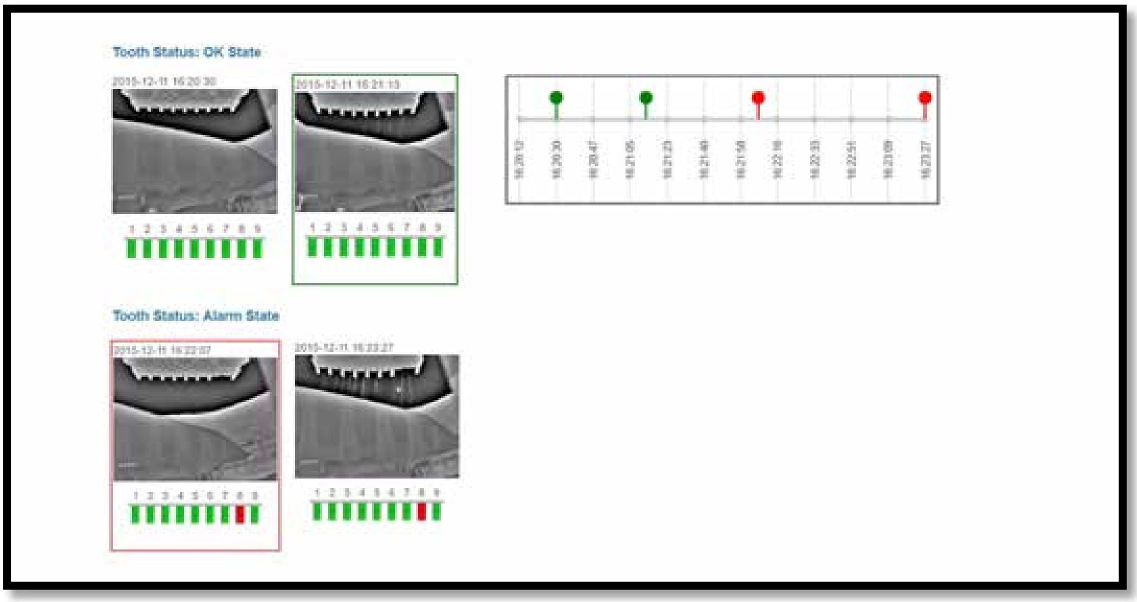


Figure 11 - Auto generated report

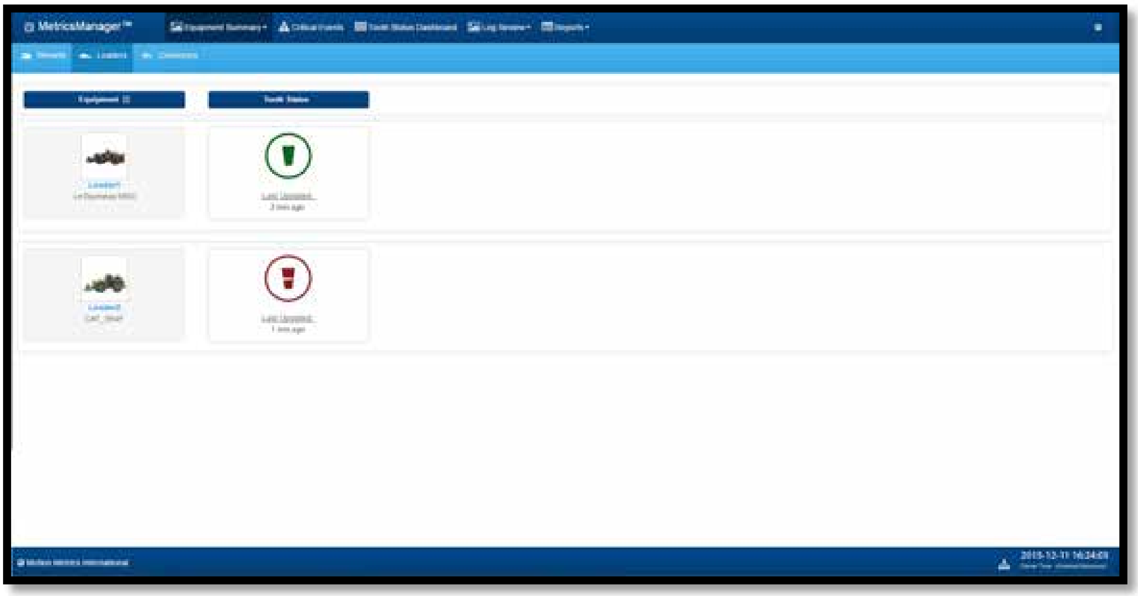


Figure 12 - Customized configuration for alerts

CASE STUDY

The value of a missing tooth detection system stems from its ability to minimize the likelihood of a missing tooth entering the crusher. In doing so, the danger and cost associated with removing a tooth from the crusher can be mitigated. Outlined below are case studies from a gold mine and a copper mine, outlining the costs associated with a missing tooth.

Case 1: A Canadian Gold Mine

Given an ore grade of 0.05oz/ton and gold prices averaging $1,290/troy oz, a gold mine with a single crusher capable of handling 40ktpd will process $2.59M USD worth of ore per day.

A missing or broken tooth that jams a crusher can result in 4 hours of downtime, which translates to productivity losses of 6.6kt, equating to $430k USD per incident (Hui, 2014).

Mine type	Gold Mine
Number of crushers	1
Crusher output	40ktpd
Ore Grade	0.05oz/ton
Price/ton	$1,290/troy oz
Avg. crusher downtime	4 hours

Case 2: A Swedish Copper Mine

Given an ore grade of 0.28% and copper prices averaging $2.75 USD/pound, a copper mine with a single crusher capable of handling 12ktpd, processes $200k USD of ore per day.

In this case, a missing tooth incident that jammed the crusher resulted in 36 hours of downtime, which translated to productivity loss of $350k USD (Hannington, 2008).

Mine type	Copper Mine
Number of crushers	1
Crusher output	12ktpd
Ore Grade	0.28%
Price/ton	$2.75/pound
Avg. crusher downtime	36 hours

CONCLUSION

The missing tooth detection solution is a vision-based system that utilizes machine vision and artificial intelligence methods to localize each tooth in each image frame streaming from a thermal camera and analyze their status. Due to the structure of wheel loaders and their operations, the bucket teethline is not easily visible to the bucket camera for most of the time. Thus, the bucket camera is installed between the machine arms and above its axle to be able to capture the teethline image when the bucket tilts down right before and after the dump of the load. The system is able to analyze the images captured and alert the operator when a tooth is missing in order to stop a missing tooth ending up in a crusher. Case studies presented outline the costs and dangers associated with missing teeth jamming the crusher. A single missing tooth in a Gold Mine resulted in losses of $430k USD, while a missing tooth in a Copper mine resulted in losses $350k USD. With an accurate missing tooth detection solution in place, safety moves to the forefront of mining operations while significant cost-savings are realized.

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A NEW DECISION SUPPORT TOOL FOR FAULT DIAGNOSIS, HAZARD PREDICTION AND ANALYSIS IN MINING INDUSTRY

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A NEW DECISION SUPPORT TOOL FOR FAULT DIAGNOSIS, HAZARD PREDICTION AND ANALYSIS IN MINING INDUSTRY

ABSTRACT

This paper presents DISESOR decision support system dedicated to the mining industry and its applications. The system integrates data from different monitoring and dispatching systems and contains such modules as data preparation and cleaning, analytical module, prediction module and expert system. Architecture of the system is presented in the paper. The work contains also two case studies presenting the examples of the system application.

KEYWORDS

Decision support systems, knowledge acquisition, machine learning, abyssal mining pomp stations, methane forecasting

INTRODUCTION

Systems for machinery monitoring and hazards prediction are used more and more frequently in the modern underground mining industry. These systems co-operate with sensors distributed all over the excavation and fixed on particular machines and devices. The described systems may belong to the SCADA class (*Supervisory Control And Data Acquisition*). These systems enable to collect streams of data, such as process parameters, control parameters, output signals, etc., as well as messages generated by control and security systems and by the maintenance ones. The data can carry information that can be used to support operators in their decision-making processes and, in case of automatic operation, to work out proper control. It is important to note that in today's industry this data is very rarely used to support decision-making. Therefore it was decided to extend significantly the functionality of SCADA systems by decision-support elements, with the use of a state-of-the-art approach based on knowledge acquired both from historical data and from experts on hazards, machines and devices maintenance applied in underground mining.

Within the DISESOR project a decision-support framework dedicated to the mining industry was developed (Kozielski et al., 2015).

The system, which is provided with the domain knowledge and based on the analysis of collected data, supports the users of supervision and monitoring systems (DS) in making decisions necessary to develop proper monitoring processes. The main purpose of the developed tool is to support and extend the functionality of DS systems employed in the mining industry, particularly hard coal mining.

PROJECT MOTIVATION

We can identify three basic advantages of using the developed system in terms of the main application area – the mining industry, particularly hard coal mining, and companies, which supply monitoring systems for this industry. Here two basic possibilities of the system application can be observed.

First, the system will enable an access to measurement data, integrated and standardized in terms of terminology and interpretation of the measured values. As far as monitoring systems in Polish hard coal mining are concerned, there is a huge diversity, both in the systems supplied by small and medium-size companies and the big ones.

The second possibility to use the developed solution is based on the assumption that data collected by monitoring systems are insufficiently exploited until now. So far, huge volumes of measurement data have been used mostly for on-line monitoring of processes. Nowadays, monitoring and supervision systems, apart from a few cases, are not equipped with any software tools that would enable to exploit domain knowledge and to use the results of historical data analyses to improve the work of operators and supervisors and to improve the progress of the monitored processes. Observing the costs of implementation and maintenance of today's monitoring systems, it is easy to defend the thesis that the cost of collecting data from the underground (sensors, transmission lines, measurement stations, etc.) is too high in relation to the benefits coming only from visualization and simple reporting.

Bearing in mind undesired situations that may happen during mining operations (e.g. serious damages of mining machines, uncontrolled and emergency stoppages of electric energy, fires, accidents, or catastrophes), it is possible to assume that at least a part of these situations could have been eliminated thanks to the analytical part of the proposed system. For example, due to short-term prediction of methane concentration and associating it with the information about the shearer location and its operating conditions, one can prevent emergency switch-offs of electric energy. This allows increasing the production output and reduces the wear of electric elements whose working time is proportional to the number of switch-on/offs. Thanks to the developed diagnostic module it is possible to accelerate the identification of machine-failure causes and to inform the operator in advance that the monitored values approach critical levels and what kind of consequences this situation may bring.

In the considered market segment it is possible to observe that suppliers of monitoring systems are becoming more and more conscious about the necessity to take the next step in these systems development. It is not an exaggeration to say that the issues related to data collection from the mines underground and their archiving have been solved to a large extent, while the suppliers of monitoring systems are seeking competitive advantage in providing knowledge engineering, modelling and data analysis methods to their systems (Hofman & Klinkenberg, 2013). Here the third application area for the project results can be defined. The suppliers of monitoring systems are more and more interested to supplement their systems with decision-support modules.

SELECTED ASPECTS OF DISESOR ARCHITECTURE

Figure 1 presents a diagram with the boundaries of the DISESOR system, sources and receivers of data/information, as well as main inputs and outputs (Przystałka, et al. 2016). As it can be observed, the developed system has a module-based structure including the following components: data repository, cleaning module, ETL (Extract, Transform, Load) module, analytical module, etc. The system co-operates with the environment whose main actors are: *system administrator* responsible for the security of knowledge, information as well as data processed and stored by the system; *knowledge engineer* who supplements knowledge bases by making analytical and prognostic models, developing complex reasoning schemes, etc.; *dispatcher* that means the end-user of the system; *SCADA/DS systems* which are to be supplemented with the DISESOR system functionality and are to be the data source for this system; *guest* who is a special user, e.g. a user whose access to the system is only for demonstration or training purposes; *external systems* which can be a source of extra information/data (e.g. context data) about the operations of devices and machines as well as technological processes and natural hazards. In the latter case an example can be *newsletters* about weather as these data can be taken into account during the reasoning process on potential hazards.

Analysing Figure 1 it can be observed that the central point for integrating and synchronizing the system operations is a data repository. This element is equipped with a module with interfaces indispensable to validate, clean and process input data and messages delivered by other system components. A structure of the repository was designed to enable a wide spectrum of measurement data to be supplied. By defining the layer of metadata, which describe the structure of the monitored object, it is possible to locate devices and sensors (independently of descriptions in particular measurement systems) and, this way, to associate the quantities measured by these devices. Additionally, the repository is a source of data/information for other components of the system.

Figure 1 – Context diagram of DISESOR system

The analytical module performs analyses of historical data (off line) and reports about identified important dependencies and trends. The results generated by this module supply the data repository no sooner than the user accepts them. Thus the module supports the user in deciding what should be monitored and predicted and provides extra information, which can be used to enrich the knowledge base of the expert system or to conduct comparative analyses. Thanks to the analytical module it is possible to learn whether measurement values (original or aggregated) related to the given aspect of the monitoring process follow a right direction or not, whether they are related to other values and whether, in the light of the collected data, it is possible to identify the cause of the registered changes. The module has the basic tasks defined (e.g. "follow trend", "compare with respect to", "what if"). The user can initiate analytical tasks by selection and parameterization of predefined operations (selection of comparable variables, aggregation method of available measurement data, determining attraction thresholds of revealed dependencies, comparison frequency, etc.). The basic representations of the revealed dependencies are rules, facts and simple statistical indicators, which illustrate the kind and strength of the identified dependencies. The results of the initiated analytical tasks are presented in a form as close to a natural language as possible.

Another important element of the system is a prognostic module. Prognostic models are subject to adaptive control based on the input data stream together with the comparative analysis of patterns learned on historical data before. The module provides access to interfaces for the selection of a group of dependent variables (which are subject to prediction) and independent ones (which support the prediction), for the determination of the preferred prediction horizon, and for the selection of indictors with their threshold values defining the minimal required quality of prognoses. The prognostic module makes use of data models developed in the analytical module. If the achieved prognoses have acceptable quality for the user, they are treated as virtual measurement data, which can be transferred to supervision systems and visualized there.

The expert system shell of the framework is responsible for fault diagnosis of devices and machines in an on-line or off-line mode, supervision, proper progress of processes, and supporting the decisions of a supervisor/expert (Kalisch, et al., 2014; Kalisch, et al., 2015). Decision support concerns both the technical state of the objects and critical situations resulting from the process going to an undesirable direction. The designed and developed expert framework enables to obtain and save knowledge in one of three representations (decision tree, fuzzy rules, Bayesian network) and to carry out the reasoning process. The process is conducted with the use of elementary methods adequate for the selected knowledge representation: standard reasoning based on sharp rules and facts, probabilistic reasoning based on belief networks, or possibilistic reasoning based on fuzzy rules. An extra advantage of the developed tool is the possibility to have a kind of reasoning that is aimed at explaining the causes of the situation and suggesting possible operational procedures.

The final element of the system is the visualization and reporting module whose task is to present the collected measurement data and other information to the user in a readable manner. The user who defines boards and maps does the visualization. Different types of reports can be defined too.

SYSTEM IMPLEMENTATION

The system was implemented with the use of RapidMiner, which is an open-source analytical environment (RapidMiner, 2015). The selection of the environment affected the program used in the project and the system architecture. The RapidMiner environment ensures many predefined functions for data processing and analysis. This allowed saving a lot of time for basic implementation of the system functionality. In addition, RapidMiner ensures a basic graphic user interface. On the other hand, RapidMiner is fully configurable and it is possible to change its functionality freely (Java). In the Gartner Magic Quadrant for Advanced Analytics Platforms (2016) RapidMiner was recognized, together with SAS, IBM and KNIME, as a leader of advanced analytical platforms. RapidMiner makes it possible to use open solutions contained in a more and more popular R statistical calculations package for commercial applications.

RapidMiner is devoted, to a large extent, to carry out tasks related to data exploration. This environment was equipped with a clear graphic interface, which allows programming with the use of the drag-and-drop method. More advanced users of the system can also use programming libraries in a standard manner. Available plug-ins, extensive documentation and an open source code allow expanding the functionality of the application significantly. The implementation of DISESOR key modules was made in the form of new RapidMiner application plug-ins. This way we achieved software in the form of a decision-support framework equipped with ready-to-use tools that enable to acquire knowledge indispensable for the system to work properly.

EXAMPLE OF THE SYSTEM APPLICATION IN ABYSSAL MINING PUMP STATIONS DIAGNOSTICS

Abyssal mining pump stations represent a fundamental solution to the problem of a coal mine dewatering. Due to large responsibility in maintaining water at a certain level that guarantees safe operation of the mine, the systems that oversee the abyssal mining pump stations are safety systems. The pump monitoring systems are installed in several dewatering stations and during normal operation they register the following pump unit parameters:

- pump unit temperature,
- the power consumed by the motor,
- the current drawn by the motor,
- the productivity of a pump unit.

The values of the parameters listed above are acquired each second. Due to the safety constraints the temperature of the pump motor should not exceed 75 °C and a pump can be started when its temperature decreases below 25 °C.

The analysis of the collected measurements enables evaluation of pump diagnostic states. The following feature vector was used during the analysis of pump diagnostic states:

$$P_i = [T_{U,i}\,,\ T_{0,i}\ t_{20\text{-}30,i},\ t_{30\text{-}40,i},\ t_{40\text{-}50,i},\ t_{50\text{-}60,i},\ t_{60\text{-}70,i},\ P_{U,i}\ Q_{U,i},\ L_i,\ D_{pi},\ D_{ki}],$$

where:

- T_U – temperature of a pump unit in a steady state,
- T_0 – initial temperature of a pump unit,
- t_{x-y} – time period when the pump temperature changes by 10 degrees (if the pump temperature did not reach a given range, a 0 value was inserted),
- P_U – power of a pump unit in a steady state,
- Q_U – performance of a pump unit in a steady state,
- L – number of starts on the previous day,
- D_p – time and date when the pump was turned on,
- D_k – time and date when the pump was turned off.

Temperature of a pump unit in a steady state (T_U) was calculated as an average value of the last two minutes of operation. Each record P_U reflects a single pumping cycle (from the unit turn on to turn off).

The analysis of the historical data and the interview with the dispatchers of the station (experts) enabled to define three diagnostic states: *a new pump unit* (also after repair), *correct operation* and *suitable for repair*.

The time periods t_{x-y} have the main impact on the diagnostic state of a pump unit. Along the pump unit operation, when it becomes exploited, the time periods t_{x-y} become shorter and the critical temperature when the pump must be turned off is reached faster. This results in pump numerous turns on during the days preceding decision of its repair. Therefore, the number of times the pump was turned on is an important diagnostic indication. It has to be regarded, however, in conjunction with the information about the temperature of a pump unit in a steady state in order to omit other than high temperature turn off reasons.

Pump state diagnostics was based on a Mamdani-type fuzzy system (Mamdami & Assilian, 1975) with the following rules (the notation (p1, p2, p3, p4) reflects trapezoidal membership function (Bezdek, 1981)):

IF	T20_30 ∈ (199, 255, 255, 409)	THEN a new pump unit
IF	T20_30 ∈ (0, 197, 255, 409) and	
	T30_40 ∈ (245, 246, 256, 362) and	
	T60_70 ∈ (826, 1159, 1473, 679715)	THEN a new pump unit
IF	T20_30 = 0 and L∈(1, 1, 1, 2)	THEN correct operation
IF	T20_30 = 0 and	
	T40_50 ∈ (0, 0, 387, 727) and	
	L ∈ (1, 1, 3, 3)	THEN correct operation
IF	T20_30 ∈ (0, 255, 255, 409) and	
	T_u ∈ (73.1, 73.41, 74.54, 81.9)	THEN correct operation
IF	T20_30 = 0 and	
	T50_60 ∈ (0, 390, 390, 551) and	
	L ∈ (3, 5, 5, 7)	THEN suitable for repair
IF	T_0 ∈ (15.14, 19.75, 19.75, 27.26) and	
	T20_30 = 0 and	
	T30_40 ∈ (206, 366, 366, 11417) and	
	L ∈ (3, 5, 7, 7)	THEN suitable for repair

Figure 2 presents the division of attribute L (number of starts on the previous day) into fuzzy sets.

In practice, the state *suitable for repair* does not lead to immediate breakdown of a pump unit. Dispatchers suggested that a pump classified to this state is able to (or sometimes has to – waiting for service) operate up to next 3 months (when a typical operation time lasts 2 years). In order to improve the accuracy of service prediction (pump breakdown) a decision tree was created by means of a decision tree induction algorithm (Sikora, 2007). Tree induction was performed only on the examples labeled as *suitable for repair*. The resulting tree classifies each vector P_i to one of two decision classes: *less than a month to breakdown* and *more than a month to breakdown*. The induced tree utilises only the time periods $t_{x\text{-}y}$. An applied train-and-test method showed the classification accuracy on a level of 90% (the class distribution was balanced, therefore, this measure is appropriate to classification quality evaluation). The induced tree (the main node) was slightly modified, what increased accuracy by 2%, and it was applied to the expert system. The decision tree is presented in Figure 3.

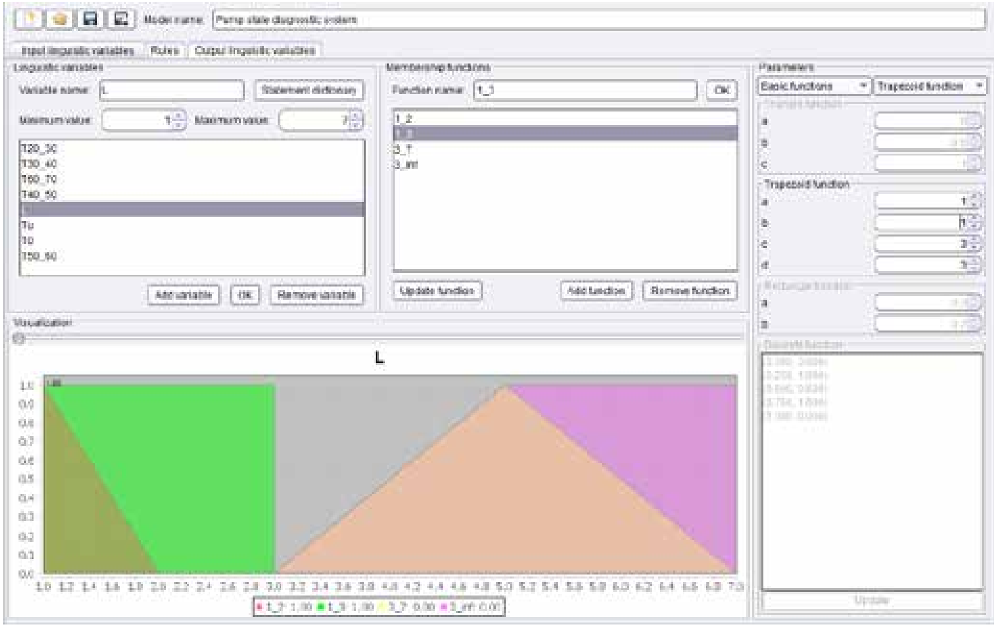


Figure 2 – Division of *L* variable into fuzzy sets

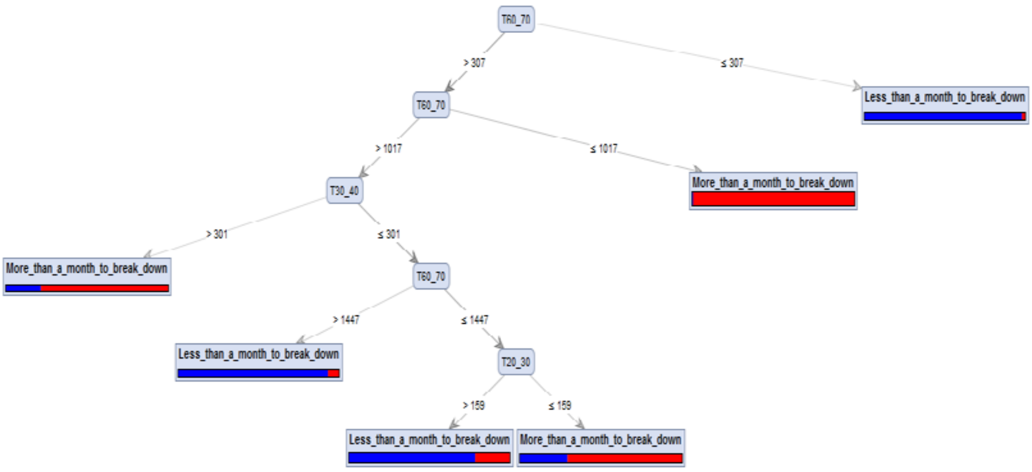


Figure 3 – The decision tree applied to the expert system

The expert system works according to the following steps: after each pumping operation the diagnostic state of a pump unit is evaluated. If a pump is classified as *suitable for repair*, then the decision tree is applied. It identifies the expected time period of the pump operation. If the pump is classified as *less than a month to break down* then the pump should be turned off because the costs of the repair of the broken unit are very high.

EXAMPLE OF THE SYSTEM APPLICATION IN METHANE CONCENTRATION FORECASTING

The DISESOR system can be applied to several different tasks solving. This section presents an example, how the system can be utilised to methane concentration prediction.

Methane concentration monitoring is one of the main tasks of the natural hazard monitoring systems in mining industry (Sikora, et al., 2011). Such a system is in charge of automatic and immediate shutdown of electricity within a given area, if a methane concentration exceeds a given alarm threshold. The power turn-on is possible after a certain time (from 15 minutes to even several hours), when the methane concentration decreases to the acceptable level. This results in large losses associated with downtime of production. Information from a soft (virtual) sensor presenting to a dispatcher the prediction of the methane concentration with a few minutes horizon can allow the prevention of electricity shut-down or can allow lowering the mining activity and increasing the air flow if possible. Therefore, these actions allow avoiding undesirable situations and unnecessary downtimes.

The task of maximal methane concentration prediction with the horizon from 3 to 6 minutes was realised within the DISESOR system. By means of ETL2 module a set of the following sensors was selected: AN321, AN541, AN547, AN682, BA1000, BA603, BA613, BA623, MM11, MM21, MM25, MM31, MM36, MM38, MM39, MM41, MM45, MM52, MM53, MM54, MM55, MM57, MM58, MM59, MM61, MM81.

The data was aggregated applying minimum operation to anemometer (AN) measurements, average operation to barometer (BA) measurements and maximum operation to methanometer (MM) measurements. The missing values were inputted applying the linear regression method. As a dependent variable MM59 sensor was chosen. A map presenting the topology of the mining area and location of the sensors is presented in Figure 5.

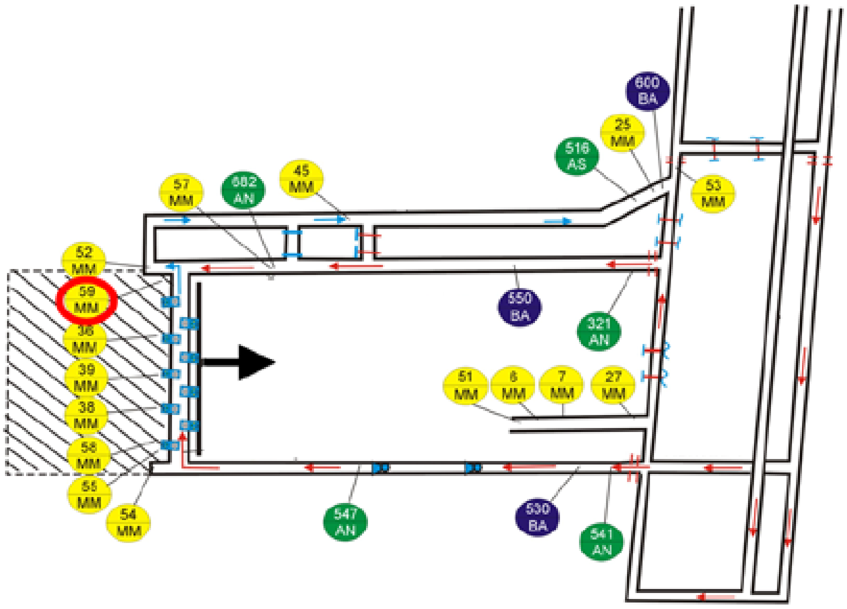
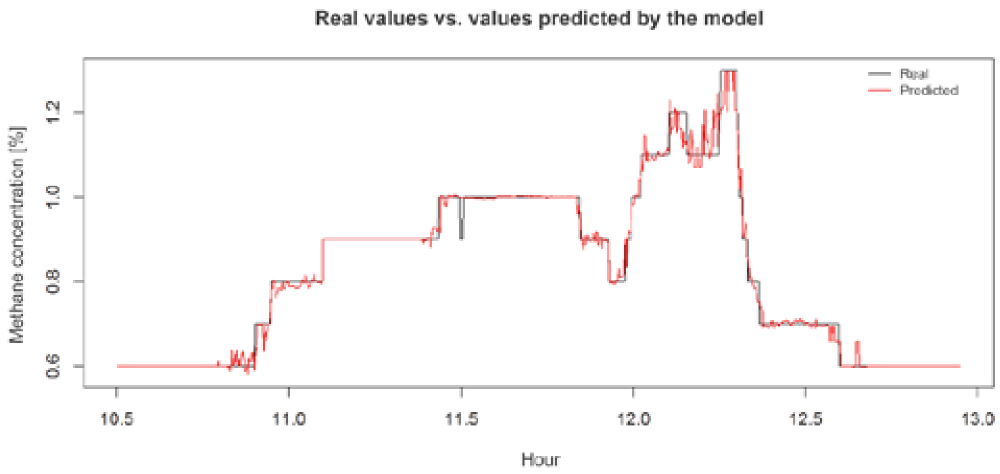


Figure 4 – Topology of the mining area and location of the sensors - MM59 sensor chosen as dependent variable is outlined in red

The initial regression tree was created on the basis of data coming from 1 shift. The model and the list of sensors (variables) together with the defined transformations were forwarded to the prediction model running a proper service. The time-window defined for prediction quality monitoring was set to 1 hour and the model adaptation was executed each hour regardless the minimum quality requirements. The adaptation could be executed more often if the minimum quality requirements were not met but there was no such situation.

Figure 5 presents the plot of the real methane concentration and the predicted maximum concentration together with the histogram of errors that are reported to a user. Currently, the user interface is in Polish as the deployment in Poland was planned in the project. However, the English and Chinese versions are also planned.



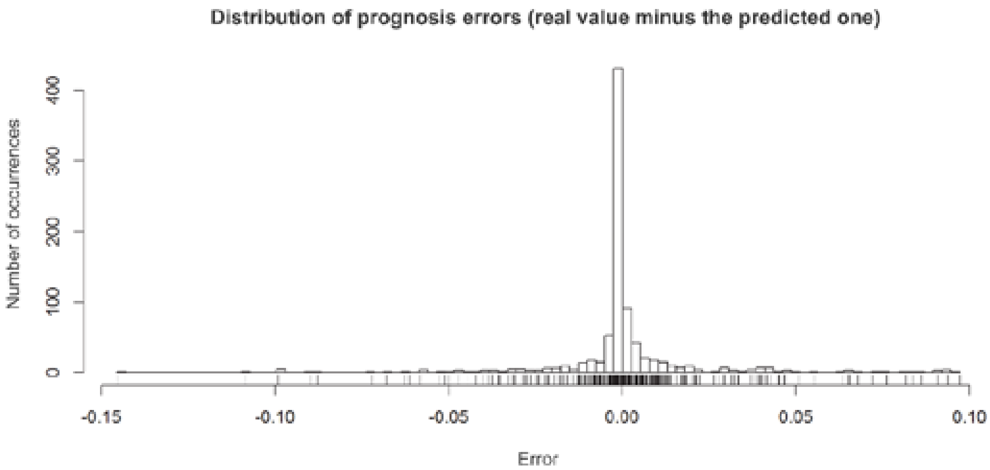


Figure 5 – The plot of the real methane concentration and the predicted maximum concentration together with the histogram of errors that are reported to the user

CONCLUSIONS

The paper presents the architecture of the DISESOR system, which has been designed to provide decision support, particularly in the hard coal mining industry. The motivation for the system development was described, along with particular modules and technologies applied in the system. Additionally, two use cases were presented.

The system is a unique and complex tool, which can be integrated with other monitoring and supervision systems functioning in hard coal mining. The system is a framework solution with knowledge-engineering and data-analysis tools, which enable to use it for decision support in a large spectrum of processes and hazards.

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A NEW DECISION SUPPORT TOOL FOR FAULT DIAGNOSIS, HAZARD PREDICTION AND ANALYSIS IN MINING INDUSTRY – A CASE STUDY

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A NEW DECISION SUPPORT TOOL FOR FAULT DIAGNOSIS, HAZARD PREDICTION AND ANALYSIS IN MINING INDUSTRY – A CASE STUDY

ABSTRACT

The paper focuses on the application of the developed decision support system DISESOR in machinery fault diagnosis. The main objective of the paper is to show how the expert system (as part of the proposed tool) can be applied for reasoning with the use of multi-domain knowledge representations and multi-inference engines. The preliminary verification of the developed expert system shell was conducted with consideration to the problem of knowledge acquisition from domain experts in the field of belt conveyor systems. The proposed tool is also tested using the data generated by means of the belt conveyor simulator in which fault-free and fault scenarios are modelled. In this example, four different states of the machine are taken into account: f_0 - faultless conditions; f_1 - the belt is ragged; f_2 - the rolling friction is increased; f_3 – the engines are overheated. In this part of the paper, classic and context-based fault detection and isolation schemes are proposed for fault diagnosis of a belt conveyor. The obtained results were elaborated in order to illustrate the merits and limitations of the developed tool.

KEYWORDS

mining industry, belt conveyor, fault detection and isolation, diagnostic expert systems, knowledge acquisition, machine learning

INTRODUCTION

Modern mining processes become ever more and more mechanized, automated and robotized. In todays coal mines modern longwall systems are introduced, which can operate in automatic manner without the need of controlling by the miners anymore. In the same time, massive amount of data is acquired that carries information about the process, condition of machinery, and the dynamic state of underground environment. This data is collected simultaneously with working parameters of machinery that carry information of the influences of the external environment on the machine. All this data can be collected by modern SCADA systems. Since two or three decades there is growing belief that these sources of data shall be exploited off-line to acquire knowledge that might help in safe and efficient operation of production systems, and could be source of valuable knowledge of how to improve safety and make easier the work of human operators who personally control the systems operating underground.

The paper deals with an application of advanced methods of knowledge engineering to the acquisition of knowledge concerning monitoring and diagnosis of a typical coal-mine equipment, which is a belt conveyor. Knowledge engineering process was focused on two different knowledge sources, namely domain experts, and databases. All the work has been carried out in the programming environment DISESOR (Sikora et al., 2016). DISESOR is a kind of modern systems that integrates data from different monitoring and dispatching systems and contains such modules as data preparation and cleaning, analytical module, prediction module and an expert system DISESOR.DIAG, which has been extensively used thorough the research reporten in the paper.

Belt conveyors are one of the most frequently used machines for material transportation in different branches of industry such as an open cast mining (Bartelmus, 2006) and an underground mining i.e. copper and coal mining (Jonak & Gajewski, 2006). Typical structure of the conveyor belt system consist of (Zimroz & Król, 2009): drive unit (electric motor, damping coupling, multistage gearbox, coupling connecting output shaft and drive pulley), pulleys (drive pulley, head pulley, tail pulley), textile or steel cords belts, idlers for belt supporting, and other equipment such as belt cleaning system, control unit, etc. Each component of the Belt Conveyor System has specific types of failure and forms of wear which depend on construction of those components. In case of multistage gearbox, failures are related to shaft cracking, gears wear and damage (e.g. pitting, broken tooth, backlash, etc.) and various defects of rolling bearings (e.g. defects of inner/outer race and rolling elements) (Zimroz & Król, 2009). In the Belt Conveyor System, a belt has the lowest durability than other components and at the same time it is the most expensive part of the whole system (Jonak & Gajewski, 2006). For this reason, a special attention must be paid to diagnosis of this component. Typical damages of belts of conveyors operating in underground mines are (Jonak & Gajewski, 2006): belt breakage (mostly in a mechanical or glued connections), belt delamination, longitudinal or transverse cuts and others.

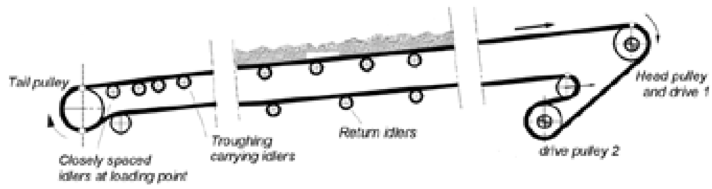


Figure 1 – the scheme of a belt conveyor (Zimroz & Król, 2009)

Diagnostic opportunities of a belt conveyor are determined by construction of their components and operating conditions such as dust. In particular, the technical state of the multistage gearbox, couplings, pulley and idlers can be evaluated by means of analysis of vibro-acoustic signals (Bartelmus, 2006) and analysis of thermography images (Błażej & Zimroz, 2013). In case of conveyor belts, diagnostic methods mentioned above are not sufficient enough, therefore methods based on visual inspection and nondestructive testing (NDT) are used (Błażej, 2012). One of NDT methods is based on magnetic testing (Błażej et al., 2015) which is suitable to diagnosing belts with steel cord core. The main idea of the method consists in the magnetization of steel cords in belt core using permanent magnets and then observation of magnetic field generated by magnetized cords by means of a system of coils. When a damage of a steel cord occurs then induced magnetic field decreases and electric current flows in the system of coils in accordance with electromagnetic induction law. In the case of textile belts, special detection sensors called antennas are built into the belt (Jonak & Gajewski, 2006). Antennas are used to transmit signal from the transmitter to the receiver. The transmitter is located near the driving pulley while the receiver is located near the tail pulley. When any fault occurs (antenna breakage) then the signal is not transmitted and the belt conveyor system is stopped for service purposes.

Not all methods presented above can be used in diagnostics of belt conveyor systems. It mainly depends on operating conditions, which are determined by exploitation area of a belt conveyor. In the area of the coal mining, belt conveyors operate in a dust and methane atmosphere. It results in high design requirements, not only for structure components of a belt conveyor but also for a measurement system. For this reason, diagnostic methods based on available process variables (e.g. the current of electric motor, the linear velocity of the conveyor belt, etc.) are developed. In particular, these methods are based on different algorithms and methods of data analysis such as harmonic analysis (Michalak & Sikora & Sobczyk, 2013) and others. They allow detecting faults but in a general case the additional expert knowledge is required (Michalak et al., 2013). For a more precise diagnosis including faults isolation, model-based methods are developed as well (Ghoshal & Samanta, 2012; Cenacewicz & Przystałka, 2015). A main idea of these methods is a change of parameters of a mathematical model of the belt conveyor due to observed process variables (Ghoshal & Samanta, 2012). Then the fault is identified by means of changed model parameters.

DISESOR.DIAG

DISESOR.DIAG is an expert system shell module of the DISESOR system (Michalak et al., 2013; Przystałka et al., 2016) that can be used for on-line and off-line diagnosing of technical objects and for monitoring processes. Another purpose of the module is to support domain experts in taking decision either on terms of the technical condition of the objects or on risk managing e.g. in situations when the process is going to an undesirable direction. The main assumption in case of operation of this module is to allow the user to acquire knowledge intuitively and to store knowledge applying different types of representations. The expert system shell is designed in such a way that the reasoning process can be based on elementary methods such as Boolean or fuzzy logic, Bayesian networks, etc.

The system consists of two layers. The first layer is called a management layer and it is used to supervise the whole system. There are included several functionalities such as mode selection that can be used to switch the system into one of three different modes. The on-line mode serves in situations when the scheduler of the reasoning process is created. In this mode a user (a knowledge engineer) is also able to observe the parallel execution of the scheduler logic. The off-line mode is often applied for ad-hoc reasoning on historical data. The last mode is necessary for editing the knowledge base. The engine of the system is implemented in the second layer (the execution layer). The user is able to clearly edit the knowledge base using three different knowledge representations: Boolean logic, fuzzy logic and Bayesian networks. The user cans also define a vocabulary of statements in order to prepare the description of a monitored object. The execution layer applies multi-domain knowledge representations for reasoning by means of the multi-interference engine. Inference procedures are executed using selected reasoning engines in order to prepare the conclusion. It is also assumed that a context may be taken into account in the condition part of the rules or input nodes of the Bayes network (it is an extended use case). In the next step the final conclusion is achieved in the result of the fusion process. The output of the system (statements) is obtained for measured and user data. The last use case of the system is the possibility to record the results of reasoning in order to realize the explanation interface.

The authors of the DISESOR project decided to apply RapidMiner software as the engine of their decision support system. RapidMiner is developed to solve data mining problems (Akthar & Hahne, 2004). It has a user-friendly interface with a drag-and-drop technique. The environment allows the users to make modifications in a source code and add new plugins created by other programmers in order to extend the functionality of the program. It was decided to use this software because libraries and plugins for ensemble learning can easily be applied for knowledge acquisition. The additional feature is that the functionality discussed above may be obtained in the direct way by creating new plugins for this software. The more detailed description of this module and the whole DISESOR system is omitted here, however it can be found in our previous publications (Przystałka et al., 2016; Wachla et al., 2016; Kozielski et al., 2015).

BELT CONVEYOR SIMULATOR

The subject of the case study is a belt conveyor simulator that has been elaborated in the framework of the DISESOR project in order to obtain benchmark data, which was needed for proper verifying and confirming the functionality of the DISESOR.DIAG module. This simulator can be also viewed as a tool created for scientists and engineers to simplify the process of evaluating and comparing different fault detection and isolation methods for devices and systems common in the mining industry. The model was developed and implemented in MATLAB®/Simulink® software taking into consideration the physical phenomena describing the origins of measurement noise, unmeasured disturbances and faults in such kind of systems. The model assumes the ability to simulate different operational scenarios of a conveyor as well as operational faults typical for belt conveyors working underground. The modular structure of the belt conveyor simulator is presented in Figure 2. As one can see, the simulator consists of numerous subsystems corresponding to the mathematical model. In the present study, the simulator is based on the construction and parameters of the belt conveyor of type GWAREK-1000 (the length of the conveyor belt equals 500m, the inclination angle of the conveyor equals 2°). The more detailed description of the analytical and numerical aspects of the model can be found in (Cenacewicz & Przystałka, 2015).

In this study it is assumed that only elementary states are investigated. Due to this, four different states of the plant are taken into account: f_0 - faultless conditions; f_1 - the belt is ragged; f_2 - the rolling friction is increased as a result of the corrosion process of rollers; f_3 – the motors are overheated as a result of backfilling them by output material. The set of process variables obtained as a result of the simulation is as follows: *V7* [m/s] is a linear velocity of the belt, *INZ1A*, *INZ1B*, *INZ1C, INZ2A, INZ2B, INZ2C* [A] denote currents of phase lines A, B and C for the first and second motor, *N1* and *N2* [rpm] represent rotation speeds of the motors, *TP* serves as a context-sensitive feature.

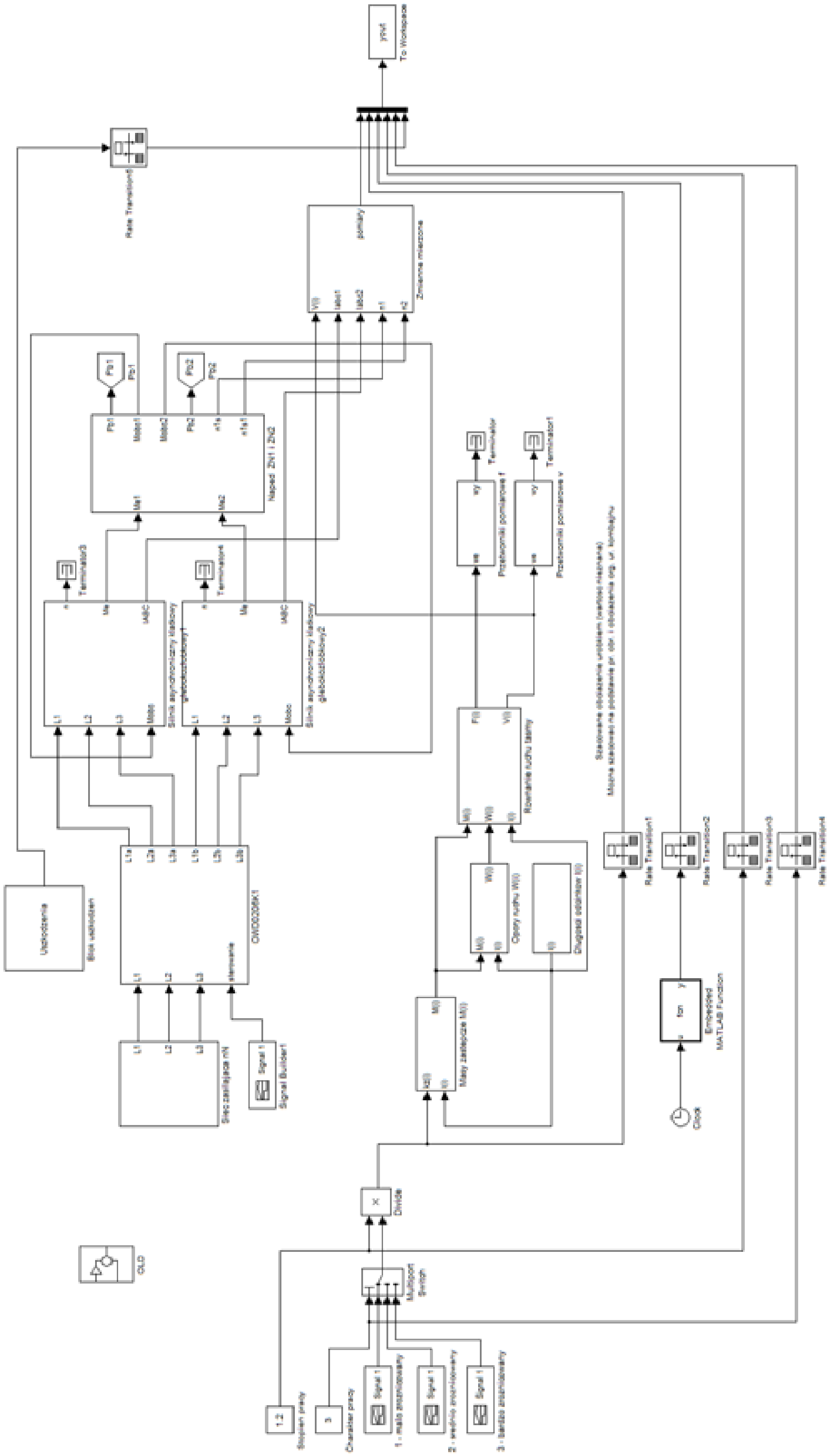


Figure 2 – Modular structure of the belt conveyor simulator

These process variables are available for fault diagnosis purposes. In order to simulate adequate operating conditions of the belt conveyor system it was decided that the numerical experiment will be related to the following operating modes of the plant: *TP* = 1 – starting mode (without the excavated material); *TP* = 2 – normal mode during the mining process; *TP* = 3 – stopping mode (without the excavated material). Moreover, the diverse types of operating conditions of a shearer (e.g. small, medium and large range of drum's loading characteristics) as well as various levels of load of the conveyor (e.g. 50%, 100% and 120% of the maximum nominal capacity) are considered during simulation experiments in this study.

KNOWLEDGE ACQUISITION FOR DIAGNOSTIC EXPERT SYSTEM

Knowledge acquisition from domain experts

In case of monitoring the work of mining machinery, expertise concerning the most frequent events / failures during the operation of these machines is fragmentary, and therefore there is a need to engage experts into the process of elaboration of knowledge base for the system. In the presented research experts such as miners, conveyor belts operators and servicemen were asked to participate in the investigations.

It is possible to acquire procedural and declarative knowledge. Knowledge of facts, objects, classes of objects, attributes of these objects and about relations between objects is declarative one (Moczulski, 2002). In the presented research this kind of knowledge has been acquired on the basis of results of the survey carried out. Generally the process of acquiring knowledge is understood as a series of complex actions of a sequential – iterative manner, which are time-consuming and often characterized by the interaction between sources of knowledge, the knowledge engineer and the final users of the knowledge base (recipients of the process of acquiring knowledge) (Moczulski, 2002).

Conducting a survey is one of the methods of acquiring declarative knowledge from experts (Moczulski, 2002). In order to gain knowledge from experts there have been developed forms and questionnaires containing questions related to the most common and most important events understood as malfunctions or failures (observed during the operation of the machines). It should be stressed out that acquisition of knowledge from the experts is considered a "bottleneck" in the development of expert systems (Jagielski, 2005). The techniques aiming in knowledge acquisition do not usually allow one to gain complete knowledge in a given area, but only a subset of knowledge, which may affect the operation of the expert system (Jagielski, 2005). In the field of mining, knowledge acquisition and its formalization is a difficult process, due to two reasons: the reluctance of experts to share their knowledge and the nature of machines operating in harsh conditions - sometimes the most important thing is that the machine works despite minor disability, less serious events are usually not signaled by the staff operating the machine, and because of that withholding or termination of machine operation caused by serious failures is being observed.

In order to acquire knowledge to the DISESOR.DIAG system the questionnaires were developed in such a way that it was possible for the experts to fill them in paper or electronic form. Development of surveys designed to acquire knowledge from domain experts required prior reading of subjects related to: type and model of the conveyor belt; construction of the conveyor belt; operating rules of the conveyor belt; events / failures occurring during operation of the conveyor belt.

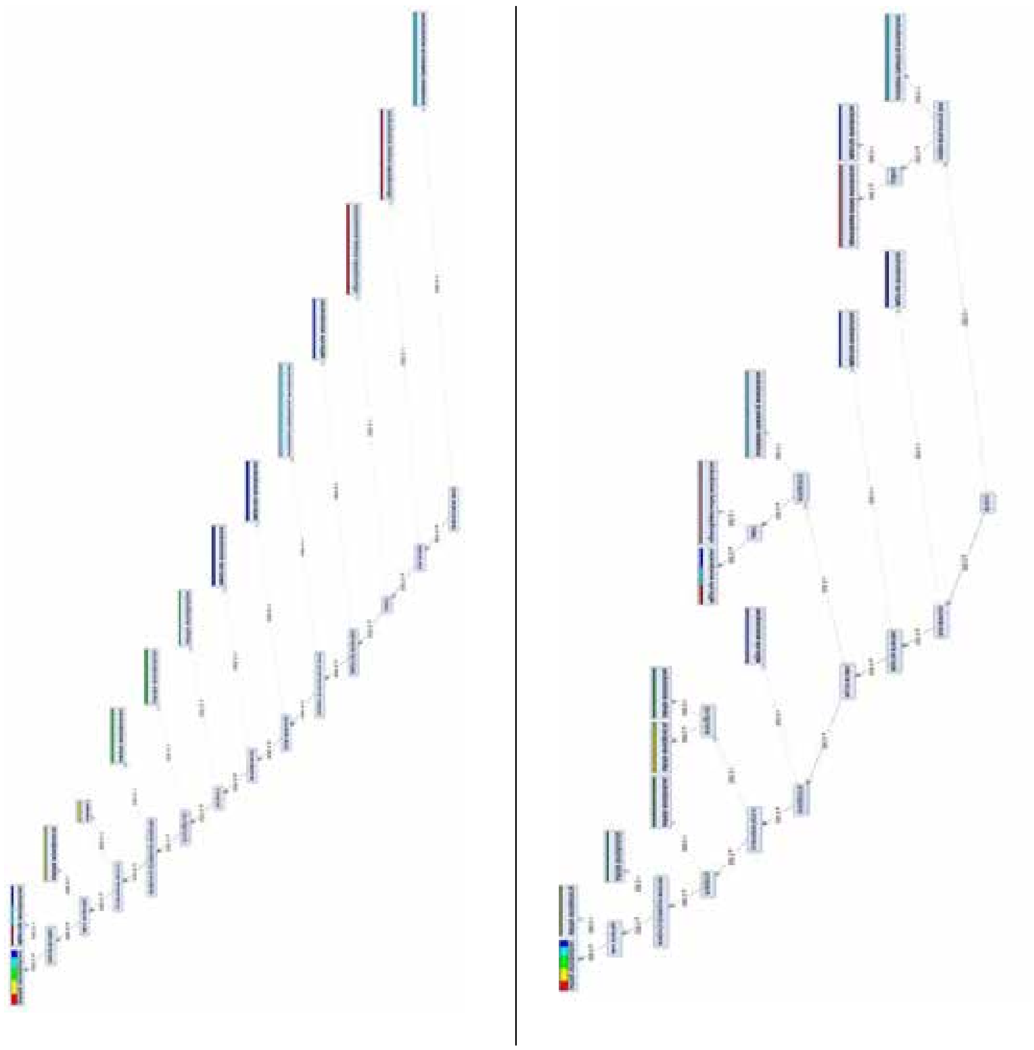
Development of the survey questionnaire was iterative and was divided into two stages. The first step was to develop a preliminary version of the form, submitting it to a few experts to verify the correctness of defined events. The second stage was to develop the ultimate version of the survey after taking into account remarks and comments of experts reviewing the preliminary version. Opinions and remarks obtained as a feedback from experts reviewing the questionnaires were very useful and influenced on the final shape of the form. The experts verifying correctness of the substantive passed remarks and changes made during the preparation of the final version of the form included:

- developing a more comprehensive introduction (manual) of the survey that includes a description of each column of the form,
- adding a question relating to the working conditions of the machine,
- adding a question relating to the type (model) of the machine,

- adding a question relating to the respondent's professional experience,
- changing the scale designed to assess the frequency of incidence of the events and their importance, on the 5-grade scale (from 0 to 4),
- modification of the names of certain operating events (common names being used by the experts were added).

The questionnaire consisted of 32 questions divided into four groups connected with the following subassemblies of the conveyor belt: main and auxiliary drives, rollers and runners, belts, and supporting structures. Questions in the survey concerned: frequency and importance of events, symptoms and causes of the events. Severity of each event was calculated as a product of frequency and importance, divided by the total number of filled questionnaires.

Knowledge acquisition process was carried out twofold: as an analysis conducted by a knowledge (exactly - data mining) engineer and second – data mining with the use of machine learning algorithms such as decision trees. Analysis of the respondents' answers conducted by the knowledge engineer consisted in estimation of the severity of each event, and exploration of the relationships between events and their symptoms. On the basis of the survey results the authors distinguished 10 most severe events occurring during operation of a conveyor belt (in the brackets the estimated severity value is given): backfilling of the rollers by the output (7.29), damage to the drive coupling (6.57), damage to the drive drum (6.57), motor overload (6.14), belt breaking at the joining points (5.86), damage to the gear reducer (5.71), scoring of the runners bearings (5,71), longitudinal trimming of the belt (4.86).



a) numeric values of attributes b) discrete values of attributes

Figure 3 – Exemplary outcome of decision tree algorithm application

Knowledge acquisition from experts was performed also with the use of machine learning decision tree algorithm. In order to apply machine learning algorithms the respondents' answers were transformed to the requisite form and served as an input to the algorithm. Exemplary outcomes in the form of decision trees have been presented in Fig. 3a and Fig. 3b. Both the decision trees were generated for the events occurring during operation of main/auxiliary drives of the belt conveyor. One decision tree (Figure 3a) was generated for numeric values of attributes whereas another (Figure 3b) was generated for discrete attributes values for the same dataset. Basing on the presented results one can state that numeric attributes values yielded better outcomes.

Knowledge acquisition by means of machine learning methods

The simulation experiment was used to obtain time series of instantaneous values of signals of process variables (74385 samples of the time series for faultless state and the same number of samples for faulty conditions). These signals were needed to estimate key time-domain features such as minimum, maximum, mean, peak-to-peak, root means square, etc. The process variable TP was applied to specify the range of the time window in which selected features were computed. In the next step, attribute-weighting method based on the symmetrical uncertainty was employed to find the set of relevant attributes for fault detection and isolation issues.

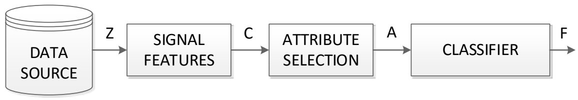


Figure 4 – Schematic diagram of knowledge acquisition with the use of single classifier learning methods

Knowledge acquisition with the use of machine learning algorithms was carried out for two cases. The first one is based on the scheme presented in Figure 4. As one can observe it is a classic scheme in which the basic learning methods may be used. The more complicated case is given in Figure 5 where two first tasks are processed in the same way as in the previous scheme, while the classification process is carried out using a set of three classifiers and a method for combining classification results obtained by these classifiers. The notation presented in the figures is as follows: Z is the set of the primary process variables (raw signals), C is the set of key time-domain features of signals, A is the set of relevant attributes (input attributes of classifiers), $\{F_1, F_2, F_3\}$ is the set of results of classification process obtained using base classifiers, F represents the final decision. The both classification schemes were designed for fault detection as well as fault isolation for the belt conveyor system. The fault detection task is to distinguish only between two classes f_0 - faultless conditions and f_n - faulty conditions. On the other hand, the main objective of fault isolation is to categorize the type of the fault (three classes are identified: f_1, f_2 and f_3). The first learning scheme (Figure 4) was prepared by means of basic classification techniques such as decision tree, artificial neural network, naïve Bayes and Bayesian network.

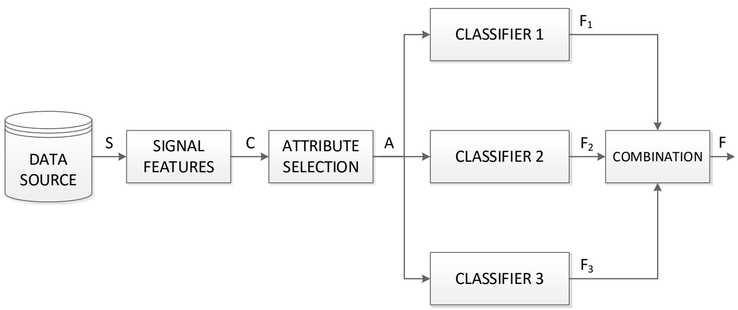


Figure 5 – Schematic diagram of knowledge acquisition with the use of classifier committee learning methods

In the case of the knowledge acquisition scheme with the use of classifier committee learning approaches (Figure 5) different variants of the first classifier and combination method were taken into account. It was assumed that the second classifier would be realized using Bayesian network, whereas the third classifier would be prepared applying naïve Bayes classifier. The following alternatives of the scheme were investigated:

- the first classifier based on a committee classifier and the majority voting approach proposed as the combination method,
- the first classifier based on a committee classifier and the meta-classification approach proposed as the combination method,
- the first classifier based on a decision tree and the majority voting approach proposed as the combination method,
- the first classifier based on a decision tree and the meta-classification approach proposed as the combination method.

The committee classifier was created with the context-based meta-learning method, where the base classifiers (decision trees) were switched using the contextual feature *TP*. A neural-based classifier was adopted in the meta-classification approach. The values of behavioural parameters of the learning methods were arbitrarily chosen by the knowledge engineer. The performance of the schemes was estimated by computing the accuracy rate. The cross-validation technique with a stratified sampling was selected in order to calculate this measure.

Table 1 – Accuracy of classifiers in fault detection and isolation tasks obtained for single classifier learning methods

Classification method	**Fault detection**		**Fault isolation**	
	Number of attributes	**Accuracy [%]**	**Number of attributes**	**Accuracy [%]**
Decision tree	76	95.56 ±2.01	15	100
Neural network	14	81.60 ±3.60	85	94.80 ±6.14
Naïve Bayes	20	64.32 ±3.68	89	66.65 ±3.82
Bayesian network	53	94.69 ±1.90	53	100

Table 2 – Accuracy of classifiers in fault detection and isolation tasks obtained for classifier committee learning methods

Classifier committees	**Combination method**	**Fault detection**		**Fault isolation**	
		Number of attributes	**Accuracy [%]**	**Number of attributes**	**Accuracy [%]**
Case No. 1	Majority voting	76	94.07 ±1.81	15	77.51 ±5.10
Case No. 2	Meta-classification		95.19 ±1.60		100
Case No. 3	Majority voting		94.94 ±1.95		99.76 ±0.73
Case No. 4	Meta-classification		95.68 ±1.27		100

The results of fault detection and isolation obtained using the first scheme are included in Table 1. The number of attributes was chosen taking into account the weights of attributes. It should be noted, that the highest performance was obtained for decision tree or Bayesian network classifiers in both the fault detection and fault isolation task. In case of the fault detection task these classifiers had the highest number of attributes, whereas in the task of fault isolation it was the opposite of that. It could be also seen, that the performance of a neural-based classifier was lower in the task of fault detection and that the naïve Bayes worked for the worst while classifying patterns in both the cases.

Table 2 presents the outcomes of fault detection and isolation obtained using the second scheme. In this case the number of attributes was chosen in the same way as above. The classifier committee for fault detection (regardless of the case of the scheme) was characterised by the performance result that was very similar to this obtained using the decision tree classifier. Moreover, a variant of the scheme based on decision tree, Bayesian network, naïve Bayes as well as the meta-classification approach allowed to obtain the highest performance result corresponding to the accuracy rate and the standard deviation.

CONCLUSIONS

The paper deals with the application of the DISESOR.DIAG expert system in machinery fault detection and isolation. The diagnostic expert system was developed as a module of the integrated shell decision support system for systems of monitoring processes, equipment and hazards. In this paper it was shown that our system could be successfully applied for knowledge acquisition and for reasoning with the use of multi-domain knowledge representations and multi-inference engines. The preliminary verification of the developed expert system shell was carried out taking into consideration the problem of knowledge acquisition from domain experts in the field of belt conveyor systems. In the next step, the developed tool was also applied in the case of the active diagnostic experiment. In this example, four different states of the plant were taken into account. Classic and context-based fault detection and isolation schemes were investigated using DISESOR.DIAG for fault diagnosis of a belt conveyor. These schemes were prepared adapting machine learning algorithms such as decision tree, neural network, Bayesian network classifiers, etc. Moreover, individual classifiers and committees of classifiers were studied in terms of their accuracy. These verification tests were carried out using the data generated by means of the belt conveyor simulator in which fault-free and fault scenarios were modelled. The performance results prove the correctness and effectiveness of the DISESOR.DIAG tool especially in the issue of fault diagnosis of belt conveyor systems.

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A PROPOSAL FOR DATA MANAGEMENT: A NECESSARY TOOL FOR TECHNICAL STAFF AND MANAGERS

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A PROPOSAL FOR DATA MANAGEMENT: A NECESSARY TOOL FOR TECHNICAL STAFF AND MANAGERS

ABSTRACT

In the current market context, it is required to propose proper solutions for continuous improvement process aiming to increase productivity with less cost performance. Technology and innovation are essential disciplines in business and without them, companies might take wrong decisions or lose the right timing on the decision making process. This occurs mainly due to the lack of information, or difficulty to access relevant information in a structured and "easy-to-interpret" way. In the mineral industry, we can notice that information plays a key role in evolution and innovation of mining production process, where many data are generated every moment. However, it is evident the existence of many gaps and deficiencies in the management of these data when it comes to production and quality controls and other key performance indicators (KPIs) management. The storage and treatment of these data, when not conducted in an integrated and systematic manner, result in mistakes, and require a lot of time, resources to treat them and probably will not represent the reality of the process. The development of new software and systems is aligned with the requirements of the mineral industry in an actual context, resulting in innovative solutions for KPIs management, productivity's improvement and targets' achievements.

One alternative to solve the demands for data processing systems is the creation of a software involving mine planning databases (long and short-term), production, geology's information and results of the metallurgical process. Through this intuitive software and user-friendly interface, it is possible to store and share data, generate and visualize different graphics and spreadsheets comparing the results and targets. The main result is a full compilation and structured availability of reports within a high level of quality control, production and reconciliation data along the complex mining process.

KEYWORDS

Database, innovation, customization, system, risk, management, kpi

INTRODUCTION

In the mineral industry a lot of information is generated in every stage of the production chain, according(Porter, 1985) for each produced tonnage, each sample collected and each controlling activity conducted inside the production process, data is generated and constantly feeding a database. By definition, database is a set of quantitative and qualitative information taken from a certain process that when organized will be able to provide a full picture of that process integrated to the production's cycle (Stair, 2010). Databases also play a key role in auditing and due diligence procedures, attesting the materiality and transparency of a work that has been done in a way that the efficiency of a certain process can be measured and adjustments can be implemented if needed.

What we see nowadays is that the mineral industry has evolved in terms of collecting numbers and indicators from the field or from the operating level. Today's employees already understand the need and importance of taking time during their work shift to register and to report what has been done, how it was made, who made it, the time and place, etc. We believe that employees are no longer complying with these activities by mere formality or obligation, but due to their understanding of the need for controls, as they want to know about their own performance as well as the performance of their respective production units, at a given time.

We highlight undeniable developments in the mining sector, in terms of generating information from the field, however, the main issue now concerns to the effectiveness of the treatment given to all collected data so they can produce information, knowledge and foresee capacity to the analyst. 'Factory floor' is fulfilling its role in portraying through spatial, temporal and numeric information from all activities performed in the field.

It is important to note, however, the existence of a gap regarding the management of all data generated, what they do reflect, how they are being stored, how and in what way they are being treated in order to have an effective result in the main Key Performance Indicators (KPIs). A KPI is an indicator defined by the company with the purpose to verify the real efficiency and performance within a certain process (Nader, 2012). It is a number that depicts the adherence of what has been planned to what is being done. It is the realistic foundation for an effective management, where a target is established and the productive system is assessed by means of the KPIs. Based on the results and depending on the measured performance through objective criteria, warnings and triggers are activated, serving as guidance for action plans.

In a continuous production process, actions being taken are required at all times and when they are taken without proper and quantitative support, without observance of KPIs, will probably lead to wrong decisions. It means that despite generating data, if they are not organized and structured, they may not properly help to understand the main indicators in time, therefore they will result in wrong decisions and ultimately in a waste of time, resources and money.

CONTROLS CURRENTLY USED IN MINING AND ITS CHALLENGES

Mining companies have been directing their investments to improve shipped technology through software applications due to the need for productivity's improvement aiming efficiency and costs reduction. It is currently noticed the existence of some options available in the market, for specific purposes such as production's and fleet's management, geological modeling, resources evaluation, mine planning, etc. It is evident the evolution of mining in the use of specific software, however it is also noticed the absence of a consolidated single database that could concentrate each and all system's results in a standardized way, in order to facilitate, organize and offer good, online and real time information access for its users and compile results and summarize and translate in a form of KPIs to managers (Sachs, 2009).

The Figure 1 shows an information flow that depicts what happens in most mining operations nowadays. A data is generated through specific software, numbers are stored in electronic spreadsheets scattered on each area's client terminal, and the users must seek for this particular piece of information in every sector responsible for the generation of each data, in order to provide consolidated reports, controls and to perform general management.

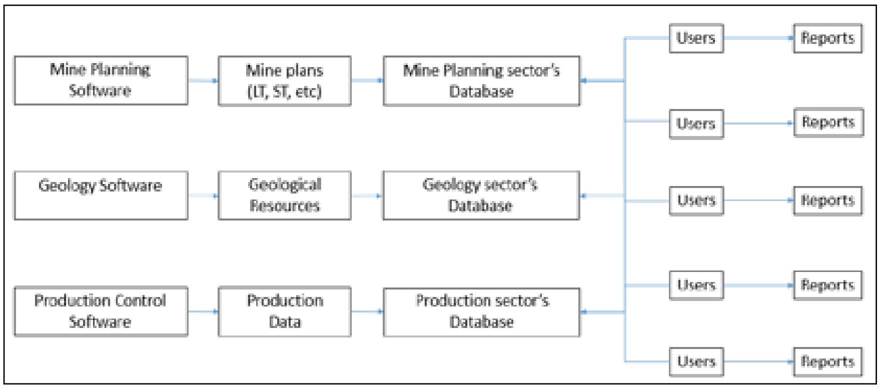

Figure 1 - Information flow in most mines today

Electronic Spreadsheets

The information flow process currently used in most mines, despite the use of modern software and hardware for generating data, usually results in outputs ultimately organized within electronic spreadsheets, which creates a huge potential for mistakes and loss of data integrity. In terms of information risk, it is known that spreadsheets carry by definition limitations on number of rows and columns, i.e. they do represent a risk in long-term data storages. Regarding the information risk, those who created a spreadsheet usually have a full understanding of what is being controlled, but do not provide the necessary training for third ones in order to allow them the same understanding to control the data. This fact is extremely risky in case of an absence or replacement of that particular employee, who owns the complete understanding of the spreadsheet and why not the path that lead to it once filed somewhere.

The outputs generated by each management staff usually do not contain a standard interface to be adopted by all users. Thus, data are 'copied and pasted' from one spreadsheet to another in order to achieve a desired format for a particular report. This procedure, besides not allowing up to date and organized reports following a pre-defined standard, it has an increased probability of errors.

Once that data is generated and stored in different places, users from different sectors can have access to the files, redundancies are created, *i.e.* the same data from a particular software, from a specific area may be replicated multiple times in different manners and with different names, resulting in a strong likelihood to be incorrectly reported. Besides duplicities, several areas store different versions of the same data and therefore consume unnecessary storage capacity on servers.

Management Software available in the market

There are some softwares in the market that provide ways to generate indicators and management tools linked with production and reconciliation data (for example: "*Smart Mine*" from Hexagon, "*Reconcilor*" from Snowden, among others), however most of these softwares are systems that require a huge investment, and come with pre-defined interfaces, tools and routines, already designed independently of the mine in which they will be installed.

Mining sector is recognized for being a slow innovation environment due to a key factor which is the resistance of its collaborators to changes, old mentality and also stopped by the high investment values usually required by these robust and complex software. As a result, companies continue with their old controls as they were used to do, using unprotected and limited electronic spreadsheets, however, they must understand the fact that costs involved with lack of information security and organization can be much higher than the investment to keep a reliable and easy-handling database system for management and control. On the other hand, when even trying to break the resistance barrier in the migration to a high level system, the company that decides to acquire the so called 'shelf product' software, normally faces challenges such as the resistance of their employees, who by having to migrate to robust, complex, little flexible software, end up again returning to their controls using spreadsheets, due to the need for quickly and dynamically generate information (but with all problems and limitations listed previously). The implementation and migration from old controls to modern management software requires time and training and very often over a continuous production cycle. This is a key factor for the failure of normally referred "black box" softwares.

EXPERIMENT APPLIED TO MINING

At AngloGold Ashanti was created the demand for a more innovative way to store data and provide control information in replacement to the electronic spreadsheets. The following challenges were initially observed:

• The existence of controls made over spreadsheets without a database format, made the process more time-consuming with low decision-making capacity when a system failure occurred.

• There was a delay of over a week for the daily/weekly mining production to be calculated and reported through a spreadsheet, in which months of production were separated in different tabs. This fact made action plans difficult in the case of low adherence between what was planned and what was being done (ROM x feed grade).

• Multiple users from other areas created links on this spreadsheet that was stored on the Geology Department drive. It means that there was a risk of violations and / or potential mistakes associated with the weakness of access to the so called “database”. The spreadsheet became so large and consequently so ‘heavy’ and slow that did not allow multiple users to have access and update it simultaneously.
• The spreadsheet was developed by a collaborator that no longer belongs to the company’s staff and there was not available any guideline or tutorial of how to handle with it. Only a few people had know-how and therefore there was a risk of losing data and challenges in the updating continuity if there was any failure within the spreadsheet.
• Lack of automatic reports and graphics, which required data to be copied to another spreadsheet to obtain the required graphs, a factor that increased the possibility of errors in the information being transmitted and also the time from the employees. The fact that reports and graphs do not have a standard format did not represent the necessary reliability needed for information.

Due to those issues, the grade control process became very slow and with an intense demand for obtaining information and preparing management reports in a period of time a way faster than it could update the numbers in the spreadsheets. To overcome this situation, the following actions were taken: First of all, a compilation of all different existing spreadsheets in a single database through a dynamic and complete spreadsheet (Figure 2):

MINE PRODUCTION AND RECONCILIATION SPREADSHEET

ROM

YEAR	MONTH	DAY	AREA	SUB-AREA	GEOMETALLURGY	LEVEL	BODY	STOPE	INTERLEVEL	POSITION	METHOD
2015	MARÇO	18/03/15	1	A	XIST	1	BERILO	a	1	LE	SUBLEVEL
2015	MARÇO	18/03/15	2	B	SULFID	2	AMETISTA	b	3	LD	SUBLEVEL
2015	MARÇO	18/03/15	3	C	BIF	3	QUARTZO	c	3	LD	CUT AND FILL
2015	MARÇO	18/03/15	4	D	BIF	4	BALANÇÃO	d	3	LE	SUBLEVEL
2015	MARÇO	18/03/15	5	E	SULFID	5	BALANÇÃO	a	1	LD	CUT AND FILL

Figure 2 – Production, grade control and reconciliation through a spreadsheet’s format database

• Additional fields were added to the database so that more information can be handled (e.g. mined lithotypes, geometallurgical information).
• The controls are now reported daily; where the historical production results were extracted from a proprietary software through the export of data to a spreadsheet and then pasted on a single data spreadsheet. This information flow process, although more organized than the old routine, still requires more dynamism and security control.

The proposed system consists in the creation and customization of import/export routines and data storage from planning and mine production areas, geology and production of the metallurgical plant in one single database, as shown in Figure 3 below:

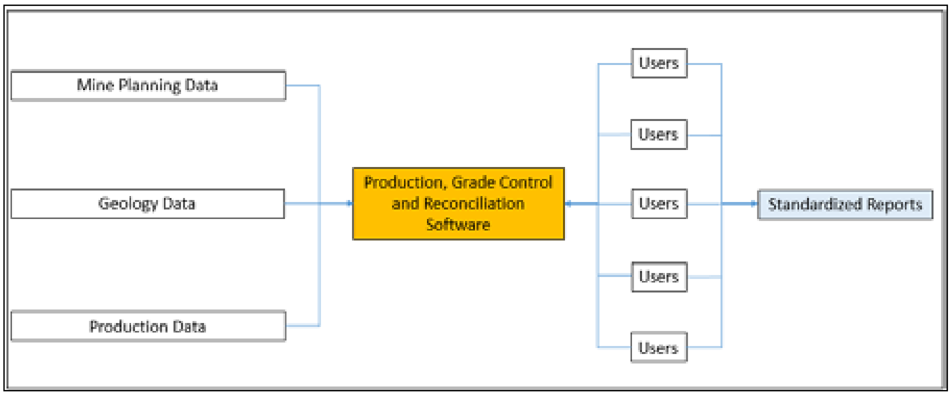

Figure 3 - Data flow customized system

Through this system, it is possible for users to filter its content based on the specific desired data and according to the requested time period.

Figure 4 illustrates system’s interface, where the main graphs were arranged in a Dashboard format, for an easier management of key indicators and online production reports (e.g. daily and accumulated production - planned **x** executed **x** crushed), Daily and accumulated grades of gold and sulfur (planned **x** rom **x** feed grade) and daily and accumulated production in ounces (planned **x** accumulated **x** bullion).

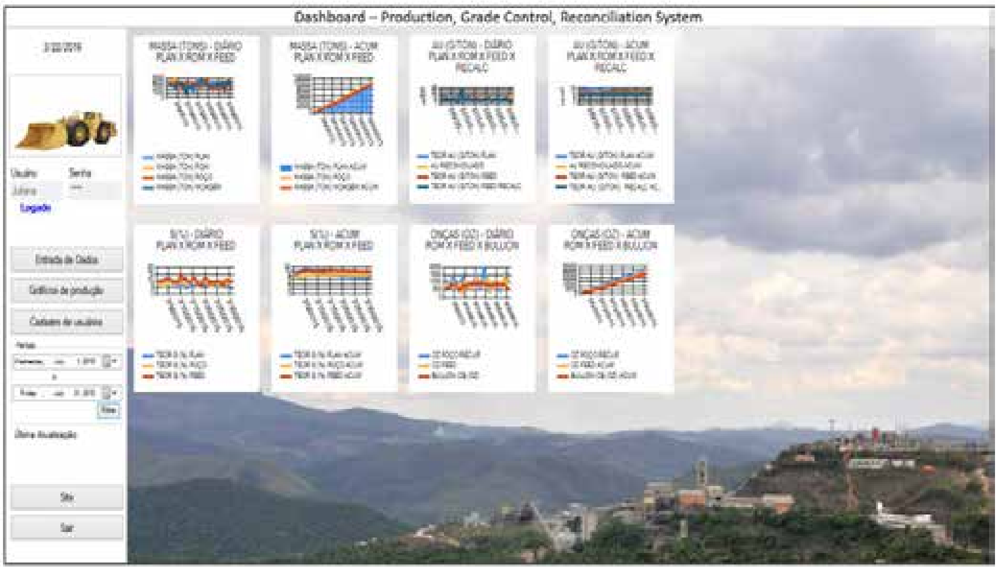

Figure 4 – Initial screen of customized system

The system has different access permissions and data input levels (Figure 5), which are managed only by system administrators. To general users, access is allowed only for graphs and report viewing.

The picture below shows different types of production reports customized for the system through the generated database. However, many other reporting and data integration could have been programmed from the same database, where each area manager is responsible for requesting the desired customization.

Figure 5 – System´s data input interface

In Figure 6 shows some of the graphics particularly designed for the system.

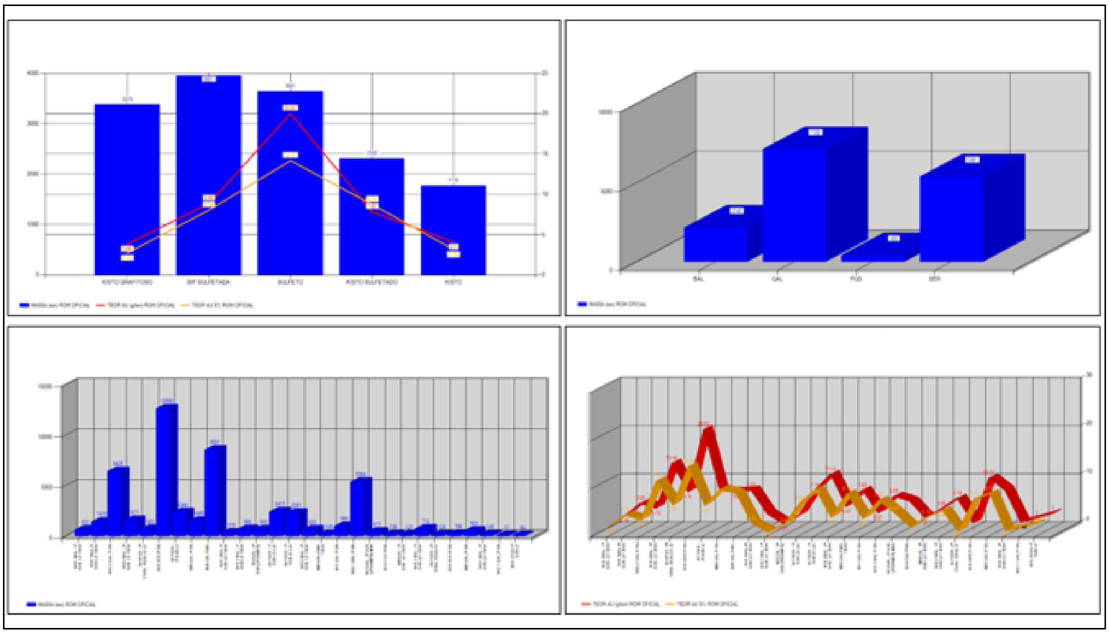

Figure 6 - Specific Graphs – Production, Grade Control and Reconciliation System

The system allowed online graphs and customized reports with more agility and safe to decision making.

CONCLUSIONS

When it comes to innovation, mining still requires a long walk ahead to get to levels where other sectors have already reached. To make it happen, the mentality of the professionals working in this sector must be focused and opened to new ideas, to incentives in favor of more dynamic processes. In order to take good ideas forward it is necessary to prove that innovative processes, even with deployment challenges faced within a continuous production cycle, will excel the old processes. And for managers to approve changes, an evidence of the proposal's effectiveness needs to be presented, and for the facts to be properly shown, data must be concisely generated.

Through the presented work and the real case study developed at AngloGold Ashanti it is clear the importance of an accurate and reliable database's formation. Thereafter, the way these data are treated is of great importance in order to avoid a bunch of scattered data, where users do not have confidence, agility and the necessary speed to handle information for a proper decision making process. According to AngloGold Ashanti, this project was a success and increased the agility up to 60% in data's acquisition, consolidate information and reports generation (Figure 7). After migration from the old system (spreadsheets based) to this customized interface there was a considerable improvement in the traceability of information, in data reliability and reports accessibility by a most of the users involved with the production process.

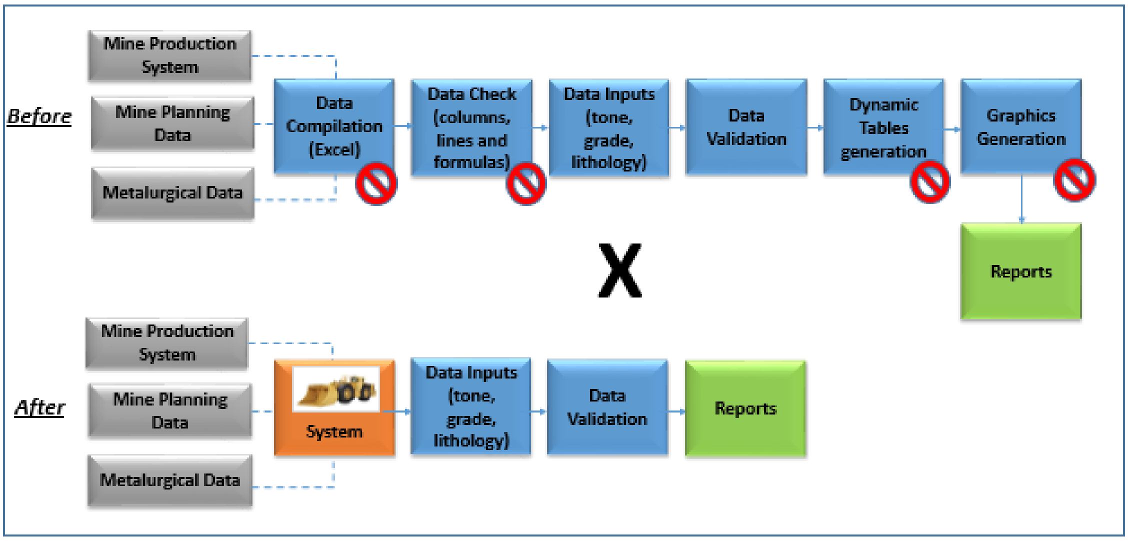

Figure 7 – Database and reports generation process - Previous X Current - AngloGold Ashanti

More than just an idea, in mining it is necessary that managers encourage their staff to create, because despite the existence of resources it is a fact the existence of a certain self-indulgence and inertia. Ideas exist but what causes the failure and non-realization of an idea is indeed its applicability, its execution. If the production team did not take the idea as theirs, it would never leave the paper. Because of that, it is believed that the solution to innovation's practical application would be, first instance, to encourage professionals to recognize new ways to make their routines more productive in order to provide solid ground to take their decisions with agility and speed.

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AN INTEGRATED APPROACH TO OPTIMISATION OF LONGWALL GAS DRAINAGE

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AN INTEGRATED APPROACH TO OPTIMISATION OF LONGWALL GAS DRAINAGE

ABSTRACT

An effective mine gas drainage system in modern coal mining need to have the ability to not only control gas emissions to the workings for safety reasons, but also to reduce mine fugitive emissions. In Australia, gas drainage in longwall mining is becoming increasingly complex and challenging, and a step-change improvement in mine methane drainage strategies and technologies is required. Under the auspices of the Australian Government Coal Mining Abatement Technology Support Package (CMATSP), a major research project has been carried out at the Blakefield South mine in Australia to develop a holistic and optimal approach of planning, design, and implementation of mine gas drainage. This paper presents the integrated approach developed for the project and, as a result, an innovative gas drainage system which adopts underground directional horizontal boreholes. The trial of this drainage system at the mine was highly successful, resulting in a significant reduction of fugitive emissions, remarkable improvement of gas control, and major reduction of production delays.

KEYWORDS

Coal mine methane, longwall mining, post drainage, greenhouse gas emissions.

INTRODUCTION

Gas drainage in longwall mining in Australia is becoming increasingly complex and challenging with deeper mines, multi-seam environments beneath existing goafs, and surface environments where drilling conventional surface gas drainage wells is more constrained. Increased longwall retreat and development rates at a number of Australian coal regions, including the Hunter Valley, Illawarra and Bowen Basin, have produced mine gas levels that are a serious threat to sustained and efficient coal production, and potentially lead to increased fugitive emissions. A step-change improvement in coal mine methane drainage strategies and technologies is required to effectively capture methane from the longwall goaf, surrounding seams and strata before the methane enters the workings and ventilation system.

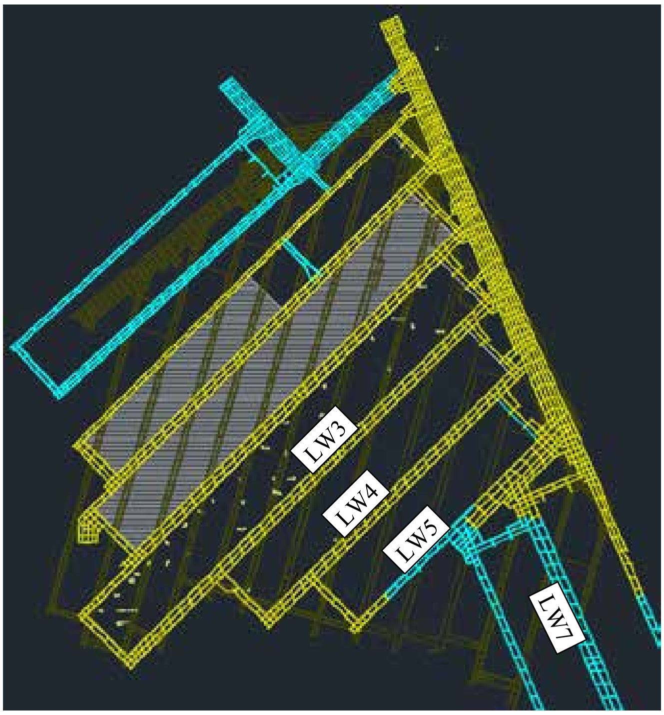
A joint research project between the Glencore Bulga Underground Operations (hereafter referred to as “Bulga”) and the Commonwealth Scientific and Industrial Research Organisation (CSIRO) has been carried out, under the auspices of Australian Government Coal Mining Abatement Technology Support Package (CMATSP). The project aims to develop and demonstrate a holistic and optimal approach to planning, design and operational control of mine gas drainage to maximise methane capture and minimise fugitive emissions in gassy and multiple seam conditions.

This paper presents the integrated approach developed to optimise mine gas drainage planning, design and implementation, and an innovative gas drainage system that has resulted in a major improvement in longwall gas control, mining productivity, and fugitive emission reduction at the Blakefield South mine in Australia.

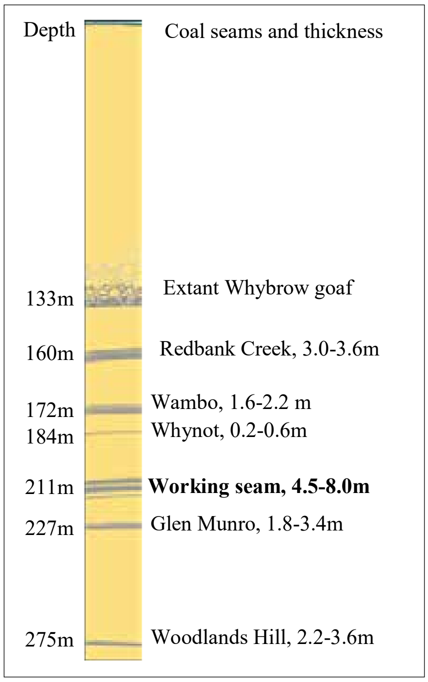
SITE CONDITIONS

The Blakefield South mine of the Bulga Underground Operations is located approximately 140 km north of Sydney, and 60 km north-west of Newcastle in the Hunter Valley Coalfield. The longwalls operate, under old mine workings, in highly gassy conditions and multi-seam environment. Figure 1(a) shows the longwall (LW) mine layout, where the studied panels LW3 and LW4 are highlighted. LW3 was 400 m wide (rib-to-rib) and about 3.5 km long.

The working seam, Blakefield, dips at about 3 degrees and its depth of cover varies between 130 and 260 m. The seam comprises a number of plies with a total thickness ranging from 4.5 to 8.0 m. The extraction height ranges from 2.8 to 3.4 m. Figure 1(b) shows a stratigraphic section about LW3. The existing Whybrow goafs lie approximately 80 m above the working seam. Surrounding coal seams shown in the figure were all predicted as sources of gas emissions with the empirical prediction method, the Flugge model.



(a) Longwall layout



(b) Stratigraphic column

Figure 1- Mine plan and stratigraphic of the Blakefield South mine.

METHODOLOGIES AND PROCEDURES FOR OPTIMAL GAS DRAINAGE DESIGN

The ultimate objective of gas drainage design is to provide sufficient drainage capacity at optimally positioned drainage points that not only captures an optimal fraction of goaf gas emissions at an acceptable composition but also reduces methane emissions into ventilation. Gas emissions during longwall mining is a dynamic process involving complex interactions between mining induced strata de-stressing, fracturing, de-watering, ventilation and gas drainage. A clear understanding of these complex interactions is therefore crucial in determining key parameters for gas drainage design, such as gas emission sources, emission patterns, and borehole placement.

Based on the mine site condition, and relevant previous studies and experiences (Balusu et al., 2002; Guo, et al., 2009, 2012, 2013, 2015a, 2015b) in mine ground and gas control, an integrated approach consisting of site characterisation, field investigation, coupled numerical modelling, design with CFD test and optimisation, site trial and monitoring, and retrospective analysis was developed. Figure 2 illustrates the components and logic of the integrated approach.

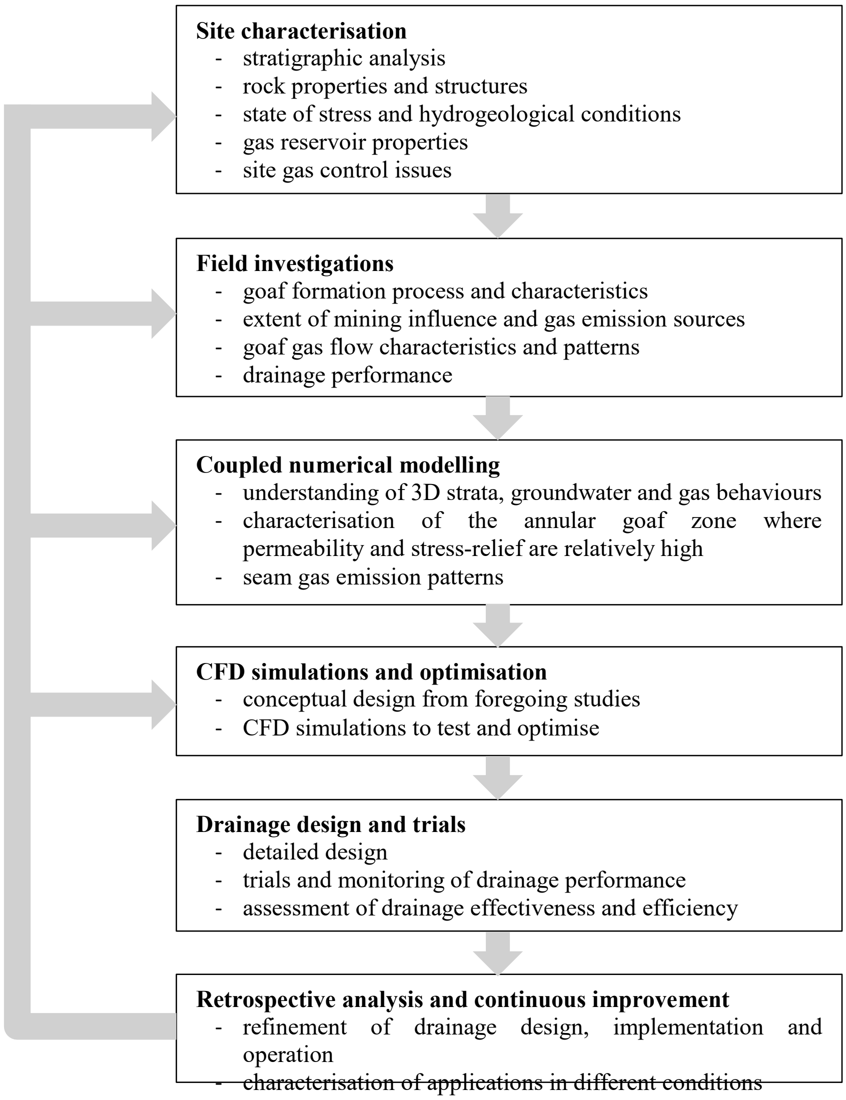


Figure 2 - The integrated approach for optimal goaf gas drainage design

The integrated approach consists of six distinctive components. This approach was adopted for the design and optimisation of a gas drainage system at the Blakefield South mine. Comprehensive field and numerical studies were conducted at LW3. The first field trial of the new drainage system was planned at the initial 400 m of LW4.

Site characterisation

All available data related to the first component in Figure 2 were collated and analysed. Key features and characteristics of mine geological, geomechanical, hydrogeological and gas conditions were characterised, and key issues associated with gas drainage were identified. Some key observations are listed below:

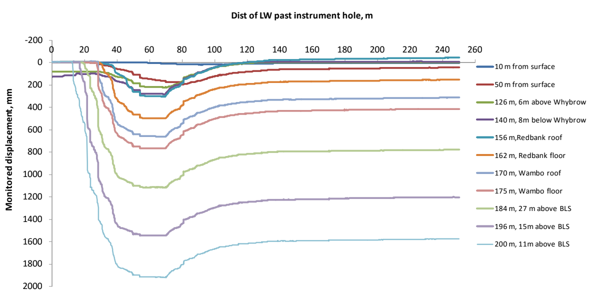
- The overburden is highly banded, consisting of sandstone, conglomerate, siltstone, shale and tuff. No major competent layers such as sandstone overlie or underlie the mining seam within the depth of interest.

- No major geological structures such as faults and dykes that have a significant influence on gas reservoir conditions are present.
- The old Whybrow goafs, overlying around 80 m above the current mining level, have a significant influence on the goaf gas drainage performance. It is evident that a significant portion of the goaf gas drained by the surface vertical wells was from the Whybrow goafs.
- The performance of the surface vertical wells has not been satisfactory in controlling goaf gas emissions over the last few years, particularly in the initial 400 m from the longwall start-up. Frequent longwall production delays were experienced during the first 400 m retreat of LW2 and LW3, totalling about two months at each of the panels.
- Some surface vertical wells at LW2 were operated for only a few days due to unacceptable high levels of oxygen in the drainage gas and coal spontaneous combustion risks.

Field investigations of strata, ground water and gas behaviours

Based on the results from the site characterisation work, a comprehensive field monitoring and measurement program was planned, designed and carried out at LW3 to investigate the responses of strata, groundwater and coal seam gas to the mining. The field investigation covered overburden caving processes with surface deep-hole extensometers, stress changes about the longwall face, seam pore pressure changes with surface deep-hole piezometers, goaf gas pressure and flow paths, continuous gas drainage performance, seam gas contents before and after mining, and surface goaf well integrity inspection with special borehole cameras. Key findings from the field investigation include:

- Mining induced fractures and delaminations extended quickly up to the two overlying coal seams (36 m inbye of the longwall face) and significant delaminations were observed. The monitoring data from one of the two surface extensometers is given in Figure 3(a).
- The piezometers installed at the unminded site chain pillar of LW3 showed coal seam pore pressure decreased quickly between 50 m outbye and 100 m inbye of the longwall face. All coal seams except Woodlands Hill listed in Figure 1(b) experienced significant depressurisation.
- Tracer gas tests showed that Whybrow goaf gases did not vertically flow down to the active goaf and contribute to the ventilation gas makes.
- Gas content measurement before and after mining LW3 indicated that gas emission sources were both the overlying coal seams Redbank Creek and Wambo and the Glen Munro seam in the floor. This result was different to that predicted by the Flugge Model in the Woodlands Hill seam which was a significant emission source.
- Vertical wells located in the tailgate (TG) side of the longwall generally captured more gas than those located in the mid panel. The former showed better connectivity to rich gases in the goaf than the latter.
- Surface goaf well inspection clearly showed the wells were often blocked at levels higher than 30 m above the mining seam. An example of well blockage is given in Figure 3 (b).



(a) Overburden movement

(b) surface vertical well blockage

Figure 3 – some key results obtained from field investigation

Coupled numerical modelling

The field investigation results of strata, gas and groundwater behaviours were extrapolated into a 3D dimensional space by numerical modelling with CSIRO's unique code COSFLOW. Description of this code can be found from Guo, et al., (2009; 2012). The shape and extent of the fractured zones and the permeability changes and distributions within the zones were first assessed from the geomechanical model validated by field investigation results. Coupled modelling was carried out to determine the characteristics of gas emission zones and various gas emission sources over the entire longwall panel.

Figure 4 shows the 3D geometry model constructed for the modelling study and Figure 5 presents some key results. It can be seen that the goaf perimeter area, shaped as an annual zone, has relatively high stress relief and high permeability. Fundamentally, goaf gas drainage from this zone will achieve a consistent and high flow rate of gas with high methane concentration.

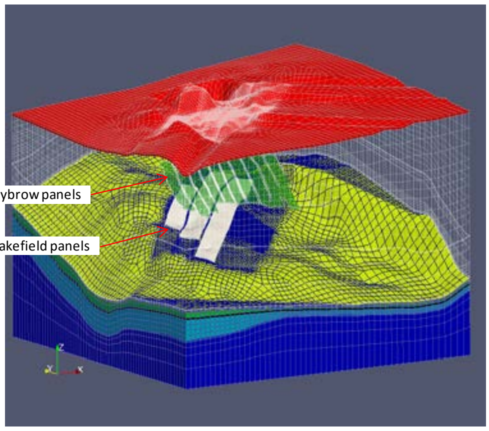
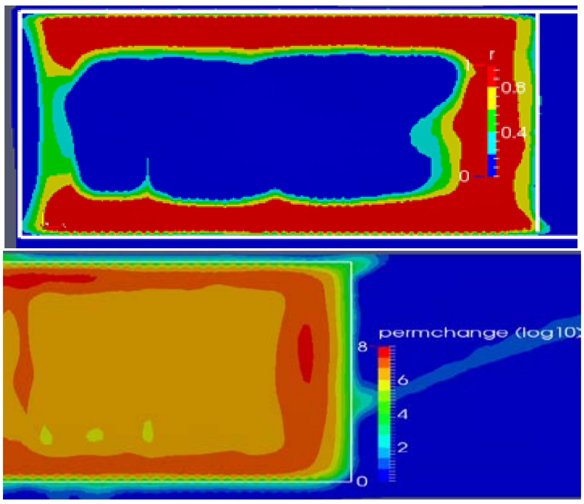
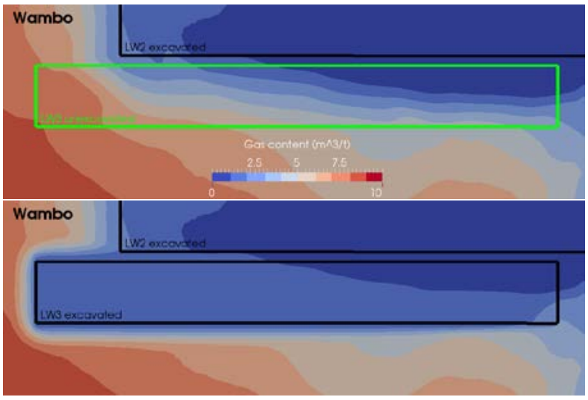


Figure 4 - 3D geometry model constructed for COSFLOW modelling studies



(a) vertical stress reduction ratios (stress reduced to pre-mining stress) (b) horizontal permeability changes



(c) gas content before mining (d) gas content after LW3 mining

Figure 5-Modelled strata de-stressing, permeability change and gas emission patterns

CFD simulations and optimisation

The field and numerical studies provided a clear understanding of site conditions, issues with the mine's surface vertical goaf wells, coupled behaviour of strata, gas and water, and gas emission sources. The surface vertical wells were inefficient in providing continuous and immediate capture of gas emissions due to blockages at 30 m and above from the workings, and not an optimal method for minimising fugitive emissions as significant extra gases were captured from the old Whybrow goafs.

As a result, underground horizontal boreholes were proposed to replace the surface vertical wells. In comparison, these underground horizontal boreholes not only can provide continuous and immediate capture of gases from the immediate area of the goaf behind the longwall face but also can avoid capturing Whybrow goaf gases. Placement of the boreholes should be (i) laterally located at the tailgate side into the annular high permeability and methane rich zone; (ii) vertically located in the lower fractured zone above the caved zone; and (iii) in the Glen Munro seam to intercept gas from flowing up to the goaf.

The conceptual design was tested and optimised with CFD simulations. Figure 6 shows the base CFD model constructed for LW3 for calibration purposes. The input parameters were based on the field data, and the model was calibrated by the methane concentration and flow rate in the surface goaf vertical wells as well as by current knowledge of goaf gas pressure distribution. After validation of the model, extensive simulations with various horizontal borehole placements and configurations were tested.

Figure 7 shows the goaf gas flow dynamics with an optimised design of horizontal boreholes which consists of five roof horizontal boreholes located in the tailgate side with a diameter of 150 mm. It can be seen that sinks of low pressure are formed around the boreholes (Figure 7 (a)) and that, under the low pressure sinks, gas is induced to flow towards the drainage boreholes away from the workings (Figure 7 (b)). These results demonstrated that the horizontal drainage boreholes can create low pressure sinks that protect the workings from goaf gas ingresses by changing goaf gas flow directions. The test also showed that, under this design, the boreholes have sufficient capacity to efficiently reduce methane levels in workings.

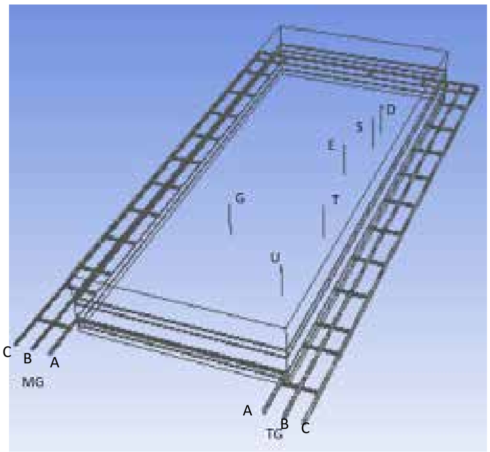
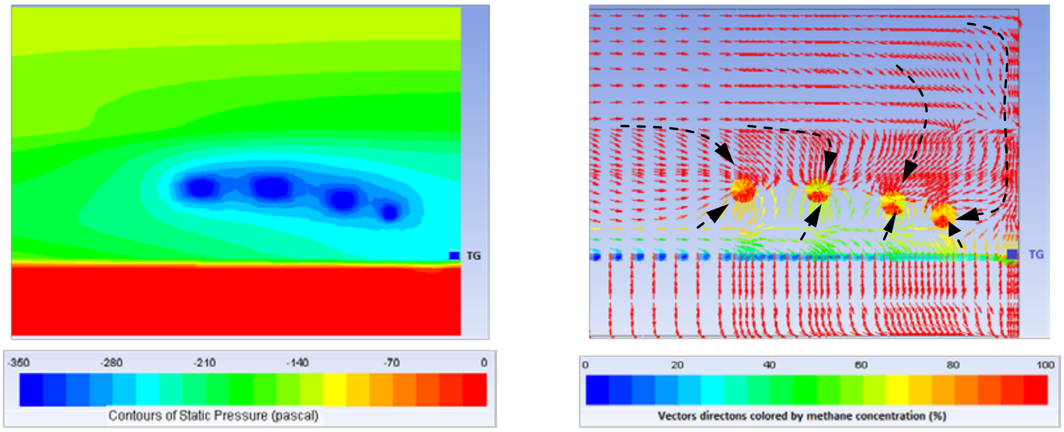


Figure 6 - CFD model geometry

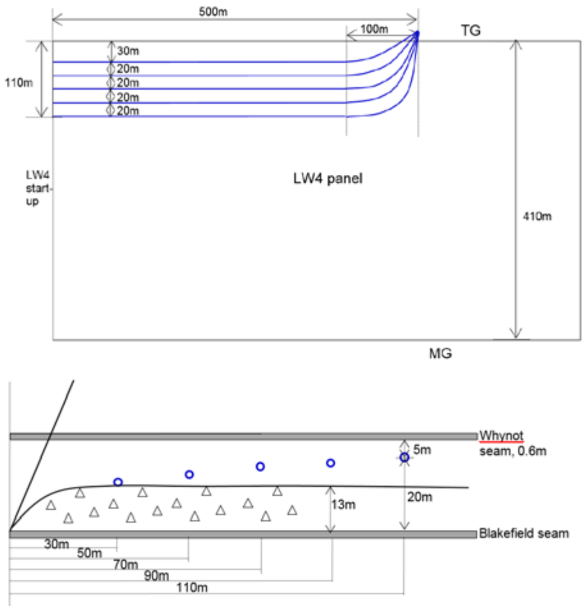


(a) Goaf gas pressure

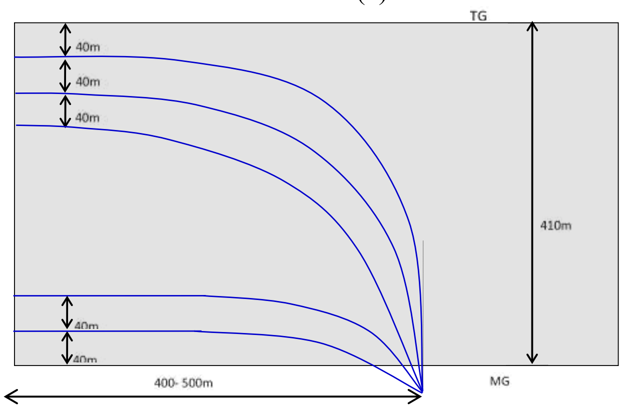
(b) Goaf gas flow directions

Figure 7 - Pressure contours and gas flow directions on a vertical section 20 m behind face

Following the CFD tests and optimisation, a design of underground horizontal boreholes was made as shown in Figure 8. In the design, five boreholes in the roof and five boreholes in the floor were included. The roof boreholes were located within 150 m of the ventilation return and vertically situated at a location between 13 to 20 m above the mining seam. The floor boreholes were designed to steer along the Glen Munro seam, with three boreholes located in the ventilation return side and two boreholes in the intake side.



(a) Plan view of roof horizontal boreholes (b) Cross-section of roof horizontal boreholes



(c) Plan view of horizontal boreholes into the Glen Munro seam

Figure 8: Design of horizontal gas drainage system with underground lateral boreholes.

SITE TRIALS AND ANALYSIS

Implementation and monitoring scheme

The design was trialled at LW4 at its initial 400 m of retreat. The implemented borehole layout is shown in Figure 9. The configuration of these boreholes is summarised below:

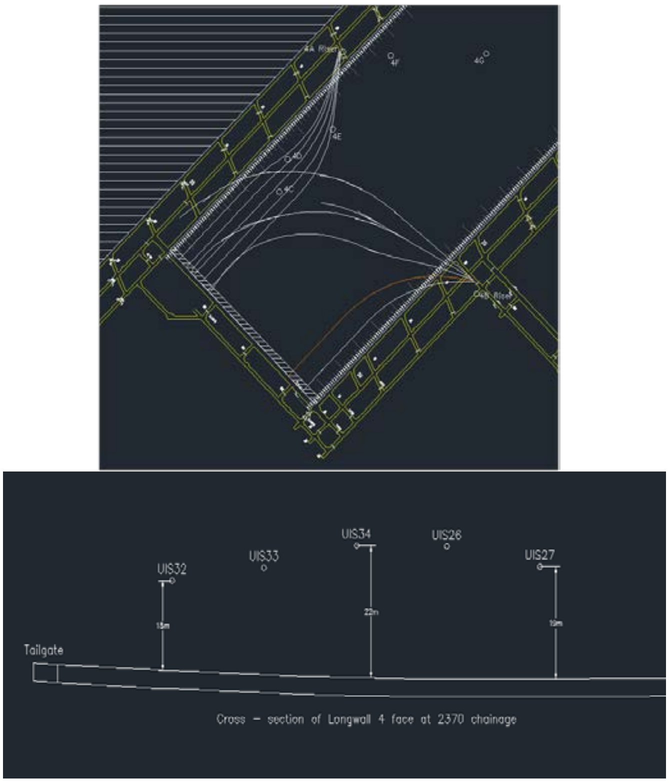
- Five roof horizontal boreholes were located within 105 m from the ventilation return, with the nearest one 25 m from the longwall void edge. The spacing between every two boreholes was about 20 m.
- The roof boreholes were situated at 15-22 m above the mining seam, with the first one from the return at the lowest level.
- The roof boreholes were 145 mm in diameter, reamed from 96 mm.
- The horizontal section covered a distance of about 350 m.

- Five boreholes were drilled into the Glen Munro seam, with three in the tailgate side (26 m outside the LW4 panel to 105 m inside LW4 panel), and two in the intake side (75 m of the maingate (MG)).
- The floor boreholes were not reamed and were 96 mm in diameter.

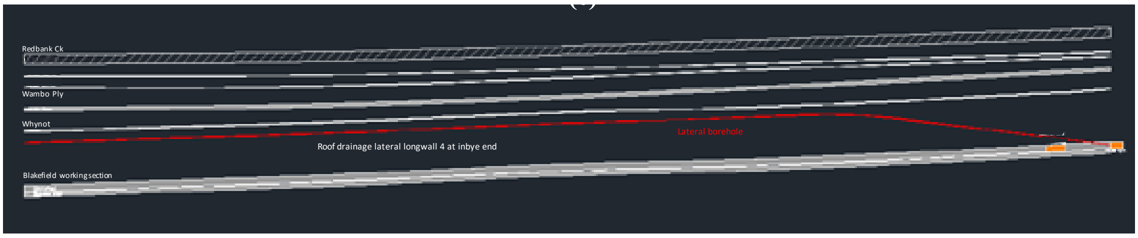
The two sets of roof and floor boreholes were each connected to a riser, drilled from the surface to a cut-through point. Both risers were 305 mm in diameter. Only the roof borehole riser was connected to the surface goaf drainage plant to provide suction pressure to the underground lateral boreholes. Gas flow in the floor horizontal boreholes was driven by seam gas pressure only.

Continuous monitoring of gas drainage performance was carried out. Both risers were equipped with a wellhead, which enabled continuous monitoring of suction pressure, drainage gas flow rate, gas composition, and operation parameters such as borehole opening percentage. Tube bundles were run down the roof borehole riser and connected to each of the roof boreholes to monitor gas composition. Continuous monitoring of the individual borehole flowrate was not enabled but manual readings were taken. In addition, a test of gas drainage performance with different operational parameters, such as various boreholes in operation and borehole opening percentage, were conducted. For the floor horizontal boreholes, no continuous monitoring of flow rate and methane concentration was implemented for individual boreholes.

Surface vertical wells were implemented in the remaining part of LW4. There were also three surface vertical wells drilled within the trial area for a transition from horizontal drainage to vertical drainage. The first one, 4C, was located 240 m from the longwall start-up.



(a) Layout of the trialled innovative goaf gas draiange system at LW4 (b) cross-section along mining face



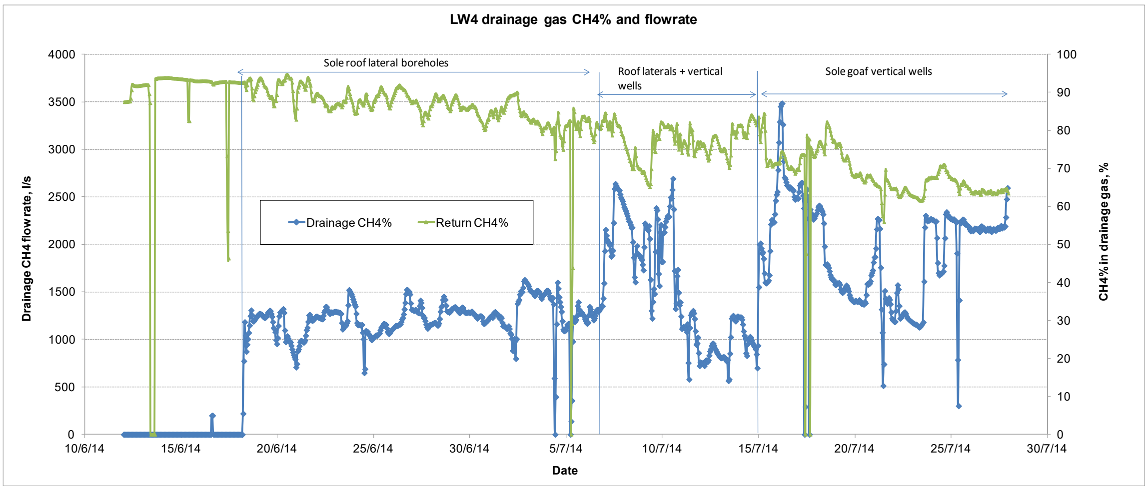
(c) Cross-section along mining direction

Figure 9 – Layout of the trialled underground horizontal drainage boreholes at LW4

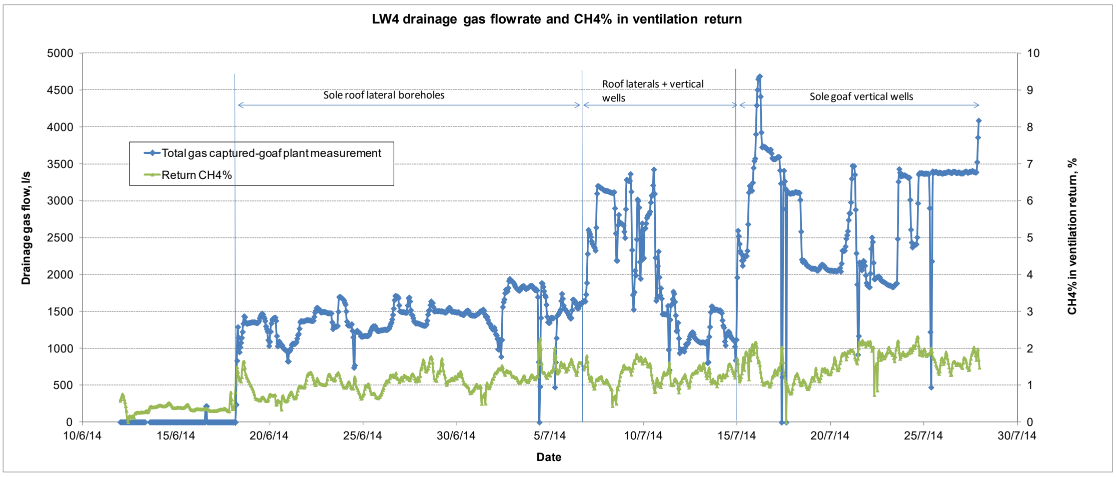
Gas drainage flow rate and methane concentration

Figure 10 (a) and (b) show the drainage flow rate and methane concentration monitored at the goaf plant in the first two months of LW4 operation, which covers both the underground roof horizontal boreholes and the conventional surface vertical wells. From the figure, it is clearly seen that:

- The roof lateral boreholes captured goaf gas with a continuous and consistent flow rate; while the vertical goaf wells gas flow rate fluctuated significantly.
- Methane concentration in the roof horizontal boreholes was high and averaged 86%; while in the vertical wells, methane concentration averaged 68.3% and varied significantly.
- Methane concentration in the ventilation return was remarkably lower when the roof lateral boreholes were solely in operation (average 1.13 %) than when goaf vertical wells were solely in operation (average 1.61%). This clearly shows that the roof lateral boreholes significantly reduced methane emissions from longwall operation into the ventilation circuit.
- Overall, in comparison to the goaf vertical wells, the roof horizontal boreholes captured less gas but achieved a better result in controlling gas emissions into the ventilation circuit.



(a) goaf drainage flow rate and drainage methane concentration



(a) goaf drainage flow rate and methane concentration in ventilation return

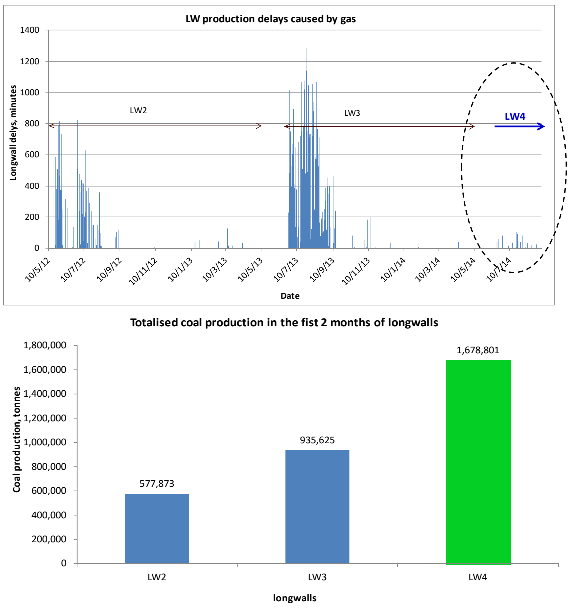
Figure 10- Goaf gas drainage performance of the field trial of the optimal gas drainage design

The floor lateral boreholes performed well with continuous and stable gas flow and consistently high methane purity (92%, the rest being mainly CO_2) prior to and during mining. The gas flow rate from the lateral boreholes was at about 400-450 l/s.

Drainage efficiency and longwall coal production

Gas drainage efficiency is calculated as a percentage of the drainage gas volume in the total gas emissions from the mining operation. The gas drainage efficiency during the trial period was significantly improved and reached around 80%, compared to 14-37% in the counterpart (the initial 300 m of longwall retreat) at LW2. The drainage efficiency fell again when the roof lateral boreholes were closed and only surface vertical wells were in operation.

Figure 11 (a) shows the recorded daily longwall production delays from the commencement of LW2 to LW4. Significant production delays were seen in the initial 2-3 months of mining operation at LW2 and LW3, where surface vertical wells were used. Conversely, at LW4 where underground lateral drainage boreholes were implemented, very limited delays occurred. The significant reduction of longwall delays at LW4 trial led to a remarkable increase of coal production in the initial mining stage, as shown in Figure 11 (b). LW4 coal production, in its first two months, increased by 79% compared to LW3.



(a) longwall production delays (b) coal productin in the first 2 month of LW2 to LW4

Figure 11 – Comparison of longwall production delays and coal production at LW2, LW3 and LW4

Reduction of methane emissions to atmosphere

The trial results showed that, in addition to a significant increase in capture efficiency with the underground lateral boreholes, the total specific gas emission was significantly reduced for longwalls operating in the same environment. This demonstrates that the optimised gas drainage reduced raw emissions from the mining operation.

At LW3, the daily fugitive emission (including methane and CO_2) was estimated at 2,847 t CO_2-e after incineration of drainage gas. During the trial at LW4, the estimated daily fugitive emission was 1,527 t CO_2-e. Based on these numbers, it is estimated that an annual reduction of 0.42 Mt CO2-e (assuming 315 days of drainage operation) can be achieved at the Blakefield South mine by adopting the optimised drainage system.

APPLICATIONS AND CONTINUOUS IMPROVEMENT

Following the successful trial at LW4, the Blakefield South mine adopted underground horizontal boreholes at the entire area of LW5 where goaf drainage was anticipated to be necessary. The roof horizontal boreholes were placed at similar locations to those at LW4, with four to five lateral boreholes intersecting the goaf, as shown in Figure 12. However, no floor holes into the Glen Munro seam were implemented due to site constraints.

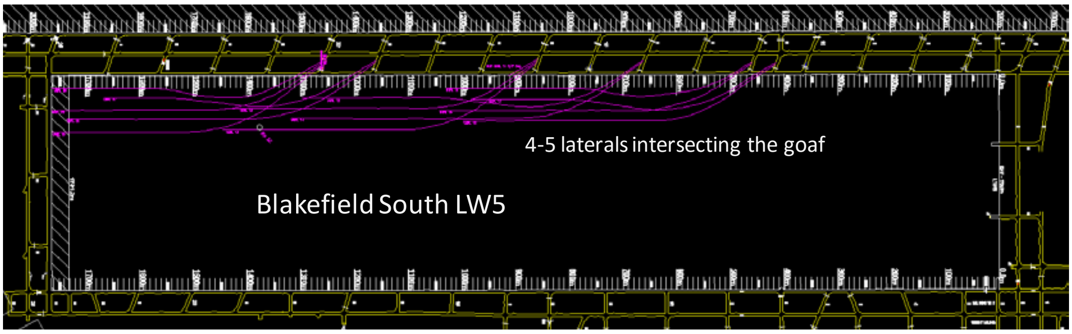


Figure 12 - LW5 goaf gas drainage borehole layout

Roof horizontal boreholes performed well at LW5 in capturing and controlling goaf gas, with their effectiveness similar to the trial at LW4. At the comparable initial mining stage, the average gas drainage flow rate and daily coal production at LW5 were 1252 l/s and 22,411 t, respectively, close to their counterparts at LW4 trial (1279 l/s and 23,121 t), and much better than that at LW3 where only vertical wells were used (1182 l/s and 13,806 t). A comparison between LW3, LW4 trial and LW5 are shown in Table 1. The performance data demonstrate that the roof horizontal boreholes were reliable and effective.

The gas flow rate in the return at LW5 averaged 1920 l/s, about 750 l/s greater than the LW4 trial. This is analysed because there were no floor holes implemented at LW5 and therefore gas released from the Glen Munro seam had to flow up to the goaf and eventually appeared in the ventilation return. This suggested that, where available, floor horizontal boreholes should be implemented to protect gas emissions from the floor of the Glen Munro seam.

Table 1 Comparison of gas drainage and emission parameters at recent longwalls at the Blakefield South mine

LW	Chainage meter, m	Gas drainage method	Daily LW tones, t	Total gas* emission, l/s	Drainage gas flow rate, l/s	Return gas flow rate, l/s	Relative gas emissions, m^3/t
LW3	35-223	Vertical	13,860	4004	1182	2822	25.7
LW4, Trial	30-259	Lateral (roof +floor)	23,121	2869	1702 (roof 1279, floor 423)	1167	10.7
LW5	30-220	Lateral (floor only)	22,411	3171	1252	1920	12.2

*Note: Gas include CO2+CH4

The retrospective analysis enabled further refinement for gas drainage design for future wide applications, and have contributed to the design for the forthcoming LW7. Gas emission rate at LW7 was predicted to be much higher than LW4 and LW5. Design for the panel included both roof and floor horizontal boreholes. In addition, a number of floor horizontal boreholes into the lower Blakefield seam were also designed in an attempt to prevent gas emissions from this seam. Furthermore, a surface borehole piezometer is installed to monitor seam pressure response for assessment of gas sources and emission characteristics.

It should be noted that, as mining and gas conditions vary from site to site, there is no unique drainage design that will work well at all sites. The integrated approach developed by this project provides a scientific methodology and practical procedures to develop and optimise the drainage design according to specific site conditions.

CONCLUSIONS

Under the auspices of Australia Government Coal Mining Abatement Technology Support Package (CMATSP), a major collaboration research project between CSIRO and the Glencore Bulga Underground Operations was successfully carried out at the Blakefield South mine.

An integrated approach of optimal gas drainage design was developed, which consists of six distinctive components: site characterisation, field investigations, coupled numerical modelling, CFD simulations and optimisation, design and trials, and retrospective analysis and continuous improvement. Following this approach, comprehensive studies were conducted at the mine, and many insights into coupled strata, gas and groundwater behaviours in complex multi-seam longwall mining environments were successfully obtained.

An optimal gas drainage system was successfully designed and trialled at the mine. The optimal system adopts horizontal boreholes into both the roof strata and floor coal seam in place of the mine's conventional surface vertical wells. In the trial period, a significant reduction in both ventilation methane levels and drainage gas volumes was achieved, and gas drainage efficiency was significantly improved to about 80% from 14-60% in the counterpart of the previous longwall. As a result, gas related coal production delays were substantially reduced and coal production was increased by 79% in the trial period. It is estimated that with such a gas drainage system, the mine could reduce fugitive gas emissions from its longwall operations by 0.42 Mt CO2-e per year.

The mine is now using this optimal gas drainage system as the main method for longwall gas drainage.

ACKNOWLEDGEMENTS

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BUILDING 3D ROCK MASS STRENGTH MODELS USING HARDNESS INDEX AND INDICATOR KRIGING

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BUILDING 3D ROCK MASS STRENGTH MODELS USING HARDNESS INDEX AND INDICATOR KRIGING

ABSTRACT

Geomechanical 3D models are not currently used in most mines particularly in Brazil. The absence of rock mass quality information affects mining design specially the ones more sensitive to the uncertainty associated with geotechnical parameters. The geomechanical models absence is mainly due to the reduced amount of geotechnical data and the consequent difficulty to interpolate and discretize them into a model. To overcome this data shortage, this study proposes building 3D geomechanical models using indirect rock hardness classes obtained from drill hole cores tactile-visual description. These data were grouped into seven rock hardness categories and spatially modeled via categorical indicator kriging. Each category was associated with a range of uniaxial compressive strength possible values. The combination of estimated category probability and range value for strength allows to assign each block to an average (E-type) uniaxial compressive strength. Furthermore, numerical values can be estimated by averaging these categorical histograms (class probability vs. center of the class interval) to fill the block model with compressive resistance values. This paper also presents the dynamic anisotropic interpolation scheme that provides a better modeling process in folded deposits. This method consist in modelling a variogram by adding dip direction information to each rock type, allowing the search ellipsoid to rotate adequately at each position within the domain. The final model was checked using a regional model where uniaxial compressive strength lab tests were conducted. The methodology proved to be efficient and was illustrated at a major iron ore deposit in Brazil.

KEYWORDS

rock mass strength, block model, uniaxial compressive strength, Indicator Kriging.

INTRODUCTION

The growing need of complex data base construction to subsidize geotechnical studies in mining has motivated the search of techniques for using all available data specially indirect information in areas with low coverage of lab tests. Additionally, the geotechnics use methods of slope stability analysis that require a detailed level of geotechnical knowledge, quality of the information and considerable volume of geological and geotechnical information.

Conversely, the amount of information and investment in geomechanical data acquisition that subsidize the geological/geomechanical models are scarce in most Brazilian open pit mines. That explains the reason why the determination of the rock mass quality parameters is generally obtained from tactile-visual observations in borehole cores and pit mapping or using tabulated values for the geomechanical parameters derived from rock mass classification systems.

Currently, the data gathered by geomechanical description are obtained via interpretation of two-dimensional cross-sections (2D) relying on the experience of the modeler, with little support of computational tools to help in the interpretations. This can cause inconsistencies due to changes in the geomodelling staff and lack of strict description rules along the process of building the database or the final interpretation of the cross-sections, for instance. These cross-sections are an important input for the geomechanical stability analysis using specific software's based on equilibrium limit or numerical analysis in 2D or 3D.

Based on this scenario, the use of three-dimensional blocks model (3D) with geomechanical parameters kriged is being broadly discussed in the scientific community (FOLLE, 2000 and

GUTIERREZ, 2009). These models may provide the values needed for the geomechanical parameters from the tactile-visual description along the entire block model using geostatistical methods for this spatial interpolation.

These rock mass georeferenced data, if properly 3D modelled can be used to improve mine pit final geometry design, making it better, safer and with lower loss of mineral resources. In this context this study discusses the application of kriging in modeling the consistence degree from ferriferous formation and its associated rock mass ratings, classifying the blocks according to a direct relationship with the uniaxial compression strength distribution proposed by BROWN, 1981. Additionally, the model proposed builds are every an estimated probability distribution for hardness values and using their relationship with the uniaxial compression strength assign to each block its strength.

ILUSTRATION STUDY

The methodology developed for this study follows the same workflow used in a geological model. It starts by assembling and preparing the database, exploratory data analysis, modelling, classification of the estimated blocks and results validation.

i. Stage 1: Geostatistical modeling of the hardness index using dynamic anisotropic concepts: application of kriging to model the hardness index based on the concepts embedded in the so-called dynamic anisotropy.
ii. Stage 2: Blocks classification

Classification of the kriged blocks considering the categorical hardness index variation and determination of the uniaxial strength by taking the weighted average of the uniaxial strength histogram (E-type) according to ISRM, 1981.

Determination of the variable Hardness index

One of the most important parameters to define the basis of a rock mass resistance is the intact rock strength, which is also called hardness index. This parameter is usually read at tactile-visual description, which associates the parameter to estimated values of uniaxial compression strength (UCS), point load tests or Schmidt Hammer readings.

For obtaining the hardness index some tests are conducted. These tests including geological hammer blows, finger pressure, and shear force applied using the hands, and saturation tests sinking the sample in the water. The data from these field tests are compared against uniaxial compressive strength.

Table 2.1: Description of the hardness index based on the table modified by ISRM (International Society of

GC - HARDNESS INDEX

CODE	GRADE	DESCRIPTION
0	EXTREMELY SOFT	Penetrable by the thumb. Crumbles easily under pressure from fingers and dissolves completely when stirred in water. R_0 (T_c 0,25 a 1 Mpa)
1	SOFT	Penetrável per slide disintegrating manually. Crumbles the hammer blow. R_1 (T_c 1 a 5 Mpa)
2	MEDIUM SOFT	Penetrable per slide; shatters the hammer blow. The fragment of the edges can be broken by finger pressure. R_2 (T_c 5 a 25 Mpa)
3	MEDIUM	Breaking with relative ease the hammer blow, the edges of the fragment can not be broken by finger pressure. The steel blade causes grooves on the surface. R_3 (T_c 25 a 50 Mpa)
4	MEDIUM HARD	The steel blade hardly causes grooves on the surface. Break up the hammer blow. R_4 (T_c 50 a 100 Mpa)
5	HARD	Break up with several hammer blows. No scratch-resistant steel for the blade. R_5 (T_c 100 a 250 Mpa)
6	EXTREMELY HARD	High resistance and are virtually impenetrable by steel blades. The fragments have rough or sharp edges. Break up with difficulty hammer blow. R_6 (T_c >250 Mpa)

Rock Mechanics, 1974-2006) BROWN, 1983 Modified by Vale in 2001.

These characteristics are then correlated to the uniaxial compression test results. The hardness index after indicator kriging interpolation throughout the model provides a distribution of the uniaxial compression intervals.

Hardness Index estimate method (Indicator kriging)

The visual or described geomechanical parameters are qualitative variables or known as categorical. Among the methodologies for estimating categorical parameters, include kriging of the indicators (Journel 1982, 1983).

According to JOURNEL, 1982 indicator kriging is weighted linear combination of probabilities, which allows us to obtain the local and global probability density function. According to CHILES and DELFINER (2012) kriging is a mathematical method used to estimate a variable Z in any position or its average value in any domain. This method is included in the nonlinear geostatistical methods where each parameter is codified into only two values: 0 (probability of not belonging to the class) or 1 (probability of belonging to the class).

As a result, at the end of the kriging process, each block has a probability value assigned in the 0 to 1 interval, which is equivalent to the distribution of 0 to 100% of chance to belong to any one of these classes.

Table 2.2: Proportion of each class in the original database.

Classes of Hardness index	Samples	%
0	5188	19%
1	8180	30%
2	3624	13%
3	4776	17%
4	1587	6%
5	2525	9%
6	1479	5%
Total	27359	100%

In order to consider a possible grouping for the variografic study of these multiple classes, it was assessed the expected average strength for each category in the original database. Considering that 60% of the dataset is sitting in the lower strength intervals, , and even that individual variograms hold the same spatial continuity, it was grouped class 0 to 2 interval for inconsistent materials and 3 to 6 for the consistent materials.

After this definition, it was constructed a dataset comprised by two large groups, the first one included the lower strength materials, called inconsistent, while the second group the higher strength materials, named consistent (Figure 2.1).

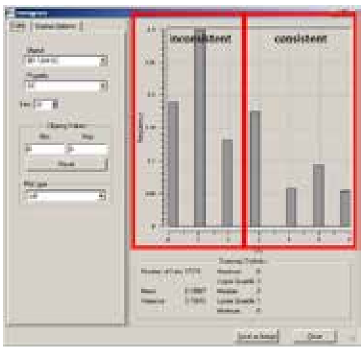


Figure 2.1: Histogram of the proportion hardness index classes.

The proportion of GC 0, 1, 2 represents 62% of the total data; whilst G 3, 4, 45 and 6 totals 38%.

Initially it was calculated the experimental omnidirectional indicator semi-variogram (figure 2.2) and next the directional ones.

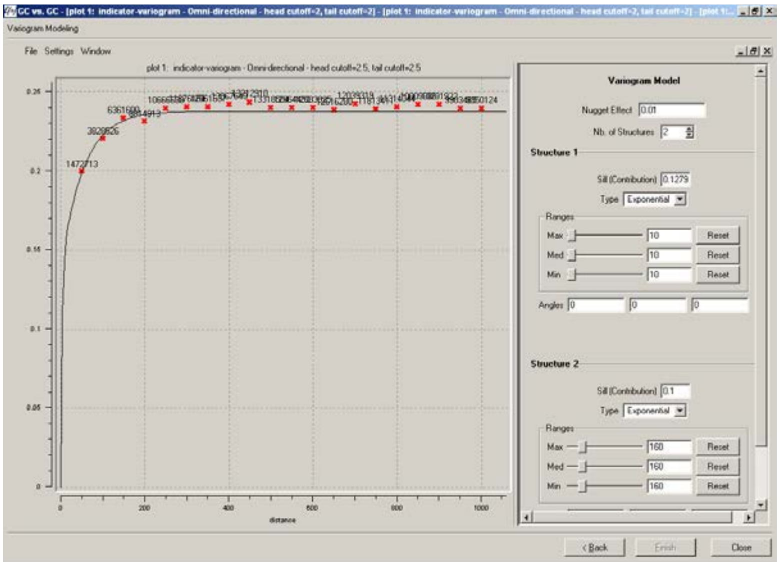


Figure 2.2: Experimental (dots) and model (continuous line) omnidirectional indicator semi-variogram.

The model adjusted is.

1) Exponential Model:

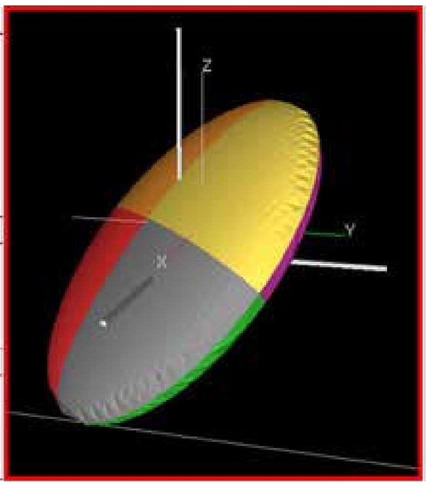
$$\gamma(h) = c^{*}Exp\left[\frac{h}{a}\right] = c^{*}\left[-\frac{h}{a}\right]$$

where nugget: 0.010, Sill 1: 0.127, Sill 2: 1

The major axis of anisotropy is aligned along N 135º at 45º dip.

Aiming at an adequate experimental semi-variograms adjustment, the use of two structures in the exponential model was necessary. The first one is the nugget effect nested with two exponential structures. The variogram model and its associated ellipsoid modelling hardness index spatial continuity along the major direction can be observed in figure 2.3.

Variographic Model	
Azimuth	135
Dip	45
Plange	0
Range 01	232
Range 02	144
Range 03	128
Nugget	0,01
1º structure	
Sill	0,12
Range 01	70
Range 02	50
Range 03	25
2º structure	
Sill	0,1
Range 01	290
Range 02	180
Range 03	160



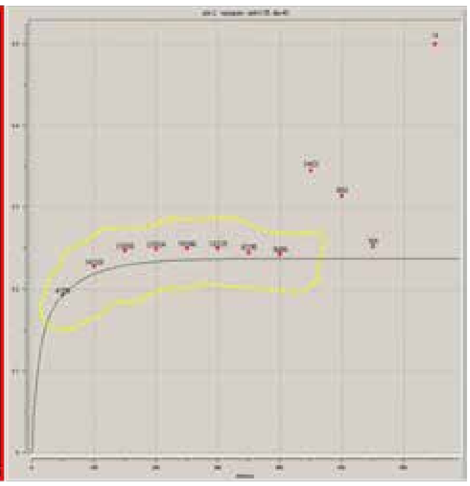


Figure 2.3: Variogram model used in kriging the hardness index categories.

For building a geotechnical model a grid was generated using the same origin coordinates and the same block dimensions as used for the geological block model (figure 2.4 and table 2.3).

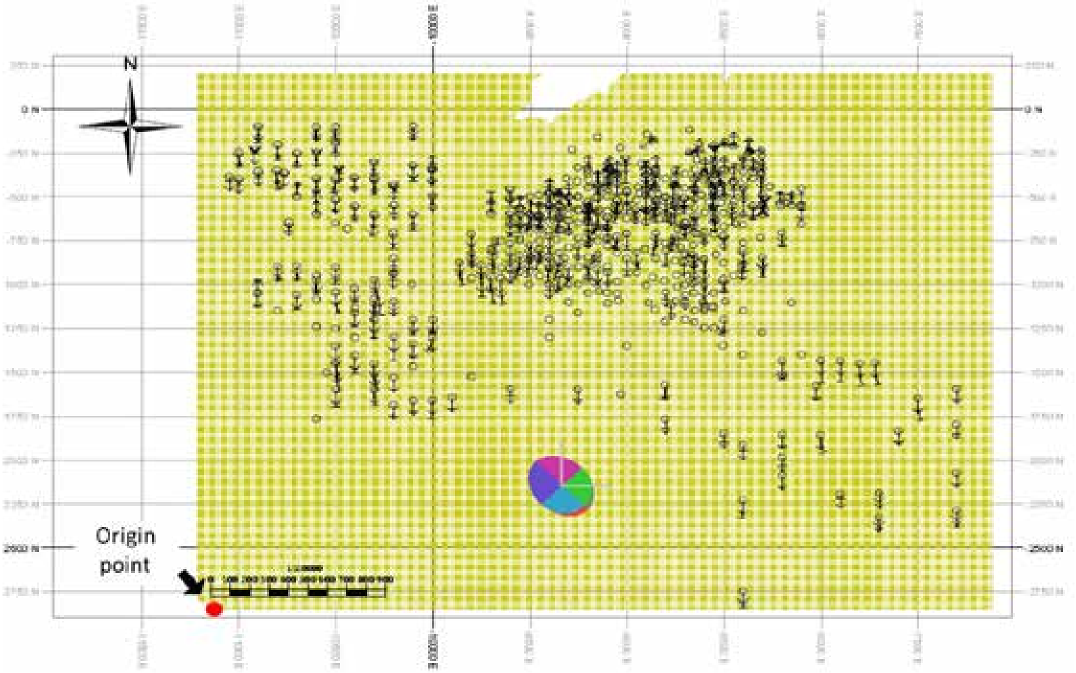


Figure 2.4 Location map showing the samples, the grid and an ellipsoid representing the search strategy.

Table 2.3: Grid details used for kriging.

Grid size					
X origin	**Y origin**	**Z origin**	**Nx**	**Ny**	**Nz**
-11125	-2850	700	430	350	100

The estimation was made considering one variogram for all seven classes. The model generated comprises 3.421.060 blocks with hardness index blocks distributed at the seven categories (table 2.4).

Table 2.4: Proportion of estimated blocks per hardness index class.

Hardness index	Block number	%
0	440923	13%
1	1169532	34%
2	647184	19%
3	388700	11%
4	396900	12%
5	315850	9%
6	61971	2%
Total	3421060	100%

Blocks Classification

The rock mass strength from the RMR classification is determined using multiple information including the uniaxial compression strength values. For this study UCS was obtained indirectly through the tactile-visual description of the rock core characteristics which define the categories and variations.

Each hardness class or category was assume to follow a normal distribution of possible UCS values with a mean and constrained by min and max limits which were determined through the consolidation of a large number of tests in several rock types throughout the world (table 2.5).

Table 2.5: Uniaxial compressive strength ranges for each hardness index class proposed by ULUSAY and HUDSON (2007) (Modified by Vale 2015).

Hardness index	Description	Resistance (ISRM) - Mpa		
		Minimum	**Average**	Maximun
0	EXTREMELY SOFT	0.25	**0.625**	1
1	SOFT	1	**3**	5
2	MEDIUM SOFT	5	**15**	25
3	MEDIUM	25	**37.5**	50
4	MEDIUM HARD	50	**75**	100
5	HARD	100	**175**	250
6	EXTREMELY HARD	250	**325**	400

The blocks classification was tested in two different ways, one assigning to the block the class estimated with the highest probability and taking the average UCS for this selected class. The other uses the probability to belong to each class (obtained by IK) to calculate the E-type or weighted UCS average illustrated in table 2.6.

Table 2.6: UCS weighted average for a given block knowing each class probability and average resistance per class.

	0	**1**	**2**	**3**	**4**	**5**	**6**	**Resistance**
probability %	-	-	10	15	20	55	-	-
-	-	-	(0.10 * **15**)	(0.15 * **37.5**)	(0.20 * **75**)	(0.55 * **175**)	-	-
-	-	-	1,50	5,63	15,00	96,25	sum =	**118.38**

It was possible to observe that the weighted average UCS leads to a smooth block model forming a progressive passage between categories, what is normally observed at the contacts (Figure 2.5).

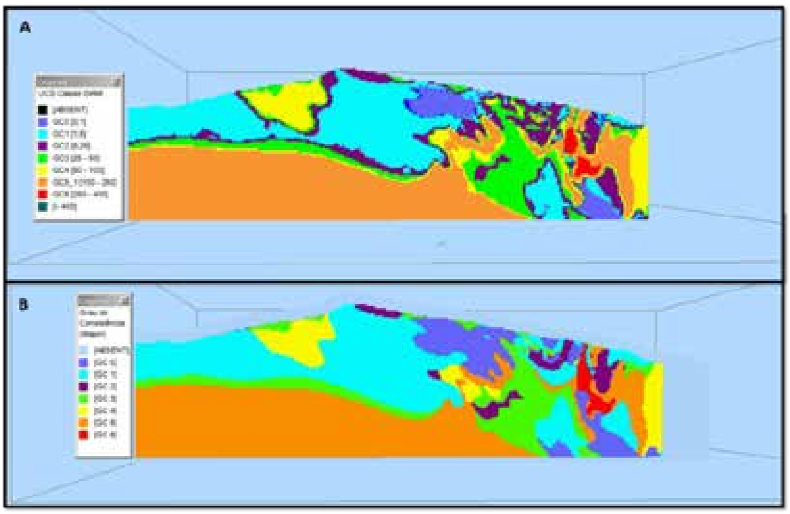


Figure 2.5: A. Vertical section example of the hardness index model, using the weighted average uniaxial resistance B. Section of the hardness index classified as the center of the class with the highest probability.

Indicator kriging provide a smooth model avoiding sharp edges along the contacts (figure 2.6). The model derived by choosing the most likely class (the mode from each IK local categorical histogram) is smoother if compared against the model obtained by weighted average UCS for all possible classes (E-type).

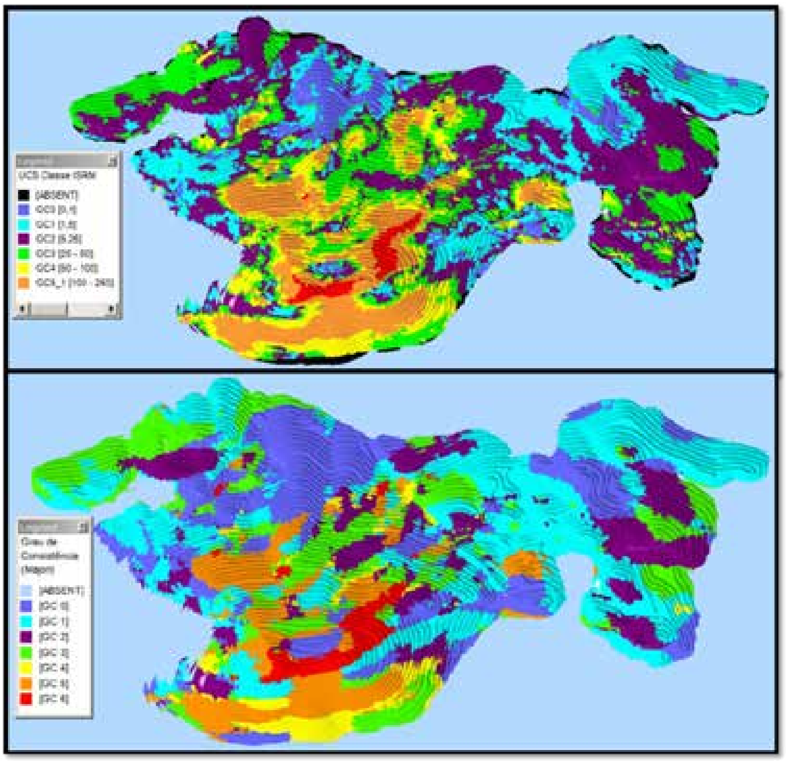


Figure 2.6: A. hardness index distribution in the final pit rated using the weighted average uniaxial resistance (E-type) B. Section of the hardness index classified by choosing the average UCS for the most likely class.

The blocks classification using the expected values(weighted average, i.e. probability versus the central value of each class), provides a more realistic model. This model is more in accordance with rock mass alteration, where the contacts are gradual.

If considered that, geologically, the banded rocks have a strength graduation due to its own characteristics, constitution, genesis and evolution of its mineralogical composition, as well as the rock alteration in its evolution process, this phenomenon is possible to be observed and may be certified in boreholes characterization in future drillings.

CONCLUSION AND CONSIDERATIONS

Basically, the objective was investigating ways to use the information description of geomechanical parameters, starting with the most relevant information about the resistance of the intact rock.

In case the parameter cited of this study was to investigate the application of indicator kriging combined with dynamic anisotropy for modelling geomechanical parameters. During the selection of which parameter should be tested and modeled, it was decided the hardness index was relevant as the construction of a 3D database is fast and leads to the rock mass quality.

The categorical variable distribution when transformed into a numerical rock mass resistance proved to be an effective solution to solve the modelling problem of mapping the spatial distribution of the intact rock strength along the deposit.

These estimates can be used to forecast the rock mass strength and used for predicting milling performance, equipment wear or a better definition of final pit increasing ore recovery. Most important the geotechnical model improves risk assessment associating weak zones mapped during kinematics analysis with low resistance areas. Additionally mapping high resistance zones leads to a slope angles optimization probably increasing ore recovery.

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CATALYSIS FOR BIODIESEL PRODUCTION USING HETEROGENEOUS REJECT MAGNETIC MINING PHOSPHATE AS A CATALYST

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CATALYSIS FOR BIODIESEL PRODUCTION USING HETEROGENEOUS REJECT MAGNETIC MINING PHOSPHATE AS A CATALYST

ABSTRACT

From the economical, industrial and environmental but also from the energy policy viewpoints, the biodiesel has been highlighted as being a remarkable alternative on the replacement or as complement to the mineral diesel as energy source. The biodiesel is industrially obtained through the transesterification reaction of triacylglicerols in bio-oils with alcohol of short molecular chain, to produce the corresponding esters. The reaction is usually catalyzed with strong basic catalysts in homogeneous medium. However, such a homogeneous chemical process has some disadvantages, as it leads to alkaline carboxylates (soaps) formation, and generates a significant volume of industrial effluents. Cleaning the final product to be put for the commercial distribution represents additional costs. An alternative way to get a cleaner chemical industrial process to produce biodiesel under lower costs is to use solid catalysts, which can be more easily removed from the reaction medium, chemically regenerated and cyclically reused. The mineral waste generated in mining may be and usually are a major environmental and economic problem. The technological destination of such mining reject is an object and a challenge to the scientific research. This report describes a work on the production of biodiesel through transesterification of soybean oil with methanol, by using the magnetic fraction from the mining reject generated by the Vale Fertilizantes S.A. as heterogeneous catalyst. The solid catalyst was obtained by mixing the magnetite from the mining reject with calcium oxide. The solid mixture was calcined at 1000 °C for 4 h in a muffle furnace. The crystallographic and hyperfine structures of the resulting catalyst were assessed by X-ray diffraction and ^{57}Fe Mössbauer spectroscopy. The transesterification reaction was performed in a batch reactor at 65 °C, in a mass ratio methanol:oil of 5: 1. The kinetic profile is sigmoidal in shape. The resulting chemical yield of the reaction was 82.46% in 5 h reaction. This result evidences the real possibility of using the magnetic mining waste in the biodiesel production. Further optimization of the chemical process and the subsequent reuse of the solid catalyst in the transesterification reaction are under progress.

KEYWORDS

biodiesel, transesterification, heterogeneous catalysis, Magnetite, reject, Phosphate Mining.

INTRODUCTION

The magnetic tailings from phosphate mining in Brazil contain a significant proportion of magnetite. The magnetic fraction of that mining reject was tested as heterogeneous catalyst for the transesterification reactions of triacylglicerols in bio-oils with alcohols of short molecular chains, most commonly, methanol or ethanol. The heterogeneous catalyst is thought to lead to cleaner industrial chemical processes, as the solid material can be removed and subsequently reused. From the economical and environmental points of view, the magnetic reject is largely available in the mining area, representing itself a threatening to the environment. This report describes the research work devoted to the use of the magnetic material from the mineral tailing from the phosphate mining in the Tapira Mining Complex, state of Minas Gerais, Brazil, as catalyst in the biodiesel (a mixture of fatty acid esters, in this case, with methanol) production. For being currently more used in the industrial context, it is adopted, in this work, the term carbonatite rock, meaning the dome intrusion of phosphate minerals (Borges et. al., 2008), instead of alkali carbonate, which would be more in accordance with the chemical nomenclature.

EXPERIMENTAL

The raw source of triacylglycerides was the commercial soybean oil. The magnetite separated from the mining reject material and calcium oxide PA (Aldrich) were mixed at a mass ratio 1:1, in a porcelain crucible. This mixture was mixed by hand, to be calcined in a muffle at 1000°C for 4 h. The produced catalyst was stored in a desiccator until being used in transesterification reactions.
A preliminary mineralogical characterization of the catalyst sample was performed by X-ray diffractometry on a diffractometer Shimadzu model XRD-6000, with CuKα radiaton. The scanning rate was set at 1º2θ min^{-1}, between 10º2θ to 80º2θ.

The iron chemical states were assessed by Mössbauer spectroscopy, making it also possible to infer about the proportions of the iron minerals (more specifically, magnetite, hematite or ilmenite) in the sample.

The triacylglicerols conversion to fatty acid methyl esters was obtained by using methanol: soybean oil in various ratios. The catalyst used corresponded to 4% by weight relatively to the oil. The reaction was monitored by thin layer chromatography (TLC) by using ethyl acetate:hexane at the ratio 5: 1 as eluant and iodine as developer on the plates. After completion of the reaction, the catalyst was separated and reused in subsequent transesesterification reactions. The reacting liquid was rotaevaporated and the methanol was recovered. The fraction containing glycerin and biodiesel was placed in a dropping funnel. The glycerol was removed, the volume of the biodiesel was measured and was subsequently washed with distilled water.

RESULTS AND DISCUSSION

Powder X-ray diffraction

The X-ray diffraction analysis revealed the occurrence of magnetite (Fe_3O_4), hematite (αFe_2O_3), and ilmenite ($FeTiO_3$). The XRD pattern of this sample was numerically fitted through the Rietveld method. In Figure 1, it is shown the FullProf®-generated theoretical pattern. The numerical method allows refining crystalline structures from the powder X-ray pattern. The difference between the two, theoretical and experimental patterns, is minimized through an iterative the least squares convergence (PARTITI, 2005). From this fitting, it is possible to obtain information about the crystal structure of the material, and the proportions of crystallographic phases in the sample (WEIDLER et.al.; 1988).

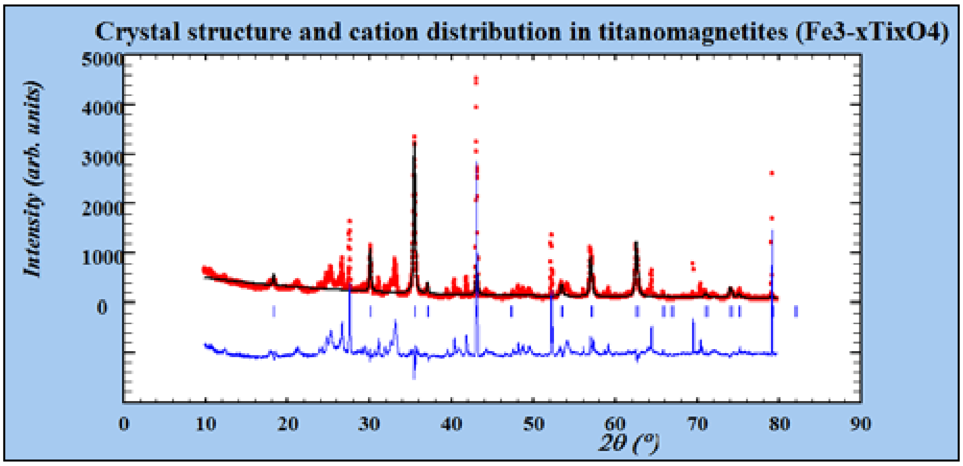

Figure 1: Refinement Rietveld XRD pattern of the sample of magnetite (FullProf®).

X-ray fluorescence analysis

The X-ray fluorescence data reveal that the magnetic material is relatively rich in phosphorus and calcium, along with silicon, iron and titanium.

Mössbauer spectroscopy

Numerically fitting the Mössbauer spectrum obtained at 298 K (Figure 2; fitted parameters are presented in Table 1) allowed quantifying the relative subspectral areas relative to the iron-bearing mineral phases, which were found to be: magnetite, 62%; hematite, 27%. The spectrum provided also information on some chemical characteristics of the material: characteristic patterns of (i) Fe^{2+} and (ii) mixed valence $Fe^{3+/2+}$ and (iii) a Fe^{2+} doublet. The Figure 2 shows the 298 K-Mössbauer spectrum of the sample of magnetite Tapira (sample CMTMGOX).

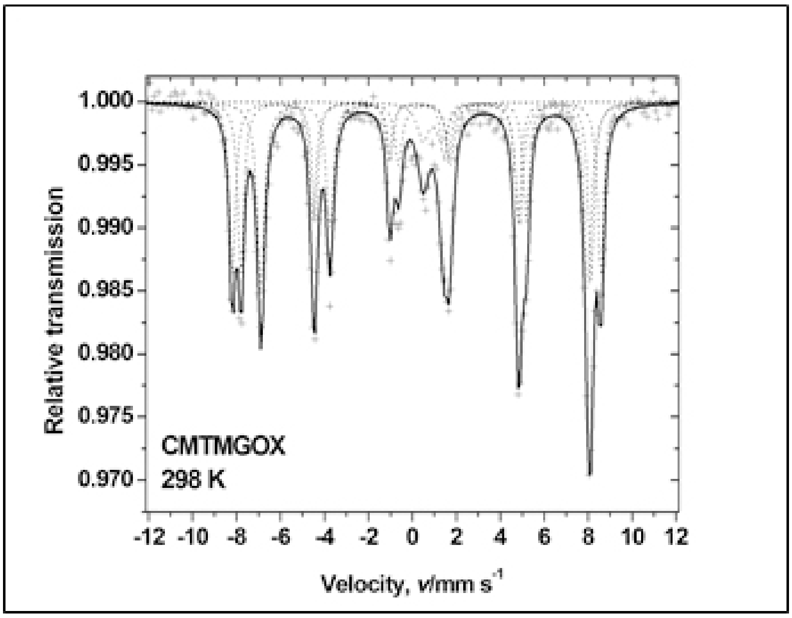

Figure 2 - Mössbauer spectra collected with the sample at room temperature (~298 K) of the Tapira sample containing magnetite.

From the hyperfine parameters (Table 1), magnetite (relative subspectral area, *RA* = 24.6 + 40.3 = 64.9%), with hyperfine field, [B_{hf}] = 49.42(3) tesla and {B_{hf}} = 46.17(2) tesla and hematite (*RA* = 25.4%), with B_{hf} = 51.98(3) were identified.

Table 1 - Mössbauer parameters of the CEMS spectrum for the archaeological rupestrian pigments.

Sample	^{57}Fe Site		δ/mm s^{-1}	□/mm s^{-1}	2ε, /□mm s^{-1}	B_{hf}/T	*RA/%*
CMTMG	α-Fe_2O_3		0.367(3)	0.33(1)	-0.137(7)	51.98(3)	25.4
	Fe_3O_4	[Fe^{3+}]	0.287(4)	0.33(1)	-0.017(8)	49.42(3)	24.6
		{$Fe^{3+/2+}$}	0.660(3)	0.41(1)	-0.003(6)	46.17(2)	40.3
	Fe^{2+}		1.04(1)	0.64(4)	0.88(2)		9.7

Fitted Mössbauer parameters of the CEMS spectrum for the archaeological rupestrian pigments, at ~298 K. δ = isomer shift relative to the αFe; *Γ* = resonance linewidth; *Δ* = quadrupole splitting; 2ε = quadrupole shift; B_{hf} = hyperfine magnetic field; *RA* = relative subspectral area. [Fe^{3+}] and {$Fe^{3+/2+}$} stand, respectively, for iron in tetrahedral and octahedral coordination sites of the spinel structure of the magnetite. Number in parentheses are standard deviation over the last significant digit of the value, as output by the computer fitting program, based on the least squares algorithm.

Catalytic transesterification reaction

The heterogeneous catalyst presents important advantages over the homogenous alkaline catalyst: easier separation and simpler purification process. There is no need to wash the solid catalyst to neutralize the product, and this reduces the amount of wastewater as industrial effluent. Moreover, the heterogeneous solid catalyst can be removed form the reaction medium to be subsequently reused. These are the main reasons why to develop new heterogeneous catalysts are in focus for the production of biodiesel. The choice to use magnetite as heterogeneous catalyst had also the meaning to ease the separation of the solid material from the reaction medium with a magnetic field, by the end of the chemical process.

The reaction time with the catalyst was 300 min, and the stoichiometric yield was 82.46%. According to these results only, the magnetic material imparted a significant chemical catalytic activity on the transesterification reaction. The mechanisms and interactions involved in the synthesis of magnetite catalyst and calcium oxide have not been fully elucidated and become essential topic for further studies.

Changzi and coworkers (2014) have developed their work in a nanocomposite core Fe_3O_4 or Fe_2O_3 coated with a silica shell. As a result, a high-stability material was obtained (acid resistance) and a large surface area, which is especially suitable for use as catalyst supports (MACEDO, 2013).

The magnetite-CaO catalyst in relation to performance and reaction time, showed significant results in relation to similar studies in the literature. In work published by Hu et. al. (2011) used the catalyst KF – CaO - Fe_3O_4 in a proportion of 4% w/w, relative to the oil in transesterification reactions of 3 h, with a yield of 95%. better results to those obtained with the magnetite-CaO catalyst developed in this work, which may be related to the percentage of catalyst used in the process, or the fact that this material is of natural origin, arising from mining tailings.

CONCLUSIONS

These results point to an effective catalytic effect of the magnetic reject from the phosphate mining in Tapira, Minas Gerais, Brazil, in the transesterification reaction of triacylglicerols in the soybean oil with methanol, to produce methyl esters of fatty acids (biodiesel). The reaction time of 300 min with the heterogeneous catalyst and the corresponding chemical yield of 82.46% make this material a strong candidate to gain still more chemical effectiveness on the transesterification and to be a potential material to be scaled for industrial chemical processes on the production of biodiesel. This would represent, from one side, an enormous environmental importance, for cleaning the tailing material from the mining area, and, for other, an economical gain by using the fully discarded material towards a technological advanced catalyst. Further developments on these magnetic materials are strongly envisaged and are currently in continuous progress by our scientific research Group.

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COMPUTER AIDED EDUCATION IN MINING ENGINEERING AT THE FEDERAL UNIVERSITY OF MINAS GERAIS – UFMG

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COMPUTER AIDED EDUCATION IN MINING ENGINEERING AT THE FEDERAL UNIVERSITY OF MINAS GERAIS - UFMG

ABSTRACT

In recent years, the Mining Engineering Department of the Federal University of Minas Gerais, DEMIN-UFMG, has invested substantially in the necessary infrastructure to allow the application of advanced computing technology to its undergraduate and graduate education and research programs. The notable areas involved are the mine planning and operations research laboratory, LPL, that serves the purpose of supporting the practical parts of specific biannual undergraduate courses such as the units of Mineral Research I, II and III, Open Pit Mine Planning and the laboratorial unit of Software Application for Mine Design and Planning. Through the utilization of academic versions of commercial leading software packages and modern hardware, the undergraduate students are able to apply and practice the knowledge obtained during the theoretical classes and to emulate real world scenarios. The development of advanced studies and algorithms is also promoted through projects that integrate the academia research with industry needs. The main fields of research interest currently in development inside DEMIN-UFMG are the Stochastic Block Model Simulation and the Direct Block Scheduling under Uncertainty. The effort the DEMIN-UFMG has put together in order to update its software and laboratory infrastructure is producing important positive results in terms of the success our recent undergraduate are obtaining in their careers, both in the industry and research centers or universities. The graduate work reflected in the new research that is currently in development is also innovative and has the potential to contribute to technological advances for the mineral sector as a whole.

KEYWORDS

Education, Mining Engineering, Modeling

INTRODUCTION

The development of new tools as an attempt to increase agility and ease the work of professionals in any field is a constant, in a daily basis evolution, leading to the professional update. Universities play a very important role on updating both professionals and students, and must be prepared to face this improvement process. In order to prepare the professionals and students, by improving teaching methodology at the Federal University of Minas Gerais (UFMG), the Mining Engineering Department (DEMIN) has decided to encourage the use of computational tools for students to solve real problems of the mining value chain. This seems to increase students' interest for the content being learnt, by facing real challenging situations, and also increases their ability in solving industrial issues. Recent technological advances over last decades have enabled the emergence of numerous computational tools which facilitated and modified the professional's routine work, mainly in mining engineering branch. Thus, it is unconceivable to think in engineering without the use of the great benefits provided by these new computational resources.

The demand for qualified professionals to work with these new tools has increased considerably in recent years. However, it is clear that in many professional training centers, including universities, the development usually did not accompany this growth in the use of computing resources, creating a gap between the recent graduated professionals and the current industry demand. The use of these new tools approaches the university to the industry, and this is fundamental to the development of modern practices and new technologies. Its application on the graduate courses allows the students to experience some of the job difficulties and to apply the theory gained from the academy at industrial situations. The mining engineering department of the Federal University of Minas Gerais (DEMIM-UFMG), realizing this new demand, is making a great effort to employ these new computing resources and the contact with the industry in the students formation. Several partnerships with mining related companies were sealed in order to provide the most updated existing tools for the students.

These tools are already being applied to some mining engineering disciplines at UFMG, and being combined with the theoretical background providing students with a more aligned academic training with market's demand. This may enable the graduating engineers to have a clearer picture of the activities that are developed in the industry. Moreover, this approximation of the technology companies with the university promotes the emergence of new research lines, due to the need of using specific tools in certain study areas. This initiative beneficiates both, the students who will be provided by a better market qualification at the time of their graduation the University itself and the industry through enabling the alignment of current demands with existing and possible emergence of new research lines.

OBJECTIVES

This paper highlights the benefits and encourages a stronger interaction between industry and research centers mentioning the evolution of the Mining Engineering Department of UFMG with the implementation of several partnerships with industry-focused companies. The possible benefits of the application of computational resources and real case scenarios are also be discussed, as well as some possible current limitations for achieving even better results. The study also introduces the current research focus currently going on at DEMIN-UFMG about Stochastic Mine Planning and Direct Block Scheduler (DBS) under Uncertainty, in a partnership with Vale Institute of Technology (ITV). Hence, the present work should represent a subjective access of the current teaching methodology identifying qualitatively the possible benefits and some of the main challenges.

UFMG AND DEMIN BRIEFLY HISTORY

The foundation of the School of Engineering of Minas Gerais is directly related to the birth of Belo Horizonte. The new capital of Minas Gerais, was built in 1897and was supported by the former capital, Ouro Preto, to develop educational institutions. On May 21, 1911, the Agriculture Society of Minas Gerais received a meeting that would set the road of the academic methodology to form engineers in Belo Horizonte, Minas Gerais. These pioneers, most of all engineers formed by the Ouro Preto School of Mines, wanted the New School to become one of the most important technical establishments of the country. The UFMG was funded and classes began on April 8, 1912, in Belo Horizonte.

Following the trend of other universities, the engineering curriculum has been based largely on a "science engineering" model, with a solid basis in science and mathematics, over the last five decades. The first two years of the curriculum, which has been evolving in many aspects since late 1950s, are devoted primarily to the basic sciences which serve as fundamental pillars for later two years of "science engineering" or "analysis" where students apply scientific principles to technological problems.

The industry and the academy identified the lack of experience to solve practical and technical challenges faced on engineering activities by the recently graduated engineers. The development of activities based on real problems assisted by computational resources was the alternative found to improve the classical methodology, a kind of problem-based learning. This learning methodology increased the level of academic challenge due the encouragement to achieve higher efforts by the students. Students tend to learn more when facing real problems and applying their knowledge to solve it. The methodology consists to teach the basis on science and mathematics before of the challenge on computational applied tools. The computational tools are the instrument to conduct the students to a real problem. The structure of the learning methodology applied can be observed on Figure 1.

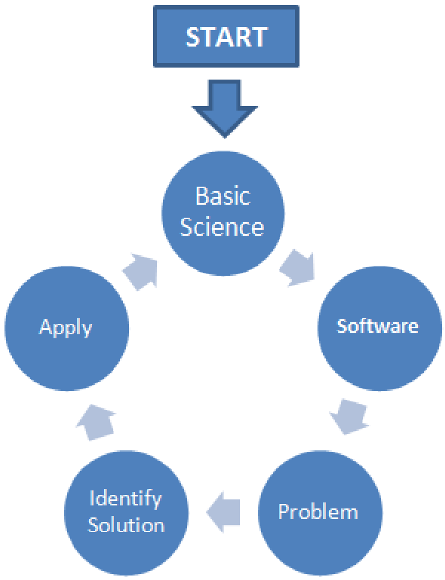

Figure 1 – Proposed learning structure methodology

This methodology has increased a group of skills needed to graduate capable engineers to solve industry's routine activities and develop the ability of both technically and theoretically solving different problems.

IMPLEMENTATION AT UFMG

The DEMIN professors, in an attempt to overcome the issues related to the lack of practical experience in students' formation, mainly on planning and production areas, and foreseeing the possible associated gains, attempted to invest efforts and capital on the evolution of the Mining Engineering Department. The recent investments on the Mineral Research Laboratory (LPL) include the purchase of new machines, which are now available for students and the acquisition of software in three different areas: mine planning, statistical/geostatistical and geospatial analysis. The main software related to the mining sector available in LPL are described in the Table 1 below:

Table 1 - Main software that are part of LPL collection.

Geospatial Analysis	Mine Planning	Statistics/Geostatistics
ArcGis	Delphos Open Pit Planner	Geo Visual
	Deswick	Stanford Geostatistical Modeling Software - SGeMS
	Micromine	
	MineSight	E(Z) Kriging - Ordinary Kriging
Envi	CAE Mining	
	Gurobi	Minitab
	Vulcan	
	SimSched	Isatis

It should be noticed that many of the licenses currently in use are products of partnerships with their commercial representatives, which is recognized and acknowledged by the Department.

Geospatial analysis has been explored on undergraduate and graduate courses in order to provide the students the opportunity to be in contact with commercially available tools for geoprocessing and providing base to learning about remote sensing and GIS (Geographic Information System).

The mine planning area registers the largest amount of available software, which reflects the large number of commercially available software. The variations in the software interface and functionality lead the industry to choose for the adequate software which best fit to their labor knowledge and technical challenges. This industry variation on the mine planning software allows students to learn various software both, at their industry experience (internships) or at the Software Application for Mine Design and Planning unit. Other consequence of the large number of available software is the increasingly number of students using it in their Final Year Thesis (TCC). This may be interpreted as a positive feedback of the new approach, reflecting the cycling relationship between the students interest growth and increasing use of software.

According to the new emerging mine planning methodologies, the DEMIN is currently developing a project on the Stochastic Block Model Simulation and Direct Block Scheduling under Uncertainties. The project is the idealization of an agreement between UFMG and ITV, and it aims to research new methodologies or recently-made feasible due to technological advances, and relate it to practical applicability. The project involves some detailed study on Lerchs and Grossmann (1964) algorithm and its further variations, and the comparison between these traditional methodologies to the new proposed methodology for sequencing and probabilistic assessing mine planning. The probabilistic approach as well as the new sequencing methodology, are common trends in the mineral sector, mainly due to the possibility of quantifying the associated risks and ensuring an increasingly adherence of the models to the reality.

In geostatistical area, in addition to traditional studies for geological modeling and grade estimation, the LPM is currently looking for exploring the simulation processes, either for grade estimation or for the determination of the lithotypes. The great importance of the geological model and orebody definition for the mining projects instigates the mentioned studies due to the constant technological and technical improvement scenario. The current challenge is not to obtain a geological orebody, but to predict the probability distribution and quantify its reliability, providing a useful tool for the decision making process. The theoretical fundamental for the application of the geostatistical tools is attempted to be provided by units such as Mineral Research I, II and III and Geostatistics.

THE ADVANTAGES OF A STRAIGHT RELATION BETWEEN INDUSTRY AND RESEARCH CENTERS

The recent technological and computing advances allowed the application of complex algorithms on different mining software. New approaches have been emerging through last decades and the stochastic mine planning is one interest area currently being developed through various research centers worldwide. This and many other methodologies applied in the mining industry require complex algorithms to solve mining problems faced by the industry. These industry challenges imply huge investments on developing tools with the objective to achieve higher and safer profits on the business. A very attractive and smart way of investing in such researches is the partnership with universities and research centers. Great advantages could be taken for both sides of this partnership. At the same time industry may improve the qualified labor available by improving the knowledge of students coming from the graduate and postgraduate courses, it may also achieve lower-cost investments, if compared to researching by their own. There are immeasurable gains of providing real data and real issues for students to practice before entering the industry and facing their own issues.

This relationship has proved to be beneficial with many examples around the world. Among the numerous examples there are the Australian Centre for Geomechanics (ACG), the Centre for Exploration Targeting and the Julius Kruttschnitt Mineral Research Centre (JKMRC) in Australia, the COSMO Stochastic Mine Planning Laboratory in Canada, the Delphos Mine Planning Laboratory and the Block Caving Laboratory in Chile, and others. The need for a stronger interaction between industry and academy has been in discussion recently and deserves attention. The research with industry's real data should encourage the production of scientific papers and the better labor preparation to overcome complex challenges. This need to be supported by the industry to become true, by providing some non-strategic data to who is interested in developing research projects and sharing the developed studies, which quite often does not happen.

The closer relationship between university and industry could also be beneficial by supplying data for new research lines and the deepening knowledge on existing ones. This should promote the scientific publications which could increase the university and courses' quality and thus contributing to technological development, which is beneficial for the whole society. The combination of all of the practices adopted and its benefits could lead to provide students with extra motivation, what should make this challenge each time easier to achieve increasingly beneficial results.

IDENTIFIED WAYS TO IMPROVE MINING ENGINEERS' FORMATION

During the process of implementation and even later after the implementation of the mentioned educational methodology changes, different factors or ways to improve the engineers' formation were identified. These could represent suggestions to the teaching methodology improvement, and do not necessarily were implemented yet at UFMG. Some identified methods for approaching students and industry may be based on the day-by-day practices at the university. The possibility of applying the theoretical knowledge on real case examples has shown many positive benefits and this is only possible with the closer relation of industry and academy. The software availability is another very relevant factor, and applied to real databases, may allow professor to provide practical examples on lectures, assignments or exams. Lectures, speeches and short courses, provided by industry professionals are also very interesting for the students once it brings them to unusual situations and this could help to retain their attention and get them involved. Field work is put in practice mostly in geology units, but it is other excellent way of improving students learning at mining units.

Conferences and symposiums are often held at UFMG, including the Louis-Ensch Mineral-Metallurgical Symposium, promoted yearly by the students council. Conferences and symposiums allow student to have contact with industry's updated challenges and to network with industry professionals. Another interesting stimulus for student's learning is the increasing popularity, at least in Brazil, of the SME Mining Games. The fun competition gets students excited with some different practical exams in which in many cases they have to apply the theoretical knowledge, such as on mineralogy and underground blasting. This may also increase the students networking with other students worldwide, who will probably become professional colleagues. This is especially important when considering the limited "mining world", in which partnerships have proved to be extremely beneficial for the parts involved and for technology development.

LIMITATIONS TO THE INCREASED IMPROVING

During the development of this study, some practical limitations have been identified as the main difficulties to the students' formation improvement. The data availability is many times limited by companies with the reason of being strategic data, and the lack of real database may lead to students' lack of interest, mainly due to the lack of applicability. The little communication between industry and universities brings the industry many times to search for solutions alone. Other great difficulty may be the lack of economic resources to afford the increased common prices practiced by mining industry.

CURRENT FOCUS AND FUTURE PLANS

As mentioned above, there is currently going on a research project at DEMIN-UFMG in partnership with ITV based on the Stochastic Mine Planning and Direct Block Scheduling methodology. The project is named Direct Block Scheduling under Uncertainties and it aims to analyze the probabilistic behavior of the economic value of the pit when subjected to different uncertainty conditions, such as on the block grade estimation or on the acceptable slope angles. The first focus of the project is to evaluate the adherence of short, medium and long term mine planning with the mine operation obtained data. The second one is to evaluate the impacts of the uncertainty associated to the ore grade and geological modeling on the final pit economic value. The final one is to evaluate the impacts of the slope angle variation on the economic value and also its stability, discussing the trade-off between the increase in economic's value and the increase in safety.

This topic is also focus on different research centers worldwide. The McGill's COSMO Stochastic Mine Planning Laboratory has many published studies in these areas, such as on Ramazan and Dimitrakopoulos (2013), Kizilkale and Dimitrakopoulos (2014), Montiel and Dimitrakopoulos (2015), Goodfellow and Dimitrakopoulos (2016), etc. Other important center with many recent publications in this field is the Delphos Mine Planning Laboratory with publications such as Alarcón, et al. (2015), Julio, et al. (2015), Vargas (2014), etc. In many cases, the simultaneous research from different centers result in partnerships such as on Marinho and Tipe (2015), Goycoolea, et al. (2015), Morales, et al. (2015), Beretta and Marinho (2014, 2015), etc.

The development of research lines in accordance with the industry demand allows for a continuous improving cycle for all the parts involved. The DEMIN-UFMG future plans involve going deeper on the DBS and Stochastic Mine Planning methodologies and also look for other interesting research lines which could fit this industry-academy requirements.

CONCLUSIONS

Since the 80's the technological and computational advances allowed the increasing applicability of computers to solve complex problems demanded by the industry. The creation of computerized tools enabled great advances on mining related areas, but it still does not fulfill industry needs. In order to follow these advances, universities play very important role on supporting the industry at developing new technologies. The DEMIN-UFMG has recently gone through an updating cycle in which capital and effort were, and still are being invested to result on a continuous improvement. The adopted approach is focused on the industry needs, thus applying real problems for the students get better prepared and also involved with current challenges and trends on mining industry.

During this continuous development some points were found to have great impact on the teaching methodology and on improvement of student preparation for the industry, rather than the conventional lectures, and this is now encouraged at DEMIN-UFMG:

- Invitation of industry labor to provide presentations and lectures;
- Acquisition of commonly used software in industry;
- Use of real data on units' lectures and assignments;
- Field work;
- Symposium and Conferences;
- Internships.

Among the limitations found to be relevant for the application of such methodology are:

- Little availability of real data;
- Lack of economic resources;
- Little or no contact with many or most companies;
- Lack of applicability of the theory learnt;
- Lack of students' motivation.

The industry approach to academy shows many benefits for all the involved parts, which are summarized below:

- Better qualifications of Engineers;
- New research lines;
- University researches usually will require less industry's expenditure;
- Improvement of the University and course quality;
- Development of technologies
- Motivated students which may ease all other steps.

The greater contact with the industry allowed the department to make an agreement with ITV to develop a research program based on Stochastic Mine Planning and Direct Block Scheduling under Uncertainties. The mine planning advances have shown detached performance recently and this probabilistic and more realistic assessment is being carried out by different centers worldwide.

Finally, the increasing proximity with the industry and alignment with its needs resulted in the current partnership and many benefits for all involved parts. The improvement of teaching quality lead to increase students' interest for the studies and also get them better prepared to face industry situations. This is beneficial for the companies, and could promote even more interaction. As it can be noticed, the cyclical aspect of the methodology promotes a continuous improvement of the teaching, learning and working skills of all parts involved.

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CONCEPTION OF AUTOMATED MANAGEMENT OF GEOTECHNOLOGICAL COMPLEX ON AN INNOVATIVE BASE

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CONCEPTION OF AUTOMATED MANAGEMENT OF GEOTECHNOLOGICAL COMPLEX ON AN INNOVATIVE BASE

ABSTRACT

This article outlines the basics of the methodological approach to the automated corporate management of geo-technological complex (ACSM GC–AKSU GK) and the method of its implementation. Management system is based on an objective, accurate and timely information, which is formed within an automated positioning system, monitoring and dispatching of mining and transport operations, and it is also acquired on options using simulation models of mining and transport processes.

The main unit of ACSM GC is an analytical unit, which includes analysis modules, planning, optimization and forecasting work of mining and transport complex taking into consideration specific geological, mining and geological, mining-geometrical, organizational and mining-economic conditions of its operation. The article gives the basic understanding and interpretation, ACSM GC building principles.

The ACSM GC principal difference from similar systems that are limited by dispatching is a complete basing on the economic effectiveness assessment of the mining and transport system functioning, which provides rapid feedback and evaluation of the response system to certain administrative decisions in online mode under mining and transport works planning and design.

Developing scientific and practical direction opens up entirely new possibilities in planning and mining and transport operations management at mining enterprises, providing an effective mechanism for modernization of mining and transportation complexes and management techniques on an innovative basis, as well as mining and transport operations cost saving and enterprises competitiveness improvement on the minerals markets.

KEYWORDS

Geotechnological complex, corporative management, monitoring, dispatching, optimization, planning, simulation modelling, economic, system, system approach.

INTRODUCTION

In a globalized economy conditions, leading to an inevitable leveling of resources cost, as well as steady increase of production costs of mining operations with reduction and the deterioration of mining and engineering and geological conditions of deposits exploitation, mining enterprises will feel the need for continuous competitive recovery.

Successful implementation of the existing potential for competitive recovery will require constant innovation implementation process at the enterprises, the development and adoption of effective solutions, while efficiency will require production technologies and production processes organization. In the short term, these pointed out factors will be the main drivers of sustainable development and of mining complex competitiveness.

At the present stage development of mining in the world the main factors of competitiveness of this branch enterprises, providing more than 60% of the total capacity are: the level of production technology (12%); the level of primary and supplementary equipment (10%); the level of integrated automated process management systems (18%); optimization degree of technological processes and operating conditions of the equipment (material and energy efficiency, product quality and its costs minimizing , 12%); the level of integrated organizational and technical automated management system of enterprise (10%) [*Assessment of the iron ore companies competitive potential - the basis of their development strategy formation in the globalization conditions, 2005*].

If the basic technological processes during the development of solid minerals deposits have not undergone any significant changes over the past 50 years but technically, one can observe significant changes in terms of increased scales, capacity and the main mining and transport equipment automation.

Management arrangement processes of mining and transport operations are undergoing significant changes. The increasing use of management systems automated options and labor accounting of the main technological equipment with dispatching elements or automated dispatching systems are used more and more.

However, in these conditions, the potential of productivity and efficiency improvement and mining cost and transport operations reduction are quite essential.

If, as shown in Table 1, accounting and control systems allow to reduce costs by 5-10% due to the decrease of non-target costs for the production of mining and transport operations, and dispatching elements make it possible to reduce costs by 5-10% due to increased productivity, as well as by reducing operational delays of the main equipment, the introduction of automated real-time monitoring capabilities of efficient monitoring, optimization and management modes and operating conditions of the equipment allows to reduce the cost additionally by 30% or more.

Table 1 - Mining and transport operations corporate management capacity assessment

<table>
<tr><th>Functions</th><th>Effect and conditions of its obtaining</th><th>Effect obtaining levels</th><th>Possibilities</th></tr>
<tr><td>Accounting</td><td rowspan="2">by 10-15% at a time</td><td rowspan="4">Monitoring and control system of mining and transport operations</td><td rowspan="4">Production discipline increase , basic technical and economic indexes monitoring</td></tr>
<tr><td>Control</td></tr>
<tr><td>Regulation</td><td rowspan="2">Promptly in the frames of the shift</td></tr>
<tr><td>Arrangement</td></tr>
<tr><td>Rationing</td><td rowspan="3">Under constant work up to 30% and more</td><td rowspan="5">Analytical block of ACSM GC</td><td rowspan="5">Main technological equipment operation modes and operating conditions optimization, works planning</td></tr>
<tr><td>Stimulation</td></tr>
<tr><td>Planning</td></tr>
<tr><td>Regulation</td><td rowspan="2">Within the frames of short, medium and long term periods</td></tr>
<tr><td>Arrangement</td></tr>
</table>

The present concept of automated management of geo-technological complex is continuation of mining enterprises sustainable socio-economic development general concept and is based on the application of the latest scientific and technological progress achievements and, in particular, in the field of information technologies [2].

BASIC APPROACHES AND PRINCIPLES OF SYSTEM DESIGN

Technical stability of the system is commonly understood as its ability to keep movement on the planned trajectory or maintain the intended operation mode, despite the factors affecting it. [*Mining enterprise social and economic development management, 2015*].

With regard to geo-technological complex, based on the specifics of the mineral resources market its sustainable development in the framework of the Concept is understood as ability to support the effective functioning mode.

Efficiency of technological complex mode functioning is due to the need to preserve and develop mining enterprise competitiveness and it involves the concepts acceptance of mining and transport operations optimal and appropriate cost aimed at minimum.

In the framework of the Concept ccompetitiveness is understood as the ability of a particular object or subject to meet the concerned parties demands in comparison to other similar subjects and/or objects. In this regard, in the management system of geo-technological complex factors and possibilities of the mining enterprise competitiveness ensuring are introduced.

Another fundamental idea that defines functional orientation of ACMS of GC is a corporate approach to management.

In this respect, ACSM GC Concept building and its development proceeds from the understanding that “Where there are no common interests, there can be no unity of purposes, saying nothing of the actions unity”, as it was fairly pointed out by F. Engels.

This statement is supported by scientifically valid conclusion that the subjects social and economic interests balance is a relevant factor of the company's development effectiveness - its growth by 1% gives efficiency growth by 1.86% [*Mining enterprise social and economic development management, 2015*].

Currently, enterprises economic efficiency systems require the use of a more functional approach, where performance is evaluated in terms of indicators characterizing, in isolation from the overall manufacturing process, the departments and divisions functionality.

The implementation of a systematic approach to the consideration of the object implies an integrated, process-based approach when cross-functional communication, the relationship of subsystems and elements of such complex and multifactorial system as geo-technological complex are taken into account.

As the object of the automated management system geo-technological complex is selected on base of the basic definition, the idea of "geo-technology", which is treated as science that studies the subsoil development methods and processes, creating theoretical bases and engineering solutions of an efficient, economically and environmentally viable deposits development, construction and operation of mining and other underground facilities, as well as industrial buildings in various engineering and geological conditions.

Here the subject matters for consideration are mining and geological and mining conditions and solid mineral deposits characteristics, opening ways and methods to geo-resources access; study and optimization of physical, technical, and physical and chemical technologies parameters; creation and scientific substantiation of natural and man-made solid minerals deposits;

development of technological methods of mining enterprise quality control and methods to increase the completeness of subsoil reserves extraction; development and scientific substantiation of criteria and technical requirements for new mining machinery and equipment establishment; interaction processes of engineering structures and rock massifs and stability of mine workings; scientific substantiation of mine engineering facilities parameters and the development of their calculation methods.

Scientific field of "Geo-technology" includes three main areas of researches. They are underground, open and construction geo-technologies.

Geo-technological complex includes such an active part as mining and transport complex, an adequate recording of its interaction with the environment -mining, mining and geological environment, mining and technical conditions, etc., is essential for effective management of the geo-technological complex as a whole.

On the assumption of the fact that the mining company as a socio-economic system represents an orderly capital totality, man-made objects, the subjects interacting in the mining industry, as well as from the need to combine in their corporate and individual interests of each subject, the concept of construction of the automated management system is built almost on the same principles that apply in this case, with respect to the components and geo-technological complex objects and occurring therein mining and transport processes.

Thus the automated corporate management system of geo-technological complex is understood as a set of program-technical and program-methodical provision integrated into a single system, implementing on the basis of objective information produced and in the automated real-time monitoring and control mode, complex optimization, planning and management aimed at efficiency interaction improvement of interdependent environments of "Ore body", " Quarry space", "Mining and transportation complex" and "Enterprise management system", as it is shown in Figure 1.

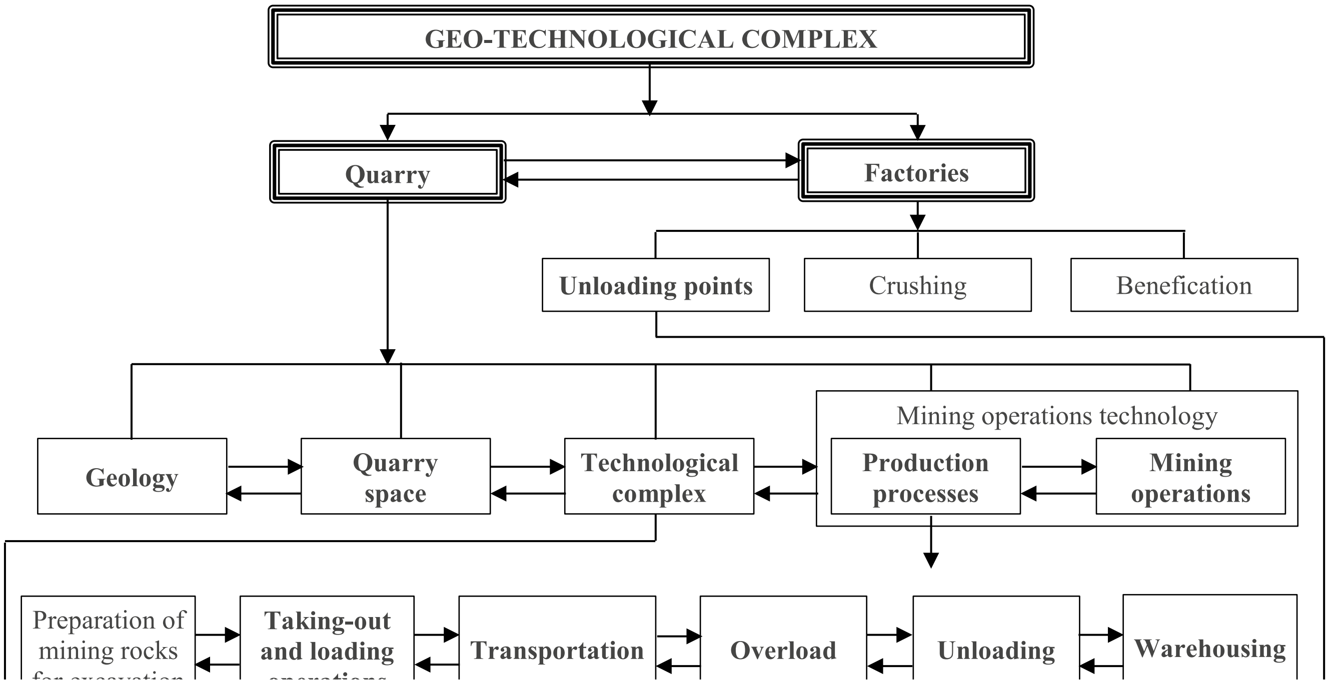

Figure 1 - Geo-technological complex structure.

Management of geo-technological complex is aimed at successive raising of the level and subsystems and components interaction improvement, as well as increase of their own perfection level.

The main subject-matters under consideration in the management process of geo-technological complex are economic, technical, technological and organizational efficiency of its functioning. Management complex profitability and unit cost of mining and transport operations on the rock mass are taken as main criteria of geo-technological complex operation efficiency.

Due to the adequate reproduction of economic-mathematical structure and content model of geo-technological complex these figures integrate all its subsystems and components efforts and make it possible to obtain reliable response of the controlled object to all managerial decisions, thus creating the possibility of high-quality economic evaluation.

The concept of the automated system of corporate geo-technological complexes management formation of mining companies is based on corporate management methodology [*Competitive Advantage: How to achieve high results and ensure its sustainability, Porter*] and involves versatility and its successful use in the technology application on open, combined, underground and open-underground methods of solid minerals deposits development.

The information base put into the automated monitoring and dispatching system is built on the analogy with the simulation methodology and mining and transport processes optimization, as well as to meet the requirements of design technology and planning of mining operations set forth in the works [*Section modeling with railway transport, 1972, Technological bases of cargo flows design and planning at ore quarries with road transport: Dissertation. 1987, The basic principles of logical and statistical modeling simulation of excavator-automobile quarries systems, 1993*].

Automated corporate management system of geo-technological complex is based on the following principles: operational, objective and reliable accounting information on the mining and transport complex; an integrated approach in general and local issues solution; an adequate detailed elaboration of accounted processes and operations; economic assessment of ongoing activities on the basis of micro- and macro- criterial indicators; operational expert assessment of the planning effectiveness and mining and transport operations implementation ; hierarchical decomposition of output information flows; comparability on the structure, format and content of information flows circulating in the corporate management system and in the process of simulation operation of mining and transport complex.

During implementation of the approach taken it is crucial to provide the opportunity of integrated and generalized evaluation of individual operations of production processes. The approach should be based on the main generalized efficiency criterion of the enterprise efficiency as a single natural-technological complex.

Geo-technological complex general schematic representation as the object of study includes: the block of "Ore body", which represents the amount of mineral reserves, limited by space and minerals potentially implemented on the mineral raw market; the block of " Quarry space" which is recorded as a dynamically developing part of the general system and determining geological and mining conditions of mining-transport complex operation; the block of "Superstructure" which is a dynamically developing active part of the geo-technological complex, which includes mining and transport complex and management structure of production.

The components of the given unit of geo-technological complex are combined in one area, as they are considered in the framework of the approach implemented as a single self-regulating system, influencing the passive part, including blocks of "Ore body" and "Quarry space".

In economic terms the system under consideration is divided into two main parts: the "Profitable" one which is provided in the course of industrially developed reserves of mineral reserves and "Cost-based" which represents the total production costs and useful components extraction.

The latter, thanks to adequately built economic and mathematical model of mining and transport processes, makes it possible to implement corporate management of geo-technological complex based on high-quality technical and economic analysis of technological processes in specific situations and conditions.

Efficiency of corporate management of geo-technological complex is provided by a complex implementation of two main subsystems in automated functions mode: automated system of mining and transport operations of monitoring and dispatching, including the monitoring functions of mining and geological and environmental situations in the quarry space, as well as mining and transport operations security; subsystem of "Analytical unit", which includes the modules of analysis and optimization of mining and transport complex operating parameters on the simulation modules basis, mining and transport work unit planning module, the module of reproduction management of production capacities.

Geo-technological complex management is a subprocess of socio-economic development of the mining enterprise. The complex of management methods of socio-economic development of the mining enterprise includes such subsystems as technical, technological, economic, social and financial.

Management functions of each of these subsystems are to a large extent also functions of geo-technological complex management system. Carrying out of analysis and control of geo-technological subsystem of the mining enterprise is a component of the similar approach to the enterprise technical subsystem.

Similar to the case of consideration of mining enterprise development management system, geo-technological complex management system represents a double-set complex of interrelated processes of strategic and operational management.

Strategic control path determines functioning of ,the geo-technological complex, its subsystems and staff aimed at achieving of long-term goals and targets (for a year or more), while the operating outline determines every day work in the frames of days and up to a month alternation.

Management in the frames of the operational outline must be subordinated to achievement of strategic objectives that requires regular and appropriate evaluation and in case of updating necessity of situationally adopted operational managerial decisions. Regulatory impact envisage carrying out of organizational, technical and technological measures on geo-technological complex that can provide proactive impact on the significant internal and external factors.

One of the essential components of the automated management systems of complex systems is the availability of information feedback, including a set of performance indicators, providing qualitative and quantitative analysis and evaluation of the reaction of the controlled object to the solutions of the manager.

In the frames of ACSMS GC such role is played by automated system for operational monitoring of mining and transport operations with an adequate view of the mining-technical, mining-geological, mining and geometrical, organizational and economic conditions of mining and transport complex operating.

Economic and mathematical model which reliably reflects the structure of the mining transport process operation by operation is integrated to ensure realization of functional and pricing method of analysis into monitoring system.

Thus, the cost of mining and transport operations, as an indicator that integrates a whole series of interactions of components of mining and transport system complex of geo-technological enterprise is in the capacity of main effectiveness criterion of the adopted management decisions.

In the frames of the enterprise technical subsystems structural and functional analysis allows to consider and evaluate the content efficiency and mining and transport technological complex operation taking into account specific mining and technical, mining and geological, mining and geometrical, organizational and economic conditions of its operation.

In the aspect of planning it is concerned with evaluating of the effectiveness of natural and productive resources use, including equipment, labor, working capital, electricity and fuel.

As part of technological subsystem of mining enterprise management of geo-technological complex lies in the adopted development systems efficiency analysis, substantiation of various technological parameters of mined-out space and mining and transport systems.

Geo-technological complex parameters as an economic enterprise subsystem can be optimized using the method of functional and pricing analysis and pricing criteria usage during optimal solutions selection.

Functional and pricing method is an effective one in searching ways to reduce costs and optimize the modes and conditions of mining-transport enterprise operation for effective management of geo-technological complex development.

Quantitative performance criteria should be used in order to compare possible options and assessment of their compliance with the set objective.

In this regard, one of the most effective factors of automated corporate management of geo-technological complex is economic and mathematical modeling in the framework of functional and pricing analysis of mining and transport systems quarries and mines development and operation.

For mining and transport process optimization it is necessary to take into account its peculiarities and consistent patterns.

By virtue of the fact that the current mining production conduct of the experiments is very expensive and practically difficult to realize, requiring changes to the manufacturing process, then in the planning, organization and mining production management (consideration of options accommodation in the quarry space or numerical ratio of mining and transport equipment), it can lead to substantial losses.

Experiments under planning, arrangement and management of mining production due to the presence of failed solutions (consideration of accommodation options in the quarry space or numerical ratio of mining and transport equipment) can lead to significant losses.

MONITORING AND DISPATCH SYSTEM TECHNICAL SUPPORT

Technical engineering of automated corporate management system of geo-technological complex is offered within the concept, providing for the use of unique technical means collected on a fundamentally new concept of the positioning and communication system formation based on the use of wireless positioning technology of NanoLock within Mesh-network and RTLS, global positioning of Glonass / GPS and data backup system through 3G/GPRS channel, which is planned to apply an automated system for monitoring and mining and transport operations dispatching in the frames of this concept.

The structure of the offered innovative positioning and communication systems includes: NanoLock, Mesh, RTLS retransmitters, the base station, on-board systems with an interactive display, the server part of the system, the client-end.

Unlike traditionally used technologies in modern systems of dispatching positioning of the main mining and transportation equipment based on satellite navigation the offered technical support makes it possible to improve the positioning accuracy up to one meter and efficiency of the inquiry of the positioned units state of the processing facility up to one second which is crucial for the applicable methodology of feasibility analysis and development of effective managerial decisions under planning and implementation of mining and transport operations on its basis.

In addition, the proposed technical engineering allows to implement effectively the modules for environmental monitoring and mining and transport operations security in a quarry in the frames of ACSM GC.

Operational records of environmental emissions and positioning of the auxiliary personnel involved in repair work as well as ongoing routine inspections of vehicles are maintained in the operation process of the automated management and dispatching system.

The possibility of effective application of the ACSM GC in open, underground and combined underground open-technology development of mineral deposits is significant advantage of this approach in technical engineering.

The ACSM GC program- technical support through a single graphical editor provides users for the Systems suitable for analysis and information reading in tabular and graphical modes.

Monitoring of mining and transport operations is carried out in 3D format with career space rendering and the main mining and transportation equipment in real-time positioning in it.

The graph of shift changes in the cost of mining and transportation projects with the average daily, monthly and annual averages are given according to the results of each shift, quickly issued busiest transport communications graphs on sites and nodes, fuel consumption charts on them.

Separate charts illustrate the distribution of current expenditure on mining and transport complex for excavator-automobile, excavator-rail and in the whole excavator-automobile-railway complex, with laying-out by loading and unloading points on highway and arrangement of tracks, dump trucks and locomotive waggonage.

ACCOUNTING AND ENERGY CONSUMPTION OPTIMIZATION

Currently, energy conservation and energy efficiency increase are important economy components of any industrial enterprise [*On energy saving and energy efficiency increasing at JSC "ALROSA" (PJSC), 2016*]. With a decrease in mining operations fuel and energy resources total cost increases and the relevance of these factors increases significantly as well.

In this regard, the ACSM GC concept includes a system of accounting and mining and transport complex energy consumption optimization which implies implementation of electricity consumption metering by excavators and locomotive waggonages, fuel for dumps in two directions: analytical (calculated) and physical (instrumental).

Energy efficiency monitoring is assumed by integrating key indicator of energy intensity of production in the form of all kinds of energy resources consumption to the production of 1 ton of rock mass, as well as on similar indicators for excavation, road and railway transport.

Analytical method within AMS&D of MTW «Net MOM» of ACSM GC involves the use of a dynamic approach to the calculation of energy consumption and fuel based on the rate of vehicles (dump trucks and locomotive waggonages movement speed) taking into account passport data; physical state and tractive characteristics; effective mass; coating quality, geometry and specifications of roads and track layout quarry schemes adopted by the transport arrangement (speed limits, admission order).

and also based on the spatial and temporal coordinates of the vehicle operational positioning. The error of the used fuel accounting method under the adequate formation of the input information (timing data, mining and technical conditions of the quarry areas) does not exceed 1-2%.

Analytical method allows not only to take adequately into account energy consumption but also to perform analysis which reveals the true causes of its existing values.

This allows not only to monitor the accuracy of the energy consumption accounting using fuel sensors and energy meters, but also to explain at the cost of what it is formed in such a way, take adequately into account energy consumption at each site of roads and track layout, which is fundamentally important for these indicators normalization, as well as under energy consumption planning and, ultimately, the cost of mining and transport operations.

Adequate energy consumption accounting on highway sections and track layout entails another big area associated with a reduction in the cost of mining and transport operations.

Revelation of the so called the most "cost-based" sections makes it possible to divide highway sections and track layout into categories and further to carry out distribution of funds for their maintenance proceeding from their priority.

This will allow to carry out more purposefully and effectively repair and other maintenance works, thereby increasing the productivity of mining and transport complex, while reducing energy consumption and, ultimately, reducing the cost of mining and transport operations.

Physical energy consumption accounting method, in respect to road transport can be implemented by using two types of sensors:

tank fuel level sensor; flowing type sensor.

The weak point of the individual application of the fuel level sensor in the tank is the lack of centralized monitoring capability due to the need for a telemetry system and communications between mobile units and the server.

In this case it is problematic to receive daily information from the sensors, since the dumps at quarry work in two shifts of 12 hours and taking data from the sensors requires additional equipment and time for reading information from each dump.

Therefore, within the ACSM GC the following variant of fuel consumption metering is offered using the fuel level sensor in the tank.

Fuel level sensors in the tank are connected directly to the terminal monitoring system (gps tracker), with subsequent transfer of the measured value of the fuel level. Fuel sensors represent measurement devices related to capacitive type.

Measurement devices related to capacitive type differ from classic resistive sensors by complete absence of mechanical or moving parts as well as fundamentally different electrical circuit. All this makes the capacitive fuel measuring sensors extremely reliable.

They do not need to be built in gasoline pipe (thus introducing changes into the car design and spending money on its installation). It is simply enough to install sensors in the tank (installation takes a few minutes) and they are immediately ready for operation.

The sensors do not affect the engine operation but at the same time they provide an opportunity to determine accurately time and volume of the motor transport fueling operation and fuel discharge that is not possible to implement when using fuel flow meters.

Taking into account that these sensors are used with quarry transport where dumps are constantly moving at different slope, the use of such sensors gives an error in the static from 5 up to 11%.

The error in energy consumption accounting in dynamics yet is to be determined, that it is possible only when the sensor data are included in the overall monitoring and telemetry system.

Flow-type sensors are available in base equipage of dump trucks and they work on default with the onboard computer. Sensors are designed as counters installed on the fuel line, and to control it in both directions as it is shown in Figure 2.

Defect of these sensors is determined by the fact that the sensors of this type in the fuel control system serves the change in the fuel system structure. One should add to this, if the fuel tank is filled with low quality fuel, the system loses its ability to count accurately the consumed fuel.

Diesel fuel calculation becomes complicated. At low temperatures paraffin is generated in solar oil, which shortens counters paraffin service life time. The error under the application of this method of accounting is minimal but it requires constant recording and additional control, since the fuel used at the dump comes

in different quality.

Instrumental energy consumption for excavators and locomotive waggonage may be provided by the arrangement of the operational data taking with existing relevant counters, as well as with the use of an additionally installed instrument parts, providing more in-depth monitoring of energy accounting on the basis of accounting equipment operation modes (lifting, turning with the load and without it, ladle lowering, shift in the work face, etc.).

It also allows to more accurate keeping of records of the mining shipped freight, which is one of the main problems of mining and transport processes monitoring.

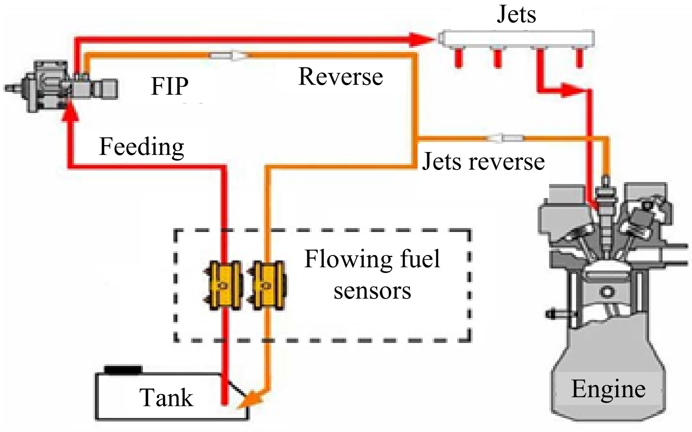

Figure 2 - Scheme of the flow-type sensor operation on the dump trucks.

In the frames of offered concept interaction of accounting system and fuel consumption is proposed. It is implemented in the frames of ASM&D of MTW «Net MOM», with the analytical unit of ACSM GC where the most effective method for such problems solution simulation work of excavator-automobile complexes (EAC) is applied. This method is presented in Figure 3.

If within ASM&D MTW «Net MOM» accounting and control of fuel consumption, as well as the distribution of the flow rate on the motorway sections and modes of vehicles operation are carried out, at that in the analytical block of ACSM GC on the base of the accumulated information on fuel consumption the optimization measures analysis of operational automated and more detailed feasibility analysis are carried out, and that improves fuel efficiency by selecting the most appropriate mode (speed, loading) and conditions (speed limits, the quality of the road surface, track sections geometry, etc.), transport vehicle operation.

Fuel consumption accounting is made on the base of the content of the dump cycle motion from stop to stop in the system of ASM&D MTW «Net MOM» and simulation of the process of mining and transport operations.

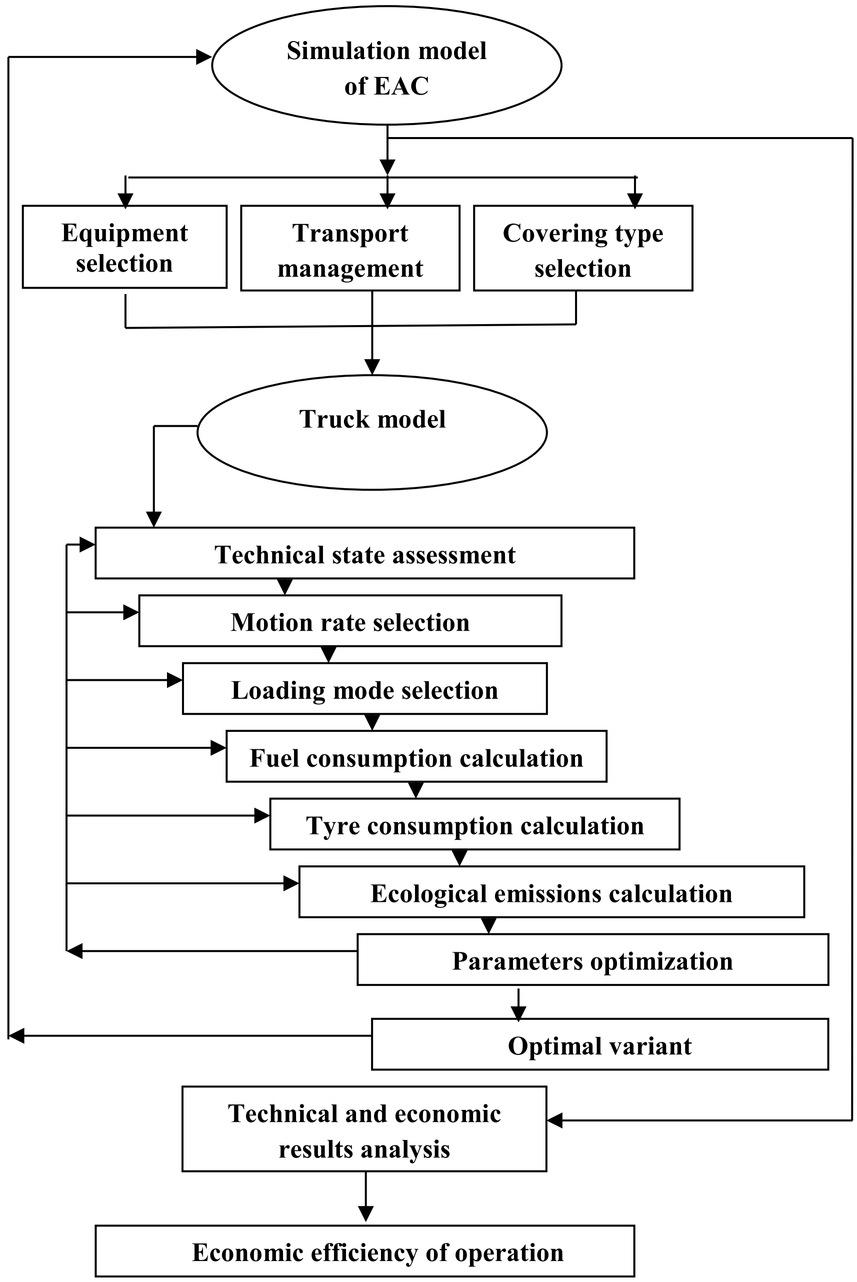

Figure 3 - The structure of the simulation model excavator-automobile complex.

Motion cycle is represented by the following dumps operation modes set in the technological process of mining and transport operations are as follows:

dump speedup from a place to its outlet to a steady state mode;

• motion in traction mode;
• motion under running out;
• braking:

o braking to a full stop of a dump;
o regulation braking;
• idle run engine operation (tickover).

Each of stated driving modes of the dump has a significant impact on the use of engine power, energy consumption, traveling time, motion and other motion parameters.

Thus, the proposed in the framework of the project concept of fuel accounting involves the use of an existing analytical method in combination with the tank fuel level sensors, having integrated them into a single information structure, based on a network mesh (communication system of ACSM GC) and on a complex integrated hardware with the help of which one can receive and process information from all subsystems used.

This approach will make it possible to process the received information from three sources and avoid maximum permissible errors (more than 2-3%).

The proposed concept of the energy consumption accounting system and optimization implementation creates significant potential in solution of social and environmental problems of geo-technological complex functioning and as well as mining enterprise functioning as a whole.

Thanks to high-quality fuel consumption monitoring, taking into account ongoing technological modes and conditions of technological motor transport operation, it becomes possible to reliably establish the intensity and volume of polluting emissions in a quarry space.

Knowing the developed space capacity and driving qualitative monitoring of its pollutants concentration in it one can effectively manage ventilation and dust control processes, thereby enhancing mining and transport operations production safety and preventing their forced operation breaks in the event of reaching of pollution quarry space limiting, also mining transport operation cost reducing due to cutting down unscheduled payments for exceeding the maximum permissible pollutants emissions.

In case of energy consumption by locomotive waggonages qualitative electricity metering enables to carry out a whole range of innovative research complex for implementation in the production of traction units with traction aggregates of electric power accumulator, which leads to a significant rail transport, mining and transport operations reduction cost, as a whole due to elimination of electricity transmission lines in a quarry space and increasing of energy consumption efficiency.

PLANNING AND REGULATION

A unique feature implemented at ACSM GC is the mining and transport operations planning process which is being based on ASM&D MTW «Net MOM» data is carried out within the frames of analytical block using mining and transport processes simulation models.

Thanks to a common information subsystem base of ASM&D MTW «Net MOM» and of Analytical block their interaction effectiveness is caused by the presence of a prompt, reliable and objective information about the work and the state of the mining and transport complex as a whole, of its subsystems and components.

In the frames of operational planning on mining and transport operations dispatching phase qualitative characteristics conversion and coordination within quarry ore flow, planned performance fulfillment for each piece of equipment, for the excavators parks, dump trucks and locomotive waggonages, as well as for mining and transport sector as a whole are controlled and regulated.

Feasibility studies is performed, optimum volume, modes and conditions of mining and transport operations are determined in the frames of the weekly and daily, mid-term and long-term planning on the appropriate methodological support.

Annual plan for mining and transport operations in automated mode is formed on the basis of monthly plans.

The degree of replaceable, monthly and planned targets fulfillment over the period is determined in accordance with each shift results. If necessary their correction is carried out.

Planning of rreproduction processes of production capacity with repairs schedule construction of the main mining and transport equipment is carried out in the frames of a separate module of automated mode.

In mining and transport operations planning process there is possibility of high-speed modes rationing, energy consumption, productivity, cost allocation for mining and transport processes maintenance based on mining and transport operations optimality and safety.

CONCLUSIONS

The accepted approach to geo-technological complex management at mining enterprises is innovative both in content based on the latest technologies, as well as it is functional, when the mechanism of innovative solutions adoption is conditioned thanks to a new set of program-technical and methodological possibilities.

New solutions are developed on the basis of operational data of automated monitoring and dispatching system of mining and transport operations analysis, search and optimization of new solutions with the use of highly effective simulation method.

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DATA STORAGE DEVICES: A REVIEW

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DATA STORAGE DEVICES: A REVIEW

ABSTRACT

Tellurium based materials are used in present day commercial optical memory devices and are potential candidates for future solid state data storage devices for industrial applications. The mainstream memory technologies and systems include solid state memory, hard disc drive, and optical disk. Each technology has its own special market and application although there is some overlap. Solid state memories, which have high speed and compact size, are mainly used as primary (internal) memories, and magnetic and optical data storage devices are typically used as secondary devices for computer systems. In connection with the improvement of digital computers, extensive research work is being done to increase the capacity and speed of their memory devices. The phase change memory has been devoted to binary activity, which is the basic mechanism for data storage in optical and electrical memory devices. The strategy is always to make the spots/marks smaller and the density much higher. This work presents some of the important contributions in terms of materials development and device fabrication for current and future data storage devices.

KEYWORDS

Phase change materials, Data storage, optical memories, CD, DVD, BLU RAY, chalcogenides, GST, AIST, CD-RW, DVD-RW.

INTRODUCTION

Advances in multimedia have resulted in computer performance getting faster and the stored information denser [1]. Tellurium based alloys (Ge-Sb-Te and AgInSbTe) have been commercialized in optical storage media and are strong contenders to replace non-volatile flash memories. An important requirement is the ability of media to be used repeatedly, that is, to be rewritable. Conventional methods such as magnetic recording, is limited in further increasing the recording density because it requires huge write field for high anisotropic media.

Phase Change Materials (PCM) based Solid State Memory devices and Digital-Versatile Disc-Rewritable (DVD-RW)/blu ray disc (BD) are a billion dollar industry in the present multi-media market. Current multimedia (Compact disc (CD), Digital Versatile Disc-Random Access Memory (DVD-RAM)) strongly relies on re-writable phase change optical memories using $Ge_2Sb_2Te_5$ (GST) as the phase change material [2, 3]. GST is also so far considered to be a desirable material for solid state memories--Phase Change Random Access Memories: PC-RAMs.

The idea to use an amorphous-to-crystalline phase transition for information storage dates back to the 1960s when S.R. Ovshinsky suggested a memory switch based on changes in the properties of amorphous and crystalline phases of multicomponent chalcogenides [4, 5]. A landmark was achieved in the late 1980s by Matsushita/Panasonic who developed phase change optical disc technology that remained stable over a million use cycles [6]. This technology became the mainstream in optical disc production and in the late 1990s resulted in the commercialization of 4.7GB Digital Versatile Disc-Random Access Memory (DVD-RAM). During the development process, various materials were examined, and the best performing in terms of speed and stability were found to be $Ge_2Sb_2Te_5$ (GST) used in DVD-RAM and an AgInSbTe alloy used in CD-RW [7, 8]

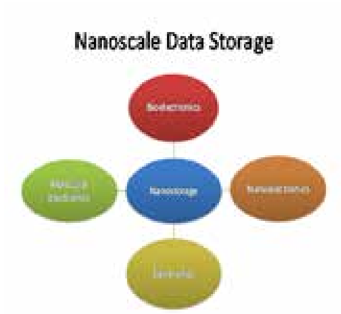

Figure 1: Techniques being explored for scaling of data storage devices.

Chalcogenide films with reversible amorphous-crystalline phase transitions have been commercialized as optically rewritable data storage media, [9, 10] and intensive effort is now focused on integrating them into electrically addressed non-volatile memory devices (Phase change random–access memory or PCRAM) [11, 12]. Since the energy required for phase transformation decreases with cell size, the write current scales with cell size and thus facilitates memory scaling. Major issues related to PCRAM devices are (i) Material Optimization. (ii) Scaling of the memory cell by synthesizing materials at the nanoscale using bottom up approaches of chemical synthesis and vapor-transport method and (ii) Investigation of phase change characteristics of nanomaterials and nanodevices. Figure 1 shows the various techniques being explored for scaling of data storage devices.

Major Parameters to be addressed for Phase change Memory Devices include

- Switching speed
- Cyclability Endurance
- Data retention time
- Storage density – Size Scaling

Material requirements for data storage

- Large atomic mobility in amorphous and super cooled states
- Short atomic diffusion distance from the atomic location in amorphous state to the lattice sites of the crystalline states.
- Melting point: 500- 1000°C; $T_g \sim 2/3\ T_m$; T_g is the glass transition temperature and T_m is the melting point.
- Thermal Stability: Long lifetime at room temperature
- Optical constants: Optical contrast between the amorphous and crystalline phases

Basic Structure of a Compact Disc

As shown in Figure 2, the compact disc consists of different layers:

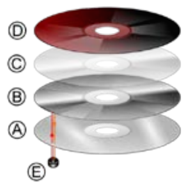

Figure 2: Diagram of the basic CD layers [13].

A. A polycarbonate disc layer has the data encoded by using bumps.
B. A shiny layer reflects the laser.
C. A layer of lacquer protects the shiny layer.
D. Artwork is screen printed on the top of the disc.
E. A laser beam reads the CD and is reflected back to a sensor, which converts it into electronic data.

The information layer is covered with a metallic reflective coating to enable information to be extracted by means of reflected light. This is achieved by focusing the light from an Aluminium Gallium Arsenide (AlGaAs) laser onto the track. This diode is a light source of considerable less power than that used for writing the master disc. The laser light concentrated into a spot of 1.87 microns in diameter, follows the track thereby striking pits and non pits alternately. Due to this, light will be lost because it is diffracted over angles larger than the lens is capable of accepting. Thus the intensity of the reflected light is modulated by the physical structure of the disc and this is detected by a photodiode which, in turn, produces a modulated electrical signal. A comparison of the three generations of optical discs is shown in Figure 3 and the parameters are given in Table 1.

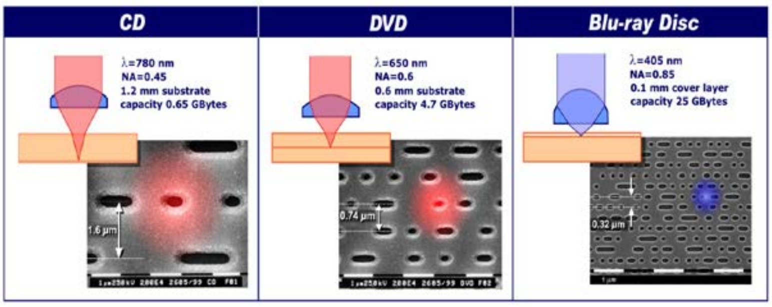

Figure 3: Three Generations of Current Optical discs. Courtesy Phillips Research Laboratories, Netherlands

Table 1: Parameters for the three generations of optical discs.

Parameters	CD	DVD (single layer (SL))	Blu-ray (SL)
Storage Capacity GB	0.65	4.7	25
Data Transfer Rate (Mbps)	1.41	11.08	36
Recording Track Pitch (mm)	1.6	0.74	0.32
Channel bit Length (mm)	300	133	
Laser wavelength (nm)	780 (1.59eV)	650 (1.91eV)	405 (3.06eV)
Numerical Aperture	0.45	0.6	0.85

Alternative Technologies for Data Storage

Getting to an optimum data storage media involves an interplay between competing materials and technologies. Some of the technologies being explored by researchers are outlined below.

Holographic recording

The holographic memory is based on the concept that some reference beam can be used for recording holograms and for reconstructing the information to achieve the reconstructed images of various holograms to be placed in exactly the same position. Therefore, the virtual images of the holograms are used for reconstruction. Holography has the ability to read and write millions of bits of data with a single pulse of light, enabling data-transfer rates of billions of bits per second. The physical mechanism is photochromism, which is defined as a reversible transformation of a single chemical species between two states that have different absorption spectra and refractive indices. Figures 4 and 5 show a simplified mechanism of holographic recording and the formation of refractive index pattern in a photopolymer upon exposure to a hologram respectively.

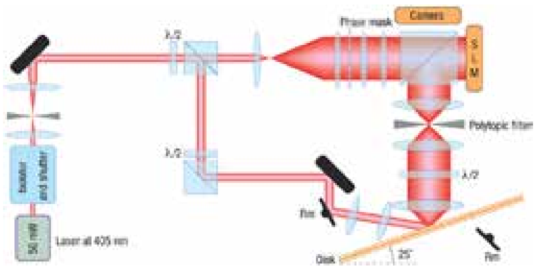

Figure 4: Optical architecture for recording data using polytopic multiplexing and phase conjugation. The data is recorded using interference between the two paths. It is read using just the lower path, terminating at the camera instead of the spatial light modulator (SLM). $\lambda/2$ is a half-wave plate and Rm is a rotating mirror. Reproduced with permission from Macmillan Publishers Ltd., Nature Materials [14]. Copyright 2008.

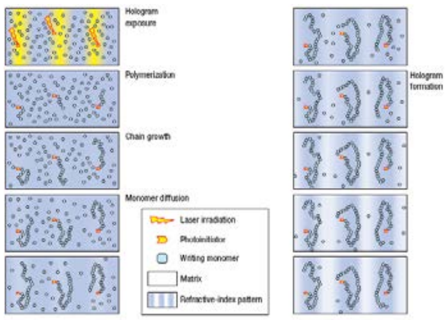

Figure 5: Formation of a refractive-index pattern in a photopolymer following exposure to a hologram. Reproduced with permission from Macmillan Publishers Ltd., Nature Materials [14], Copyright 2008.

Fabrication of Patterned Media by Nanoimprint Lithography (NIL)

Patterned media using techniques such as photolithography, e-beam lithography, X-ray and interferometric lithography offer significant control over achieving high areal density structures. For example, a 50-nm-period patterned medium corresponds to a density of 250Gbit/in^2, and a 25 nm-period one corresponds to a density of around 1Tbit/in^2. Therefore a patterned medium is a promising candidate for the future recording system with a capacity beyond 1 Tbit/in^2 [15]. Nanoimprint lithography provides a way to fabricate sub 10 nm structures.

Two Photon Technology

The optical data-storage technologies, such as CD, DVD and Blu-ray, use one-photon-absorption processes for writing and reading information. In particular, a bit of information is recorded when photons are absorbed and induce a change in the local reflectivity of the medium, often as phase or structural changes, or burning.

In contrast, a two-photon absorption process occurs when two photons are absorbed simultaneously at the same spatial location. As two-photon processes are nonlinear and require high-optical intensities to occur, they typically only take place at the focal spot of a light beam. This characteristic offers the ability to store data within the volume of a disk, thus achieving three-dimensional (3D) storage by simply changing the focusing position of the writing beam within the medium. Using two-photon processes it is possible to store hundreds of layers within the volume of a 1-mm-thick DVD-type disk.

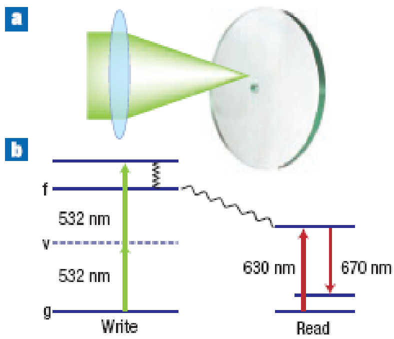

Figure 6: Two-photon processes. a, Writing within the volume of a disk. b, Two-photon-absorption energy levels of writing and reading. g: ground state; v: virtual state; f: final state. Reproduced with permission from Macmillan Publishers Ltd., Nature Materials [14], Copyright 2008.

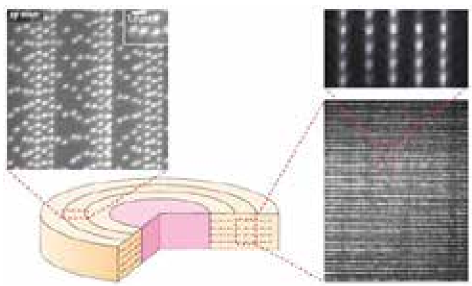

Figure 7: Layers recorded in a 3D disk. Reproduced with permission from Macmillan Publishers Ltd., Nature Materials [14], Copyright 2008.

In two-photon 3D optical data storage, writing is performed by short laser pulses at a wavelength of 532 nm (green), for example (Figure 6). One 532-nm photon alone will either be absorbed at the surface (if its energy is enough to excite the medium) or it will be transmitted through the volume of the medium if it is of less energy. However, when two photons overlap simultaneously where the laser beam is focused, two-photon absorption can occur imparting energy (with an equivalent wavelength of 266 nm) of twice that of the individual photon energy to the medium (Fig.). This energy can be used to locally change the medium's properties (such as making it fluorescent) and thus write a data bit of information. In order to read the data, the written bits are illuminated with a red laser diode that induces fluorescence from the written bits, which is collected by a photomultiplier tube or a photodiode array.

Third Harmonic Generation

THG on 3D bits in a transparent material were first reported by Lionel et al [16]. The contrast mechanism is neither a change in refractive index nor a change in absorption, but a change in the third-order susceptibility $\chi^{(3)}$ induced by femtosecond laser irradiation in a zinc phosphate silver containing glass. They demonstrated the possibility of 3D optical data storage inside a specific zinc phosphate glass containing Ag by using THG. The data are stored inside the glass by femtosecond laser irradiation below the refractive index modification threshold. By the accumulation effect, stable Ag clusters are created, giving rise to a formation of a nonlinear resonant interface. THG readout thus becomes possible.

Near Field Phase Change Optical recording

Near field optical recording was first proposed and demonstrated by Betzig et al. [17] that by means of near-field scanning optical microscope (NSOM) it is possible to achieve ultra-high density data storage. Fourier optics can be used (18, 19) to demonstrate that the diffraction limit to resolution in optical microscopy is not fundamental but rather arises from the assumption that the detection element (that is, a lens) is typically many wavelengths away from the sample of interest. However, by laterally scanning a source or detector of light in close proximity to the sample, one can generate an image at a resolution functionally dependent on only the probe size and the probe-to-sample separation, each of which can, in principle, be made much smaller than the wavelength of light.

Semiconductor -Electrical Memory

Classification of semiconductor memories is given in Figure 9. The basic mechanism of threshold switching is shown in Figure 10.

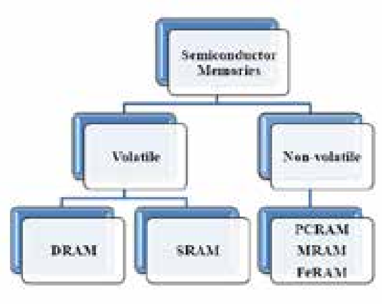

Figure 9: Classification of Semiconductor memories

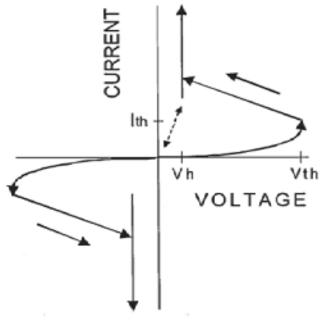

Figure 10: Principle of threshold switching

Phase Change Random Access Memories: PCRAMs

PCRAM is also known as Ovonic Unified Memory (OUM) and was originally developed by Energy Conversion Devices, Inc. (ECD), and licensed exclusively to Ovonyx, Inc. OUM uses a reversible structural phase-change - from the amorphous phase to crystalline phase – in a thin-film chalcogenide alloy material as the data storage mechanism. The small volume of active media in each memory cell acts as a fast programmable resistor, switching between high and low resistance with >40X dynamic range. The programming pulse drives the memory cell into a high or low resistance state, depending on current magnitude. Information stored in the cell is read out by measurement of the cells resistance. GST is so far considered to be a desirable material for solid state memories--Phase Change Random Access Memories: PCRAMs. Self-assembled nanowires-based phase change material memory devices offer an attractive solution owing to their sub-lithographic sizes and unique geometry, coupled with the facile etch- free processes with which they can be fabricated.

CONCLUSION AND SUMMARY

The issue of addressing the need for improved data storage for next generation optical memories has been led by Blu Ray. Blu Ray has addressed this issue by reducing the spot size for recording data by moving to 405nm from 630nm and has doubled the capacity. Nevertheless, it is difficult to envisage how shorter UV wavelengths can be used to further enhance the density of data storage, as few materials have the transparency in that region. Thus several proposals have been made to overcome the limits facing the industry, none being more promising than 3D storage using holography. This technology is poised to offer truly gigantic increases in the capacity to Terabytes (TB) in disks of the size of current DVDs, but significant challenges exist in solving ghosting and higher order diffraction issues. Reading speeds are also a limitation, as these tend to be slow. Other technologies have also been suggested, such as second and third harmonic generation, which may allow multilayer (>10) writing to become reality, increasing storage capacity many-fold.

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DETERMINATION OF GEOTECHNICAL MASS PARAMETERS IN THE PHASES OF EXPLORATION AND EXPLOITATION OF MINING THROUGH A WEB APPLICATION

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DETERMINATION OF GEOTECHNICAL MASS PARAMETERS IN THE PHASES OF EXPLORATION AND EXPLOITATION OF MINING THROUGH A WEB APPLICATION

ABSTRACT

A web application to define geotechnical massive strength parameters is proposed. The aim of this study is to develop a practical system and easy access to project data and geotechnical works as well as its interpretation for the phases of exploration, development and mining, for open pit mining or underground. Thus, this enables up by inserting the survey data SPT, calculation the parameters strength of the geotechnical mass instantly for various published correlations world, also allowing the user to insert formulations that have not yet been catalogued. The results can be displayed according to each geological layer characterized by probing or per meter to meter hole in tables or charts, where it is possible to compare each of the methodologies applied in a simple and fast way. In order to validate the application presented a case study to a survey carried out in the region of Belo Horizonte is proposed. Other advantages of using the web application can be related to its rapid access, data storage for future uses and queries, allowing scope to any sector involved and interested in the acquisition and processing of data, helping to improve the communication between different teams. Also, are software categories in frequent evolution, directed to a unique content of interest and reduced development time.

KEYWORDS

Web application, correlations parameters, SPT.

INTRODUCTION

It is known that during the exploration phase and exploitation of mining, SPT surveys are widely used in order to know all the geomechanical parameters of the massive study, for later identification of the materials that constitute it, its physical properties, and finally determining the type of mining and shape of the dig.

These parameters can be obtained by traditional mechanics laboratory testing of soil and rock mechanics or through correlations with field tests published worldwide. Facing the difficulties in the process of obtaining undisturbed samples, especially in sandy materials and rock formations, correlations with field tests have been used more often than the execution of tests in laboratories. According Schnaid and Odebrecht (2014) the geomechanical parameters obtained through field tests represent reliable and safe way what happens in reality, in addition to having less time getting compared to the performance of laboratory tests.

The SPT survey is notably one of the most widely used tests for application in geotechnical projects and subsurface investigations. According to Bowles (1997) the test, developed in 1927 runs up to 90% of projects in South America and North America. Its use includes so many methods of direct calculation, as indirect. In the approach by the direct method, by correlation with the value of the penetration resistance index, it becomes possible knowledge representative strength parameters of the behaviour of the ground, such as: angle of friction, cohesive intercept deformability modulus and specific weight.

Looking for an improvement in the design and execution of works related to geotechnical engineering, it is inevitable the use of innovative tools that aim conciliate the challenges faced by companies to technology advances. The web applications are categories of software that has been with advantages to be used in engineering projects, because, at the same time that assist in the practicality and availability of data, are in constant evolution and have short-term development. These characteristics, position the applications as key tools in the preparation and execution of geotechnical projects, resulting in communication improvements and agility in them. And they operate while minimizing the presence of a possible geotechnical consultant.

This paper proposes the use of a web application for the calculation of the main parameters of resistance of a geotechnical massive, based on a survey in a mining in the region of Belo Horizonte in the state of Minas Gerais, Brazil.

Besides the advantages already mentioned for the professional market, the application also features improvements related to teaching and learning model in academic communities. According to Sales (2010) web applications are tools that have proven highly beneficial in the learning of the students, as they help in the design of different theoretical situations imposed in the disciplines the practical reality and still drive the question of capacity new information and its subsequent transformation into knowledge.

The application presented in this paper confirms the author's view, since it is an aid program for teaching subjects involving calculations of massive geotechnical strength parameters. It is mentioned also as a point positive the fact that the application, directed the analysis and interpretation of results, and not only the calculation itself, enable the graduate student knowledge of numerous correlations, as well as the interpretation and comparison of the results provided by each one of them.

THE WEB APLICATION

Technical information

For the development of CsA Geo software was used oriented programming language C # objects with MVC architectural pattern - Model-View-Controller. The creation of the screens and the graphical interface was through HTML coding 5 and Bootstrap Framework. The system can be accessed from any platform (hardware) such as computers, tablets and mobile phones depending only on a connection to the Internet through any web browser. Because it is a web application server with the "clouds" (Figure 1) the possibility of loss or input data loss or program output is zero, becoming thus a highly efficient tool.

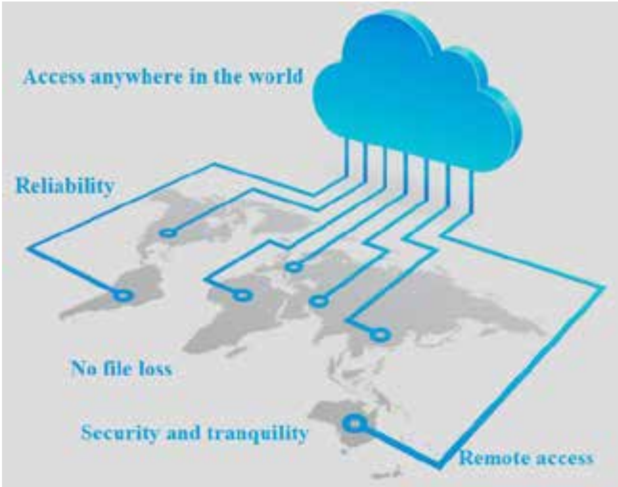

Figure 1 – Schematic representation of the scope / potential web systems. (Adapted by Porto, 2015)

Execution: Case Study

In subchapters below will be detailed implementation of the proposed application. To validate it, a case study to a survey in a mine located in the region of Belo Horizonte in Minas Gerais state, Brazil is presented. The survey taken for this case study is one of the 175 made in the industrial area of mining, the mixed type, the method was percussion. The type of soil found predominantly in the survey was the sandy type, and in the final meters found granite rock type, is shown in Figure 2 the core sample taken during polling, in Table 1 and Table 2 details the data from the bulletin.

Figure 2 – Core sample taken from the survey under review

Table 1– Details the data from the bulletin required for analysis in the web application

Identification	SM – 01
Elevation	285,4 meters
Start / End	17/11/2012, 22/11/2012
Water level	6,1 meters
Total depth	13,9 meters

Table 2 – Description of the materials found in the survey

Depth (m)	Type of material	Identification
0 a 1,00	Sandy soil brownish color, zero plasticity. Colluvium.	Layer 1
1,00 a 8,90	Soil sandy-silty yellowish brown, low plasticity and compactness ranging from little compact to very compact. Residual soil gneissic granite.	Layer 2
8,90 a 10,60	Gneissic granite consistent and little changed. The rock is fractured and has an incipient banding.	
10,60 a 13,40	Gneissic granite, very consistent. The rock is moderately fractured.	Layer 3
13,40 a 13,90	Gneissic granite very consistent and very changed. The rock is heavily fractured.	

<u>Cataloguing survey of polls</u>

Following, in possession of bulletins probing of the area being studied the inclusion of the initial data is conducted, they are: elevation, stoppage of depth and height of the water table (if any). Simultaneously with this process the application generates a representative illustration of the hole depth and the corresponding level of water. In Figure 3 there is shown screen of insertion of the initial information.

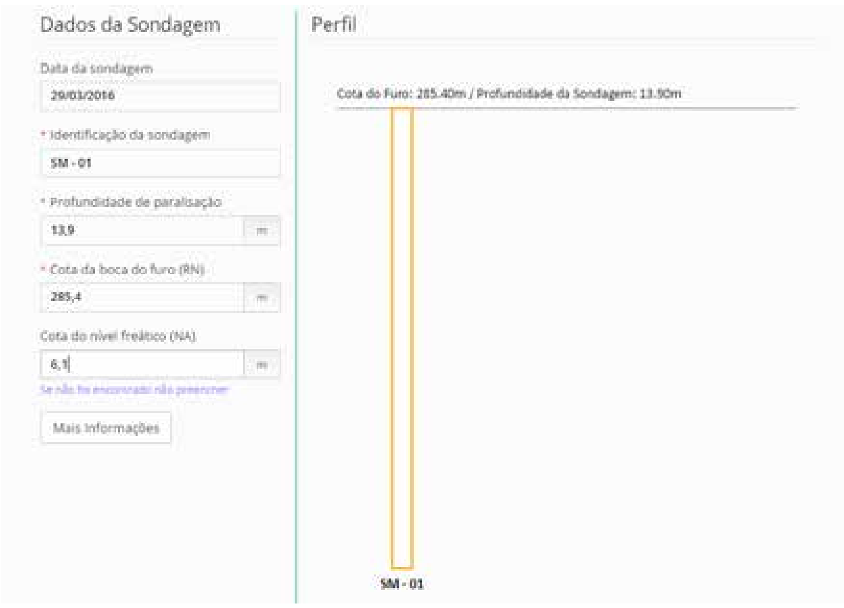

Figure 3 – Insertion of the initial screen of the survey information

<u>Geological classification</u>

After insertion of the primary data, the application provides the "Geological Classification" tab. At this stage, the characterization of geotechnical massive obtained by survey should be detailed, for the top layer to the depth of downtime. The required information is the type of soil and the thickness of the layer composed by him. The application provides this soil type illustration on each layer until it reaches the final depth, generating the geological profile, as seen in Figure 4. Additionally, it allows the user to select or loading hatch which will be assigned to each type of material, to facilitate the correlation between the results generated by the application as described in the survey.

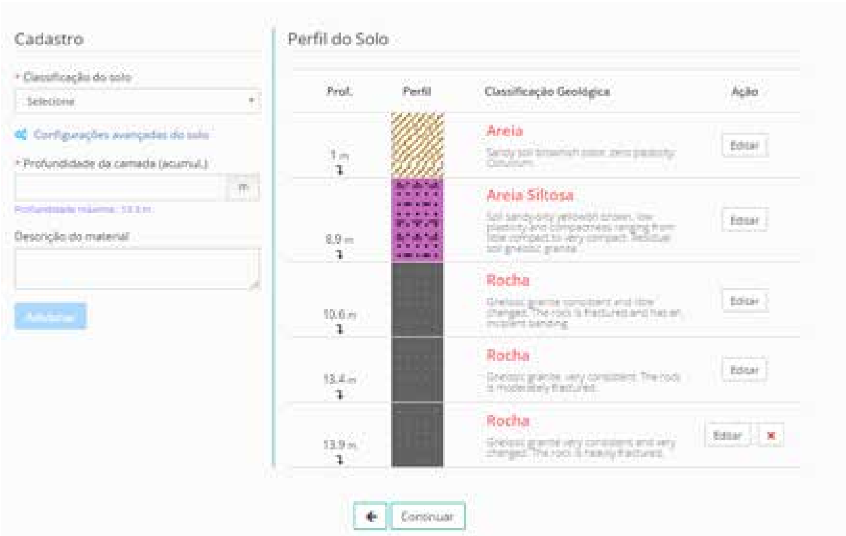

Figure 4 – Soil profile analysis

<u>Calculation settings</u>

Entered all the data collected during the course of the SPT survey, the user must select which parameters wish to calculate to the displayed profile and which methodologies. In Figure 5 there is shown application screen with the "System Default", "Custom", "Insert formula" that correspond respectively to the calculation settings already previously selected by the user, the possibility of changing the parameters and methodologies previously selected and the inclusion of a formulation that is not yet available. For the case study presented in this work, all the methods shown in Table 3 were selected.

Figure 5 – Calculation options available in the application

Table 3 – Methodologies and parameters selected in the case study

Parameter	Methodology
Angle of friction	Bowles (1997), Godoy (1983), Hatanaka & Uchida (1996), Meyerhof (1956), Schmertmann (1975), Teixeira (1996)
Intercept Cohesive	Bowles (1988), Décourt (1989), Hara et al. (1974), Hettiarachchi & Brown (2009), Terzaghi & Peck (1972)
Deformability Modulus	Teixeira & Godoy (1996)
Specific Weight	Bowles (1997), Godoy (1972)

Profile of survey and soil parameters

From the data provided by the user and the selection of which correlations will be used the web application provides as the answer three display options of the results:

1. Representative profile of the probe into the hole indicated with the average values of the soil parameters and penetration resistance index by layer material reported on the "Classification Geological" (Figure 6);
2. Table informative containing the mean values of each parameter the meter hole meter (Figure 7);

Figure 6 – Representative profile of the probe into the hole indicated with the average values of the soil parameters by layer

Cota RN (m)	Nspc (golpes)	Tipo Solo	Min	Médio	Máx
284.4	7	Areia	26.8	41.3	106.6
283.4	9	Areia Siltosa	28.1	42.6	110.4
282.4	15	Areia Siltosa	31.2	46.8	122.6
281.4	50	Areia Siltosa	39.5	58	129.5
280.4	50	Areia Siltosa	39.5	57.6	129.5
279.4	50	Areia Siltosa	39.5	57.2	129.5
278.4	50	Areia Siltosa	39.5	57	129.5
277.4	50	Areia Siltosa	39.5	56.7	129.5
276.4	50	Rocha	0	0	0

Figure 7 – Table informative containing the mean values of each parameter the meter hole meter

CONCLUSION

Results generated are shown in Table 4, are presented the same reliable. The results are generated instantly, is concluded that the web application is able to eliminate calculation errors caused by human distractions, reduces the calculation time and present as a result the illustrations and diversified forms of interpretation.

Table 4 – Results generated by the application

Parameter	Methodology	Average values per layer		
		Layer 1	Layer 2	Layer 3
Angle of friction	Bowles (1997)			
	Godoy (1983)	29º	31º	35°
	Hatanaka & Uchida (1996)	27º	33º	35°
	Meyerhof (1956)	27º	35º	35°
	Schmertmann (1975)	28°	32º	35°
	Teixeira (1996)	27º	38º	35°
Cohesive intercept (kPa)	Décourt (1989)	-	-	-
	Hara et al. (1974)	-	-	-
	Hettiarachchi & Brown (2009)	-	-	-
	Terzaghi & Peck (1972)	-	-	-
Deformability modulus (kPa)	Teixeira & Godoy (1996)	21	85	-
Specific weight (kN/m³)	Bowles (1997)	1,4	1,6	-
	Godoy (1972)	1,7	2,0	-

Besides the advantages already mentioned, using the application is possible to notice other positive points of emphasis both for use in the professional market and for use in the academic community, among them can be mentioned:

1. Practical system and easy access to project data and geotechnical works;
2. Generation of multiple results (geological, tables and graphics), allowing the user to choose the most appropriate analysis of its interest;
3. Instant calculation of soil strength parameters for different correlations with SPT published worldwide;
4. Teaching and learning tools: allows students to compare results obtained by different correlations available;
5. Are software categories in frequent evolution, directed to a unique content of interest and reduced development time;
6. Data storage available for subsequent uses and consultations with anywhere access through the Internet;
7. Replacement of a possible consultant required the geotechnical design firms.

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DIVERSIFICATION OF THE MINING AND METALLURGICAL COMPLEX IN KAZAKHSTAN

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DIVERSIFICATION OF THE MINING AND METALLURGICAL COMPLEX IN KAZAKHSTAN

ABSTRACT

The place of Kazakhstan in the world by the reserves of various kinds of minerals and the role of mining and metallurgical complex (MMC) in the country economy are shown. The data on the minerals extraction and production over the past 5 years are provided. The data on the minerals extraction and production over the past 5 years are provided. It is noted that in the deteriorating situation on the world metal markets there is an urgent need to diversify the MMC, providing increasing comprehensiveness and completeness of all the useful components contained in mineral raw materials and the mastering of the subsequent repartition. Also a strong demand for noble, rare and rare-earth metals for the needs of high-tech and knowledge-intensive branches of industries is taken into account. The urgency of the scientific work acceleration is based, aimed at the development and introduction of new technologies, processes and technical tools, providing a more complete extraction in a marketable product of all the components contained in the ore. It is proved that such results can be achieved with the full conformity of the technology of ore processing to its natural properties and technological characteristics. It is shown that all technologies of geological-prospecting, mining, mineral processing and chemical-metallurgical industries, providing increasing comprehensiveness of the mineral useful components extraction, consist in increase their numbers, enhancing the values of the coefficients of their extraction in the concentrate, metal. It is illustrated by specific examples. It is calculated that the total revenue from the sale of related noble and rare metals exceeds the income from core metals (copper, molybdenum) in 9.33. The second direction of the MMC diversification is connected with deep processing of the resulting product, the creation of high-tech and knowledge-intensive manufactures and obtaining products of higher commodity readiness. Implementation of works in this direction is included in the State Program of Industrial and Innovative Development for 2015-2019 years. Restructuring of production of the Kazakhstan mining and metallurgical complex due to reduction of the raw material part and increasing the finished component will allow keep its leading role in the country economy.

KEYWORDS

Mining and metallurgical complex, noble, rare and rare-earth metals, technological indices of minerals processing, useful components, comprehensive utilization.

INTRODUCTION

Modern State of the Mining and Metallurgical Complex of Kazakhstan

As known, Kazakhstan is a major mining power. It is ranked first in the world reserves of zinc, tungsten and barite, the second place by the reserves of silver, lead and chromite, the third place by the reserves of copper and fluorite, the fourth place by the molybdenum reserves, the sixth place by the gold reserves. Kazakhstan is the biggest rhenium producer (second-third place), beryllium (first-fourth place), titanium sponge (second place), tantalum, niobium, gallium, technical thallium, arsenic (third place), uranium (first place), vanadium (fifth place), bismuth (sixth place) (Dzhantureeva, 2015). Mining and metallurgical complex (MMC) has a great influence on the formation of macroeconomic indicators of the country. Share of the industry branch is 13% of Gross Domestic Product (GDP), 23% - in the total industrial production, 48% - in the production of manufacturing industry output, 20% - in the country's export (Rau, 2015a).

Dynamics of extraction and production of the base metals over the past five years is characterized by the data in the table 1. They imply that the production volume of the mining and metallurgical complex is located almost on the same level. However, most of them are exported to the foreign countries in the form of a concentrate, even as raw materials, the value of which by one or two orders below the value of the finished product of the second-fourth stage of processing.

Table 1- Extraction and production of minerals for 2011-2015 years

Product name	Units	2011	2012	2013	2014	2015 till May
Coal	Th. Tons	116 343,1	120510,9	119 860,1	113 843,5	42 317,8
Uranium	Th. Tons	19,45	20,9	22,5	22,83	9,2
Iron ores	Th. Tons	26055,5	25997,8	25 241,8	24 628,3	9 600,9
Iron in ore components	Th. Tons	7 803,2	7360,3	6 919,7	6 250,5	1 987,3
Chromium ores	Th. Tons	5 059,0	5 233,1	5 255,0	5 410,4	1 796,6
Manganese ores	Th. Tons	2 963,0	2 975,0	2 852,1	2 617,3	604,6
Copper ores	Th. Tons	34 491,6	38352,9	41 731,7	38 660,6	13 441,1
Copper powder type	Th. Tons	338 524	367 161	350 837	293 948	151 871
Tin-zinc ores	Th. Tons	4 857,6	4 805,6	4 909,9	5 260,2	2 819,2
Zinc in zinc concentrate	Th. Tons	350,4	370,5	361,1	346,6	133,8
Lead in lead concentrate	Th. Tons	34,6	38,5	40,8	37,8	15,4
Gold ore type	Tons	26, 680	21,134	23,219	26,680	12,772
Silver ore type	tons	976,434	963,179	963,580	976,434	517,784
Bauxite ores	Th. Tons	5 007,8	4 852,0	5 170,2	5 192,8	1 944,2
Asbestos	Th. Tons	223,1	241,2	243,3	213,2	77,1

The global financial and economic crisis of recent years demonstrates the unpromising character of the industry branch policy. So, in 2015 demand for the production of MMC decreased sharply (in 1.3-1.8 times), and its prices fell in 1.5-2.0 times.

DISCUSSION

The Importance of a More Complete Extraction of Useful Components from Ores

In such a situation there is an urgent need for diversification in the mining and metals sector, which should start with a comprehensive and full use of all the useful components contained in mineral raw materials and the development of subsequent stages of processing. Most of the mining and metallurgical enterprises of Kazakhstan often do not extract from the raw material the precious components (platinum, gold, palladium, rhenium, osmium, thallium, etc.), concomitant to the core metals, and they go to waste of the processing and metallurgical industries. At these enterprises the extraction coefficient is very low (about 0.4) (Rakishev, 2013). This paradox stems from the fact that by the process of the approval of the deposit's reserves, the concomitant useful components often are not estimated and so are not put to the balance sheet. There are no requirements to the subsoil users on extraction of useful components, identified in ores in the process of the deposit exploitation.

At the same time it is known that with the development of high-tech and knowledge-intensive manufactures (electronics, robotics, aircraft and rocket construction, space technology, software, nanotechnology, nuclear, solar and hydrogen energy, biotechnology, genetic engineering, etc.) demand for noble, rare and rare earth metals grows very quickly. Moreover, the cost per mass unit (ton, kg.) of these elements is in a thousand times higher than the cost of core metals (copper, zinc, lead), and the cost of osmium is more than a million times greater.

In this connection, the question of accelerating scientific work aimed at the development and introduction of new technologies, processes and equipment to ensure a complete recovery in the commercial product of all the components contained in the ore, becomes extremely relevant.

The solution of this major problem could be based on the account of the features of substance transition from one state to the other, respectively, by the geological exploration and mining extraction works, enrichment and metallurgical processing. Selected technologies and means of processing should ensure the maximum extraction of useful products at the each of these stages. These results are achieved with full accordance of the ore processing technology to its natural properties and technological characteristics (Rakishev, 2010).

By this the control of amount and quality is carried out on the basis of the mathematical models of the mineral raw state (MR) at each of the processing stages, represented in the form of (Rakishev, 2010, 2013):

$$M_b = \sum_{i=1}^{n} m_i \; ; \quad M_{oe} = \varepsilon_{oe} \sum_{i=1}^{n} m_i \; ; \quad M_{lr} = \varepsilon_{oe} \sum_{i=1}^{r} \varepsilon_{yi} m_i \; ; \; M_c = \varepsilon_{oe} \sum_{i=1}^{p} \varepsilon_{ci} m_i \; ;$$

$$M_t = \varepsilon_{oe} \sum_{i=1}^{s} \varepsilon_{ti} m_i \; ; \; M_{м} = \sum_{i=1}^{q} M_{mi} = \varepsilon_{oe} \sum_{i=1}^{q} \varepsilon_{ci} \varepsilon_{mi} m_i \; ; \quad M_{mw} = \varepsilon_{oe} \sum_{i=1}^{1} \varepsilon_{wi} m_i \; . \tag{1}$$

where, M_b are the ore mass at the counter of the balance reserves; M_{oe} is mass of the extracted ore; M_{lr} is mass of the lumpy rock, removed from ore mass; M_c is mass of the all concentrate (concentrates); M_t is mass of the tails of enrichment; M_{mi} is mass of the *i*-th finished product (metal); $M_{м}$ is mass of the whole finished product (all metals); M_{mw} is mass of the waste of metallurgical processing; m_i is mass of the *i*-th component in the balanced reserves; ε_{oe} is the coefficient of ore extraction of mineral; ε_{yi} is the coefficient of removal of the *i-th* lumpy rock from ore; ε_{ci} is the coefficient of extraction of the *i*-th component from ore into the concentrate; ε_{ti} is the coefficient of extraction of the *i*-th component from ore to tails; ε_{mi} is the coefficient of extraction of the *i*-th component from concentrate to metal; ε_{wi} is the coefficient of extraction of the *i*-th component in waste of metallurgical processing; n is the number of all the components in the reserves volume, including waste; r is the number of components extracted from ore; p is the number of minerals extracted from ore to the concentrate; s is the number of components extracted to tails; q is the number of minerals extracted from concentrate to metal; *l* is the number of components in waste of metallurgical processing.

By the wide used technologies of the ore extraction and processing, n>p>q, ε_{oe}=0,5÷0,97 (the low limit corresponds to underground mining, the upper limit – to open-cast mining), ε_{yi}=0,15÷0,4, ε_{ci}=0,4÷0,98, ε_{ti}=0,02÷0,5, ε_{mi}=0,85÷0,98, ε_{wi}=0,02÷0,15.

For the sustainable exploration of the mineral reserves, full and comprehensive use of MR at every stage of its processing it is necessary to perform a certain amount of the survey and measurement works, as well as provide the possibility of application of the advanced and efficient technologies of processing.

For example, during the exploration phase with the use of modern high-precision equipment it is necessary to improve the completeness and reliability of the geological study of the individual blocks of deposits, determine more thoroughly the material composition of the core components as well as of the concomitant useful components, explore more fully the technological properties of ores, limit clearly the volume of the balance reserves. For each deposit it is necessary to approve the list of the useful components, subject to extraction, with indicating the minimum value of the extraction coefficient into the concentrate and metal.

By mining works it is necessary to refine systematically the mineralogical and technological parameters of useful components, provide the most complete extraction from the depths of all the balance reserves, envisage extraction of the over-balanced reserves.

At the stage of enrichment it is necessary to build and use several technological schemes of the ore processing into collective concentrate, which sharply increase the extraction coefficient for each useful component.

In the cycle of metallurgical processing it is necessary to create additional manufactures, non-standard technologies, ensuring maximum extraction of all the useful components, apply the repeat sequential processing of the concentrates.

The Results of the Integrated Use of Useful Components from Raw Ores

Analysis of the mathematical models of the ore production state (1) at each processing stage shows that all the technologies of geological prospecting, mining extraction, mineral processing, chemical and metallurgical industries, providing improvement of quality, increase fullness and completeness of extraction of the core and concomitant components of minerals, consist in increasing of the numbers *n*, aspiration the numbers *p* and *q* to *n*, increase the values of the extraction coefficients ε_{oe}, ε_{yi}, ε_{ci}, ε_{mi} and decrease the values of the extraction coefficients ε_{ti}, ε_{wi}. This conclusion reflects the essence of the technical, technological and organizational solutions to improve the completeness and the complexity level use of ore raw materials (Rakishev, 2010; "Complex processing", 2008; Rau, 2015b).

Let us demonstrate the possibility of implementing a processed ore quality management system on example of the model deposit, developed by the open pit method. It is close in ore composition to the real copper-molybdenum deposits as Aktogay and Bozschakol. The content of useful components in the ore, the mass of the individual components at the ore and the value of 1 ton metal of the same name is shown at table 2. As can be seen from the table 2, the mass of the extracted ore from the quarry field is taken to be 950,000 tons, taking into account ε_{oe}=0,95. Coefficient of the lumpy rock removal from the ore mass is assumed to be ε_y = 0.2. The mass of all the useful components (UC), received at the processing plant, is 6832.514 tons, including the noble and rare earth metals of 40.014 tons, mass of the waste rock (WR) is 754,933.99 tons, all components (AC), including the waste rocks, are 761,366.5 tons.

Table 2 – Mass of the separate components in ore and price of the same metal

Components in ore	Content of components	m_i, t	ε_{oe}	$m_{iи}$, t	ε_y	M_{lr},t	$m_{iф}$,t	c_i, $/t
Copper (Cu)	0,7%	7000	0,95	6650	0	0	6650	$5,5 \cdot 10^3$
Molybdenum (Mo)	0,015%	150	0,95	142,5	0	0	142,5	$27 \cdot 10^3$
Gold (Au)	1,0g/t	1,0	0,95	0,95	0	0	0,95	$34,23 \cdot 10^6$
Silver (Ag)	10 g/t	10	0,95	9,5	0	0	9,5	$0,44 \cdot 10^6$
Bismuth (Bi)	0,0005%	5,0	0,95	4,75	0	0	4,75	$19 \cdot 10^3$
Platinum (Pt)	0,0005%	5,0	0,95	4,75	0	0	4,75	$27,81 \cdot 10^6$
Palladium (Pd)	0,0005%	5,0	0,95	4,75	0	0	4,75	$18,08 \cdot 10^6$
Cobalt (Co)	0,0005%	5,0	0,95	4,75	0	0	4,75	$24 \cdot 10^3$
Selenium (Se)	3,8g/t	3,8	0,95	3,61	0	0	3,61	$21 \cdot 10^3$
Tellur (Te)	2,6g/t	2,6	0,95	2,47	0	0	2,47	$44 \cdot 10^3$
Cadmium (Cd)	0,4g/t	0,4	0,95	0,38	0	0	0,38	$0,1 \cdot 10^6$
Rhenium (Re)	0,3 g/t	0,3	0,95	0,285	0	0	0,285	$1,4 \cdot 10^6$
Indium (In)	2,0 g/t	2,0	0,95	1,9	0	0	1,9	$0,26 \cdot 10^6$
Osmium (Os)	0,02g/t	0,02	0,95	0,019	0	0	0,019	$20 \cdot 10^9$
Thallium (Ti)	0,0002 %	2,0	0,95	1,9	0	0	1,9	$0,7 \cdot 10^6$
Useful components		7192,2	0,95	6832,514	0	0	6832,514	
Waste rocks		992807,8	0,95	943167,486	0,2	188633,49	754533,99	
All components		1000000	0,95	950000		188633,49	761366,5	

Let consider change in the mass of all components, depending on the indices of mineral raw processing. The mass values of specific useful components in the concentrate, the metal, by various values of the coefficients of component extraction from ore in the concentrate, from the concentrate to metal, are shown at the table 3. At the first option, for the core metal, ε_c varies in limits 0,7-0,8, ε_m – in limits 0,7-0,85, for concomitant components ε_c = 0,5-0,6, ε_m = 0,6-0,75. At the second option, for the core metal ε_c=0,8-0,9, ε_m=0,8-0,9, for concomitant components ε_c = 0,6-0,7, ε_m= 0,7-0,85. Masses of the useful components and waste rock in the ore and concentrate at the second option of the processing are adduced at the figure 1.

Table 3 – Mass of the separate component in the concentrate, in the enrichment tails, in the metals and in the waste depending on the processing indices

	ε_c	m_{ki}, t	ε_{ti}	m_{ti}, t	ε_{mi_1}	m_{mi_1}, t	ε_{mi2}	m_{mi2}, t	ε_{wi1}	m_{wi_1}, t	ε_{wi2}	m_{wi2}, t
First option												
Cu	0,8	5320	0,2	1330	0,8	4256	0,85	4522	0,2	1064	0,15	798
Mo	0,7	99,75	0,3	42,75	0,7	69,825	0,75	74,8125	0,3	29,925	0,25	24,9375
Au	0,6	0,57	0,4	0,38	0,7	0,399	0,75	0,4275	0,3	0,171	0,25	0,1425
Ag	0,6	5,7	0,4	3,8	0,7	3,99	0,75	4,275	0,3	1,71	0,25	1,425
Bi	0,5	2,375	0,5	2,375	0,6	1,425	0,7	1,6625	0,4	0,95	0,3	0,71,25
Pt	0,5	2,375	0,5	2,375	0,6	1,425	0,7	1,6625	0,4	0,95	0,3	0,71,25
Pd	0,5	2,375	0,5	2,375	0,6	1,425	0,7	1,6625	0,4	0,95	0,3	0,71,25
Co	0,5	2,375	0,5	2,375	0,6	1,425	0,7	1,6625	0,4	0,95	0,3	0,71,25
Se	0,5	1,805	0,5	1,805	0,6	1,083	0,7	1,2635	0,4	0,722	0,3	0,5415
Te	0,5	1,235	0,5	1,235	0,6	0,741	0,7	0,8645	0,4	0,494	0,3	0,3705
Cd	0,5	0,19	0,5	0,19	0,6	0,114	0,7	0,133	0,4	0,076	0,3	0,057
Re	0,5	0,1425	0,5	0,1425	0,6	0,0855	0,7	0,09975	0,4	0,057	0,3	0,04275
In	0,5	0,95	0,5	0,95	0,6	0,57	0,7	0,665	0,4	0,38	0,3	0,285
Os	0,5	0,0095	0,5	0,0095	0,6	0,0057	0,7	0,00665	0,4	0,0038	0,3	0,00285
Ti	0,5	0,95	0,5	0,95	0,6	0,57	0,7	0,665	0,4	0,38	0,3	0,285
UC		5440,80		1390,6		4339,08		4611,86		1101,72		828,94
WR	0,04	30181,36	0,96	724352,63	0	0	0	0	1,0	30181,36	1,0	30181,36
AC		35622,16		725743,23		4339,08		4611,86		31283,08		30930,30
Second option												
Cu	0,9	5985	0,1	665	0,9	5386,5	0,93	5566,05	0,1	598,5	0,07	418,95
Mo	0,8	114	0,2	28,5	0,8	91,2	0,85	96,9	0,2	22,8	0,15	17,1
Au	0,7	0,665	0,3	0,285	0,8	0,532	0,85	0,56525	0,2	0,133	0,15	0,09975
Ag	0,7	6,65	0,3	2,85	0,8	5,32	0,85	5,6525	0,2	1,33	0,15	0,9975
Bi	0,6	2,85	0,4	1,90	0,75	2,1375	0,8	2,28	0,25	0,7125	0,2	0,57
Pt	0,6	2,85	0,4	1,90	0,75	2,1375	0,8	2,28	0,25	0,7125	0,2	0,57
Pd	0,6	2,85	0,4	1,90	0,75	2,1375	0,8	2,28	0,25	0,7125	0,2	0,57
Co	0,6	2,85	0,4	1,90	0,75	2,1375	0,8	2,28	0,25	0,7125	0,2	0,57
Se	0,6	2,166	0,4	1,444	0,75	1,6245	0,8	1,7328	0,25	0,5415	0,2	0,4332
Te	0,6	1,482	0,4	0,988	0,75	1,1115	0,8	1,1856	0,25	0,3705	0,2	0,2964
Cd	0,6	0,228	0,4	0,152	0,75	0,171	0,8	0,1824	0,25	0,057	0,2	0,0456
Re	0,6	0,171	0,4	0,114	0,75	0,12825	0,8	0,1368	0,25	0,04275	0,2	0,0342
In	0,6	1,14	0,4	0,76	0,75	0,855	0,8	0,912	0,25	0,285	0,2	0,228
Os	0,6	0,0114	0,4	0,0076	0,75	0,00855	0,8	0,00912	0,25	0,00285	0,2	0,00228
Ti	0,6	1,14	0,4	0,76	0,75	0,855	0,8	0,912	0,3	0,342	0,2	0,228
UC		6124,05		708,46		5496,86		5683,36		627,19		440,69
WR	0,03	22636,02	0,97	731897,97	0	0	0	0	1,0	22636,02	1,0	22636,02
AC		28760,07		732606,43		5496,86		5683,36		23263,21		23076,71

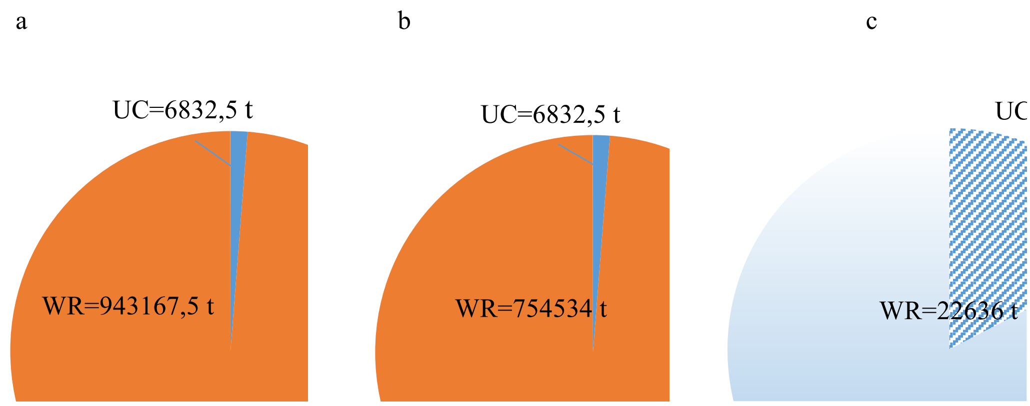

Figure 1 – Masses of useful components and waste rocks in:
the loaded ore (a), ore received at the processing plant (b) and in the concentrate (c)

From the data of the table 3 it follows that with increasing εc from 0.8 to 0.9 the copper mass in the concentrate increased to 5985 from 5320 tons (per one million tons of ore), i.e. on 12.5%. By increasing εc from 0.5 to 0.6 the bismuth mass in concentrate increases from 2.375 to 2.85 m, i.e. on 20%. By increasing εm from 0,6 to 0,75 the platinum mass increases from 142,5 t to 213,75 t, i.e. for 50%. By changing εc from 0,6 to 0,7 the gold mass increases from 0,57 to 0,655 t, i.e. for 16,6%, changing εm from 0,7 to 0,85 increases the gold (metal) mass from 0,399 to 0,5652 t, i.e. for 41,65%, etc.

Data of the table 3 clearly demonstrate the role of technologies of mineral raw processing in raising of the level of the useful components extraction from the raw material, and their high potential. Through the development and implementation of the innovative technologies and technical facilities the current level of extraction of noble and rare metals can be increased in 2 times.

The economic consequences of fuller use of the mineral raw are required careful attention. To their determination it is necessary the values Moe, Mc and Mm from the formula (1) to multiply by the cost of the 1 t of the i-th finished product.

Calculations by the prices for the metals, listed at the table 2, show that in the second option of the ore processing with mass 761366.5 t (see the table 3, the column with the value m_{mi2}) the cost of the obtained copper is 30 613 275, cost of molybdenum is 2 616 300, cost of gold is 19 348 507,5, silver - 2 481 100, bismuth - 43 320, platinum - 63 406 800, palladium - 41 222 400, cobalt - 54 720, selenium - 36 388,8, tellurium - 52 166,4 cadmium - 18 240 , rhenium - 191 520, indium - 237 120, osmium - 182 400 000, thallium - 638 400, cost of the total final production is 343 366 257,7 US dollars. If the cost of core metals (Cu, Mo) is 33 229,575 US dollars, the cost of the noble and rare earth metals is 310 136 682,7 US dollars. Masses and costs of the core and rare metals are plotted at the figure 2.

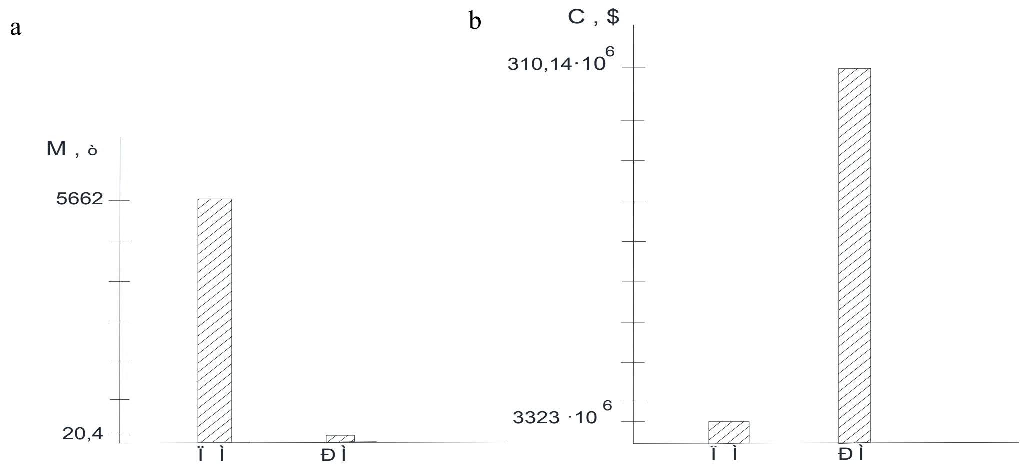

Figure 2 – Masses (a) and costs (b) of the obtained core and rare metals from 950 000 tons of ore

In these conditions, total revenue from the sale of concomitant noble and rare metals exceeds the income from the core metals (copper, molybdenum) in 9.33 times. Income from the possible sale of osmium is 5.5 times higher than the total revenue of the core metals. This example also shows that the current size of the revenue from the sales of MMC production by integrated use of ores can be achieved when the ore amount in at least 8.0-10.0 times less than at present.

For large-scale implementation of the measures to improve the comprehensive utilization of mineral raw materials on the legislative, the state level it is necessary to solve the question of the necessity of extraction all concomitant, especially high-value mineral components from raw material. This naturally requires the construction of additional workshops, industries, tangible investments, to which investors are reluctant. However, the state's interests require the decisive action.

Deep Processing of Primary Products of MMC

The role and place of mining and metallurgical complex in the state economy can be improved also at the expense of deep processing of the obtained product, the creation of high-tech and knowledge-intensive industries and produce higher commodity readiness of products, increase product range, the production of high value-added products, including new types of construction, composite and other materials.

Some experience in this direction has already been accumulated. Thus, during the first five-year plan, such large projects were implemented as launching the second phase of the Kazakhstan electrolysis plant with capacity to 250 thousand tons of primary aluminum per year, the plant for production of 70 thousand tons of cathode copper at the LLP "Kazzinc", a new ferroalloy plant with capacity up to 440 thousand t at the JSC "TNK "Kazchrome" in Aktobe, and also a new refining plant, the LLP "Tau Ken Altyn" with capacity of 25 tons of the refined gold and 50 tons of the silver in Astana (Smirnov, 2015).

On the basis of JSC "Ust-Kamenogorsk titanium and magnesium plant" reconstruction of the forging press PA-1343 was realized for cutting into the layers of sponge titanium. Here also construction of a plant for the production of titanium ingots and slabs was completed.

Within the projects of the Industrialization Map, production was assimilated, which was not produced previously in Kazakhstan, for example, aluminum rod at the JSC "KazEnergoKabel", the steel panel radiators at the LLP "Kazterm", the seamless pipes at LLP «KSP Steel» (Smirnov, 2015).

The National program of industrial and innovative development for 2015-2019 years is focused on diversification and improving the competitiveness of the manufacturing industry.

The steel industry provides production of the high quality raw materials to produce steel (granular iron and hot briquetted iron), production of the new types of steel (pipe, corrosion-resistant, rail and car). By this, annual steel output is expected to increase from 3 million tons to 6 million tons; production of pig iron will be increased from 2.6 million to 3 million tons; production of the pipes of various assortment – from 780 thousand to 1.1 million tons, production of fittings – from 350 thousand to 1,1million tons.

There were planned creation in the Almaty region of pig iron production with capacity of 400 thousand tons per year, and in Almaty – of the pipe plant with annual capacity of 230 thousand pipes, construction of the rail and girder plant in Aktobe with capacity of 200 thousand tons of rails of 120 meters long; organization of production of briquetted iron with capacity of 1.8 million tons in the Kostanai region; development of pipe rolling production in the Pavlodar region with an increase in capacity up to 270 thousand tons of pipes per year (Smirnov, 2015).

In non-ferrous metallurgy it is planned to establish production of titanium slabs of 5,8 thousand tons, commercial ferronickel production with capacity of 40 thousand tons, production of aluminum wheels for passenger cars - up to 360 thousand units per year.

In Almaty region it is planned to build a plant of LLP "Aluminium of Kazakhstan" for production of aluminum profiles in capacity up to 12 ths. tons per year. At the large metallurgical plants the production of the goods from core metals will be provided, in particular wire rod, wire, profiles, foils and commodity nomenclature for related industries.

In 2015 LLP JC «SARECO» produced 67 tons of bulk concentrate of rare-earth metals (REM). JSC Ulba Metallurgical Plant produced 404.05 tons of beryllium, 30.08 tons of tantalum and 4.67 tons of niobium production. In 2016, it is planned to build another plant that will separate the rare-earth oxides and metals from the ore concentrate.

For the production of REM some kinds of coals are suitable (Karazhyra deposit, East Kazakhstan Region), and also natural shungites, phosphates, fluorides, etc. The raw material for extraction of the dispersed rare metals (indium, thallium, selenium, tellurium, germanium, gallium, rhenium) are the products of processing of lead-zinc, copper and aluminum-containing ores, ash from burning coal, as well as solutions and sledges of producing sulfuric acid.

Together, these sources could provide REM in sufficient amounts to Kazakhstan could take its rightful place in the global rare-earth metal market.

CONCLUSIONS

1. The developed mathematical models of the mineral raw state successfully interconnect the indices of MR processing with the final results. They allow to estimate objectively the level of the applied technology of processing and to find ways to improve the extraction of core and concomitant useful components from MS.
2. Technical facilities and technologies adapted to the natural and technological properties of mineral raw provide a high level of extraction of noble and rare metals.
3. Implementation of scientific developments of the Kazakhstan scientists concerning full and comprehensive utilization of mineral resources (including oil, uranium, coal) into practice will significantly improve the efficiency of the natural resources sector and the economy as a whole.
4. An important direction of the MMC diversification is a deep processing of the obtained product, the creation of high-tech and knowledge-intensive industries and obtaining of the products of higher commodity readiness.
5. Effective implementation of diversification will keep a leading role of MMC in the state's economy.

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DNPM'S UAV PROJECT – UAV APPLICATION FOR MONITORING AND INSPECTION OF MINING ACTIVITIES AND DAMS

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DNPM's UAV PROJECT – UAV APPLICATION FOR MONITORING AND SURVEILLANCE OF MINING ACTIVITIES AND DAMS

ABSTRACT

The National Department of Mineral Production (DNPM) developed, in partnership with the University of Brasilia (UnB), an unmanned aerial vehicle (UAV) used in monitoring and surveillance of mining activities. This paper presents the project's results as well as future perspectives for the use of the UAV in the mining industry. A twofold approach was tested, involving both qualitative and quantitative analysis of UAV images and the digital elevation model (DEM). Firstly, based on the qualitative approach, UAV images were employed to identify the mining scenario, including irregular operations. DEM yielded from UAV data were used in volume calculation, providing the information required for cross analysis with those supplied by companies about the extracted ore volume. The data was also useful to monitor discarded tailings in dams. The UAV constitutes an important tool to assist DNPM in their mission to monitor and regulate mining operations.

KEYWORDS

UAV, DNPM, mining, surveillance, volume calculation, dam.

INTRODUCTION

Faced with the need to optimize DNPM's inspection activities, thus increasing efficiency and reducing risks for field inspectors, in 2011, the DNPM entered into partnership with the University of Brasilia for the development of a µUAV and evaluation of its use in the inspection of mining activities. As a result, a plane-type µUAV (figure 1) was developed, tested and evaluated in various types of terrain and mines. Its main features are: 1.90 meters wingspan; 1kg payload, 2.5 kg total weight, and 45 minutes flying range. The µUAV is equipped with a Sony Cyber-Shot small format digital camera, non-metric, 20.2 megapixels, DSC-RX100 model. It includes a VGA video camera for navigation that allows real-time video recording during flight.

In 2013, DNPM's µUAV was granted the Experimental Flight Authorization Certificate (CAVE) by the National Civil Aviation Agency (ANAC), the first UAV belonging to a public civilian institution to be authorized to operate in the Brazilian airspace. Since then, the DNPM has been inserting the products generated by this tool in monitoring routines, including: (i) recognition surveys, which require only qualitative image evaluation without the need for high accuracy (used in the context of identification of mineral exploration, mainly focusing on irregular activities), and (ii) precision quantitative surveys, requiring accurate measurement of the mined area as well as the volume of ore extracted.

This paper aims to demonstrate the capabilities and restrictions in the use of UAVs in different inspection actions by DNPM, based on field tests and evaluation of the accuracy of µUAV-generated products. The article comprises comparisons with other methods for the production of high resolution images and Digital Elevation Models (DEM).

Figure 1 – μUAV developed by UnB into partnership with DNPM. In detail, non-metric camera, model DSC-RX100, used to obtain the air photos.

QUALITATIVE EVALUATION OF DATA GENERATED BY THE UAV

Many of the DNPM inspection actions require only visual observation of the scene, not requiring the generation of orthomosaics of high cartographic precision. Even though precision does not always constitute an essential factor, it is important to assess and understand the magnitude of the errors and the variables involved in aerial survey. By controlling the process, one can plan flights in such a way as to meet the desired objectives regarding the precision and efficiency of inspections. Concepts and tests prepared by the DNPM team to define flight parameters, processing and precision evaluation will be described as follows.

One of the core aspects of flight planning is to ensure that the entire interest area will be imaged, avoiding “holes" in the final mosaic, due to loss in overlap. The presence of strong winds and the μUAV's low weight are frequent causes of instability during flight. Such instability prevents aerial photographs from maintaining the same geometry. The large variation in the camera's view angle often prevents full coverage of the area. In order to avoid this problem, flight planning should allow a minimum lateral overlap of 60%, and longitudinal up to 90%, between the flight lines.

Regarding the processing of aerial photos, it is important to mention that most UAVs are equipped with non-metric cameras, since the size and weight of metric cameras are incompatible with small and medium-sized UAVs. This makes it impossible to obtain prior geometric parameters. Such fact, combined with the constant changes in the camera's view, which changes photo geometry, prevents the generation of orthomosaics by traditional photogrammetry methods.

Thus, for processing UAV images, one need to use a new approach, focused on the algorithms used by the Computer Vision technology, in order to achieve automatic visual understanding based on images. Computer Vision tries to describe the world we see in one or more images, and to reconstruct its properties, such as shape, light, and color distributions.

This new approach requires major information redundancy, in order to generate correlations among the same points in images with different scales, rotation and lighting. Therefore, the large overlap between photos is important not only to allow correct area imaging, but also to provide the amount of information required for the correlation of points and generation of orthomosaics.

When the flight is performed with large overlapping of photos, the high correlation of tie points minimizes the mosaic's internal errors. Thus, the major errors found in orthomosaics generally result from positioning errors of the navigation GPS aboard UAVs, which are easily corrected using a few control points to eliminate the trend.

The elimination of the trend can be accomplished using ground control points (GCP), as demonstrated in the experiments carried out by Silva et al. (2015). The author evaluated the accuracy of an orthomosaic of a pile of limestone tailings obtained by DNPM's µUAV, post-processed in three situations: (i) without support points, using as reference only GPS navigation data aboard the aircraft; (ii) using 5 ground support points; and (iii) using 10 support points. Evaluated of all orthomosaics were based on another 10 pre-marked reference points, georeferenced by geodetic GPS, other than those used for correction of Orthomosaics.

Table 1 shows the results of accuracy evaluation performed in the three Orthomosaics, allowing for weighing the Root Mean Square (RMS) of the planimetry discrepancy, and the value generated by the CE90 standard (circle error 90%), which establishes that ninety percent of residues in the measurements of the coordinates of reference points are below a certain value. All results were considered satisfactory regarding the use of UAVs in DNPM's inspection routines. Not all cases require an accurate and precise product such as Orthomosaic III. The result obtained in Orthomosaic I shows that the greatest discrepancy between the observed coordinates and reference coordinates was only 2,37m, an insignificant error for most DNPM applications.

Table 1 – Accuracy Evaluation of Orthomosaics generated by µUAV.

	Orthomosaic I (No GCP)		**Orthomosaic II (5 GCP)**		**Orthomosaic III (10 GCP)**	
	X	*Y*	*X*	*Y*	*X*	*Y*
Average	-0,4546	-1,9302	-0,0065	0,0319	0,0088	0,0180
Standard Deviation	0,3766	0,2080	0,0590	0,0474	0,0536	0,0424
RMS	2,134		0,083		0,072	
CE90	4,588		0,178		0,155	

Another important aspect concerns the incorporation of new technologies that make use of the data generated by the UAV. Not all inspection actions require aerial photographs. In some cases, the real-time videos obtained are sufficient for decision-making, eliminating the time for processing photographs and, thus, streamlining field inspections. To that effect, the DNPM has been working on adapting videos to use the full motion technology, which allows real-time viewing of extraction activities, on a georeferenced database.

The use of UAVs has several advantages as compared to satellite images or conventional aerophotogrammetry surveys, such as: 1) immediate availability of images; 2) aerial photographs with higher resolution than those provided by today's satellites; 3) ability to obtain images even in the presence of clouds, since the UAV flies at low altitude and; 4) low acquisition and maintenance costs.

The UAVs are being introduced into DNPM's routine activities, mainly in the inspection of small areas. For areas with tens of kilometers, the cost-benefit of using the UAV is less advantageous than that of satellite images. However, as mentioned, real-time availability and weather conditions are sometimes decisive for the choice of the UAV even in large areas.

In inspection activities involving the identification of the mining context and of irregular activities, evaluations are essentially visual (qualitative) and high precision is not required. In these cases, images are processed automatically in the field, without the addition of control points, allowing for immediate notification of violations. Figure 2 shows an example of limestone extraction where, based on orthomosaic generated by UAV, it was possible to catch the illegal miner acting during an inspection.

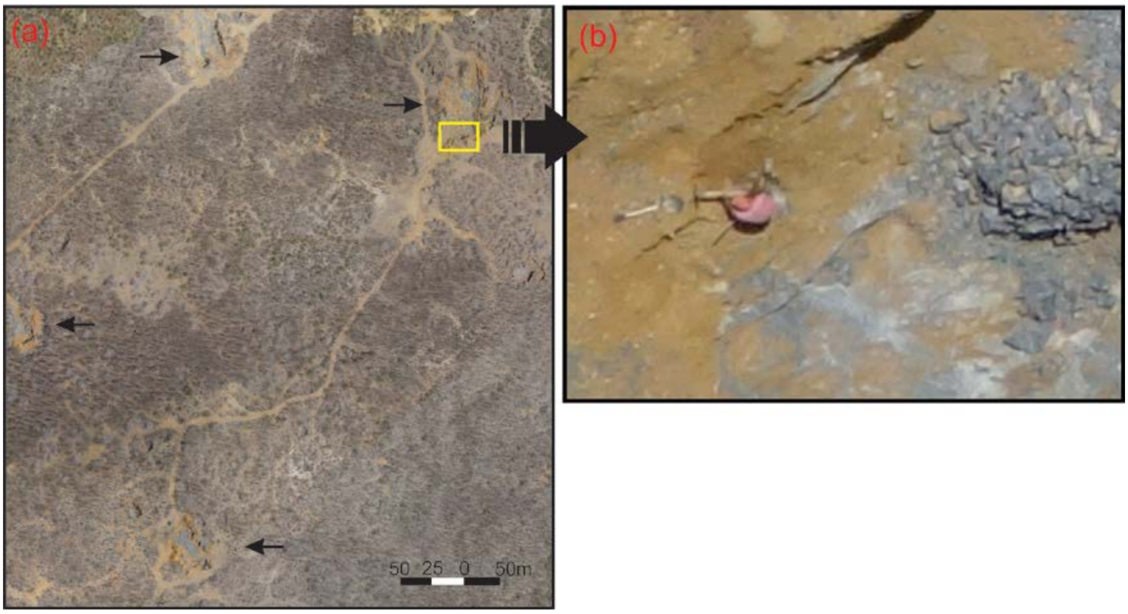


Figure 2 - (a) Orthomosaic generated by UAV with a 5cm resolution. (b) Orthomosaic detail showing an illegal miner

For tailings dam inspections, the qualitative approach is also very important, as it allows the evaluation of the maintenance conditions of the dam structure and risky situations on slopes, in addition to providing an overview of the entire complex, including tailings disposal.

QUANTITATIVE DATA EVALUATION

Even though most DNPM's inspections require essentially visual information, in some cases it is necessary to measure the volume involved in exploration activities. Volume calculations are derived from DEM generated by the UAV, and it is being tested by DNPM to be used in the following contexts: (i) calculation volume for cross analysis with those supplied by companies about the extracted ore volume, so as to verify possible distortions and non-compliances; (ii) monitoring the volume of tailings discharged into dams; (iii) quantification of the volume of material accumulated at each point of the terrain in case of dam break.

Regarding dams, the DEM is also used in the preparation of flood maps, which is essential for defining Emergency Action Plans (EAPs). In these cases, however, the DEM aims to define terrain morphology and the modeling of areas to be flooded, rather than focusing on the volume itself.

The next sections show the results of DNPM research on volume calculation and the possible use of DEM for dam monitoring and inspection, including comparison with other traditional methods.

Examples of Volume Calculation

<u>Volume Calculation of a Pile of Tailings</u>

In order to assess the accuracy of the DEM generated by µUAV and compare with other traditional volume calculation methods, DNPM conducted an experiment in a pile of tailings from "*Pedra Cariri*" limestone, in the state of Ceará (Silva et al., 2016). To that effect, three different methodologies were used: 1) µUAV overflight at 100m height; 2) terrestrial LiDAR scanning; 3) geodetic GNSS survey in kinematic mode. The evaluated accuracy of the DEMs generated by the three different methodologies was based on 10 pre-marked reference points in the tailings pile.

To generate a DEM consistent with the actual pile surface, the point clouds obtained by the three surveys should be generated from measured points that, in terms of quantity and positioning, allow identifying nuances of the uneven surface of the tailings pile without oversampling or undersampling.

As shown in Figure 3, point cloud distribution varies greatly depending on the method used, reflecting directly on DEM's ability to represent the actual shape of the tailings pile surface. One can see that the DEM generated from the μUAV point cloud was the one that best represented pile geometry, detecting nuances of its uneven surface. The DEM generated from LiDAR data only provided good detail of the pile surface near the base where the scan was performed, whereas in areas with lacking information, there was a simplification of the model. Regarding the surface modeled by the GNSS method, there was overall simplification of the geometry on the top of the pile.

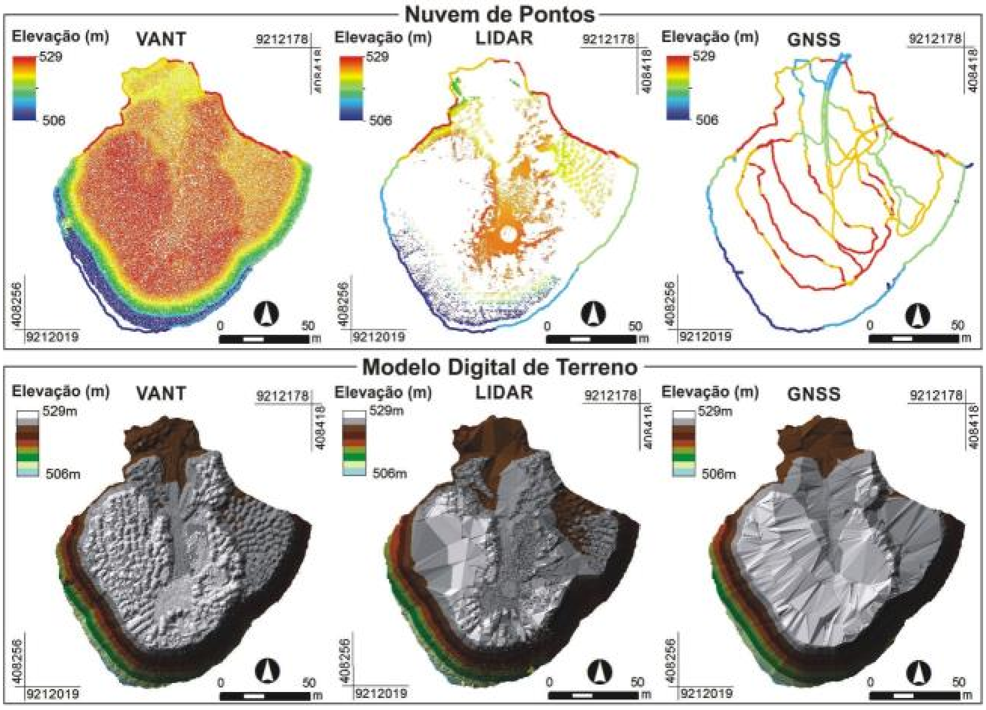


Figure 3 – Comparison of point clouds and DTMs generated by UAV, LIDAR and GNSS

As seen in Figure 4, both LiDAR and GNSS show shading areas, where it is not possible to obtain information on the surface. The enhanced result generated by the μUAV is mainly attributed to the survey's aerial perspective and nadir, which generated a point cloud evenly distributed over the entire study area, including in-between the small clumps of material at the top of the pile.

Regarding accuracy, the DEMs generated by UAV, GNSS and LiDAR data showed precision for vertical equidistance of 0.3m, 1m and 1.3m, respectively. The result was inversely proportional to the volume calculated by the different methodologies, i.e., the lower the DEM accuracy, the greater the volume measured. This is due to the model simplification generated by LiDAR and GNSS, because of absence of information in some points of the tailings pile. As surfaces were simplified, indentations were not accounted for, resulting in increased calculated volume.

Based on research results conducted by Silva et al. (2016), particularly as regards imaging precision and quality as compared to other technologies, UAV-generated DEMs can be used for volume quantification in a consistent manner. Thus, aiming to inspection the royalty collection, it is possible to identify inconsistencies between volumes reported by miners and the actual volume extracted. The same applies to the calculation of indemnities to be paid by mining operations due to illegal ore extraction.

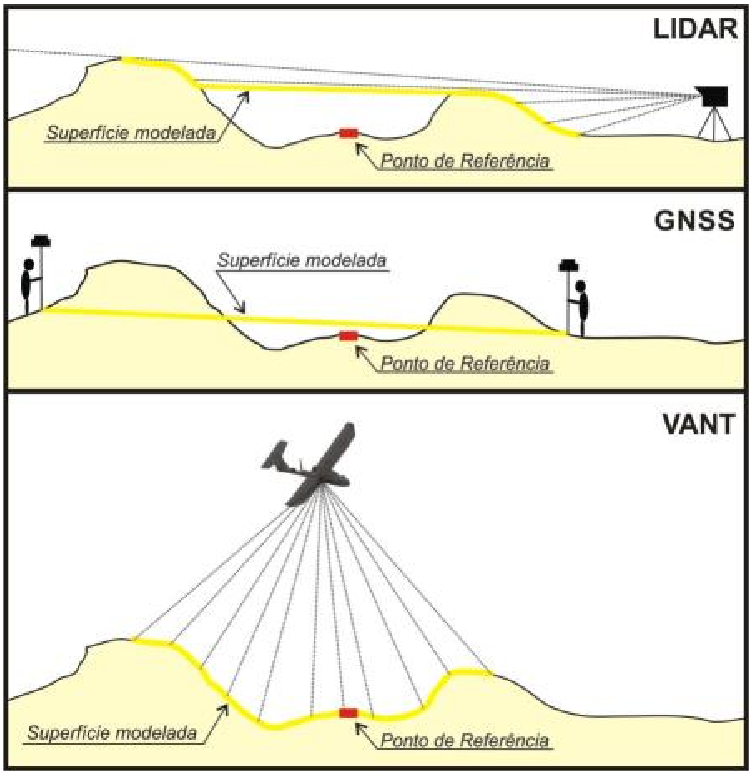


Figure 4 – Schematic representation of the surface modeled by µUAV, LiDAR and GNSS.

Volume Calculation: dam inspection and monitoring rescue and recovery in affected areas.

The use of UAVs allows the generation of DEM in a swift and accurate manner. Thus, it is possible to monitor the volume of tailings deposited in a dam as regularly as considered appropriate by inspection authorities, by simply subtracting DEMs generated at two different moments in time.

DNPM's work in a tailings dam, which failed in 2014, illustrates the advantages of UAV use in monitoring the recovery of affected areas. In the case presented, the company had a DEM generated by a UAV four days after the accident. The DNPM flew over the area two months later. The comparison between the digital models generated by the company and by DNPM defined the accumulated volume of tailings in each land portion.

The figure 5 shows the results of the comparison between the two models. Through the image in shades of gray, all the recovery work carried by the company can be tracked. Light shades represent areas where the amount of material decreased, resulting from tailings removal, while dark shades show areas where material was added to level the areas.

Area imaging and DEM generation can be carried out swiftly. In the above-mentioned case, it took approximately 30 minutes for the UAV to image the entire area affected by the dam failure. In the case of an accident, the comparison between a pre-existing DEM and that generated by the UAV after the break, enables the determination of the volume of tailings deposited over a swamped house, for example, helping to prioritize rescue operations.

It should be highlighted that the UAV is a low-cost technology, affordable to small, medium and large mining companies, and can be flown on a daily basis to control tailings disposal and recovery of areas affected by accidents.

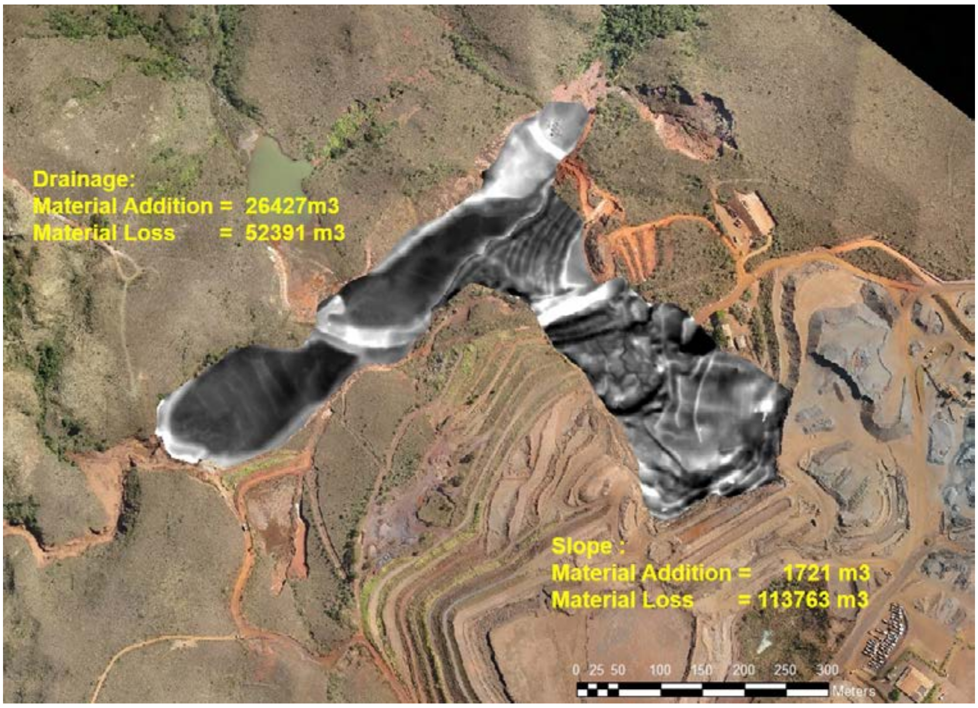
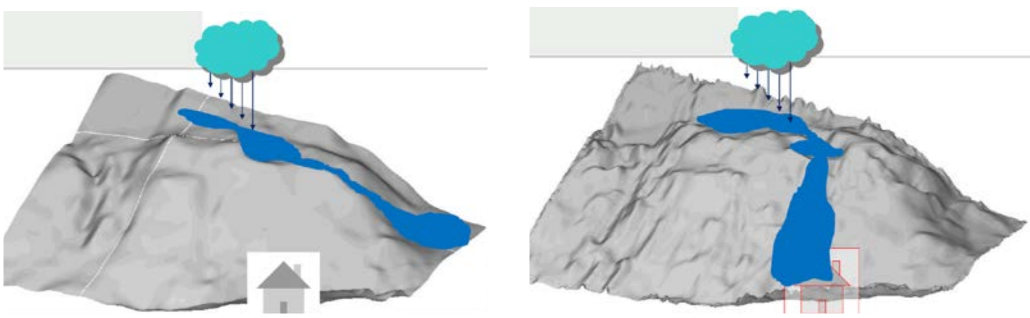


Figure 5- Difference between the DEM generated by DNPM two months after dam break, and the DEM obtained by the mining company four days after the accident . Light colors represent regions where material was added for leveling the areas, while dark colors show areas where tailings were removed. Background VANT Image: Geomil Serviços de Mineração Ltda

Generating DEM for the Preparation of Flood Maps.

The simulations of dam collapse impact for the generation of flood maps, which inform the preparation of Emergency Action Plans (EAPs), most of the time are performed using low-precision DEM. Imprecise data may lead to gross errors in the assessment of affected area, thus, misinforming emergency actions, as show in Figure 6. In the example, one can see the difference between a flood map generated by low resolution Digital Surface Model (DSM) and high resolution Digital Terrain Model (DTM).



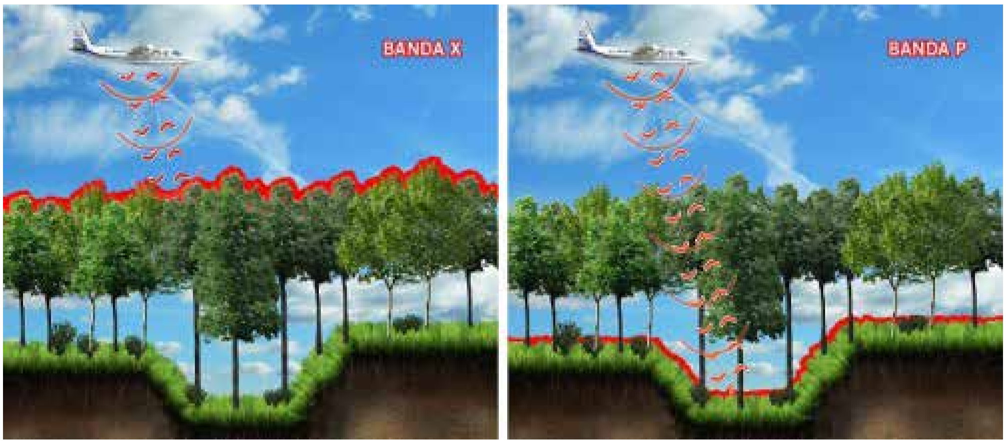
A B

Figure 6 - A- DSM of low resolution (30-90m) and vertical precision (10m). B - DTM of high resolution (2.5 m) and vertical precision (1.6m). Please note the difference in the area to be affected depending on the quality of the model. In these circumstances, the emergency action plan would be totally mistaken. Source: Bradar

After the Samarco dam break in Mariana (MG), companies are seeking insurance for dam operations. The absence of consistent flood maps poses greater difficulty to hiring adequate insurance products, making insurance policies overly expensive or unsuitable to companies needs. In these cases, when accidents occur, the absence of insurance policies and companies' impossibility to bear the financial costs of the accident may lead to additional costs to governments and to society.

Work carried out by DNPM shows that DEMs generated from UAV data meet the needs, in terms of scale and precision, of flood map generation in scarcely vegetated drainage basins. However, the optical sensor used by the UAV / DNPM does not allow data collection below the tree canopy. When there is vegetation, the DEM generated by the UAV represents the Digital Surface Model (DSM), which corresponds to the topography above the tree canopy. Therefore, in the presence of well-developed riparian forest, the Digital Terrain Model (DTM) must be generated, i.e., the representation of relief under the tree canopy. Without DTM generation, fictional models are created, with little adherence to the actual relief, which distorts the final result of the actual area to be affected.

In order to obtain DTMs in densely vegetated areas, along with the UAV, other techniques must be deployed for obtaining the quotas under the tree canopy, such as topographic surveying, laser and P-band radar. Unlike DTMs generated by other methods, those created by P-band radar requires no interpolation, and the result is more precise and consistent with the terrain, as shown in figure 7.



A B

Figure 7 - DSM representing tree canopy relief. Generated by X-band radar or optical images used by the UAV (red line). B - DTM generated below tree canopies by P- Band radar (red line). Please note the lack of adherence to the actual relief in image A. Source: Bradar

The cost of laser and P-band radar surveys in small areas is very significant and can represent an impediment to smaller companies. The same is true regarding "orphan" dams, whose responsibility is transferred to the Union, State and Municipal governments. Thus, even though vegetated areas require field support for the development of interpolated DTMs, the UAV is quite a useful tool to validate Emergency Action Plans.

CONCLUSION

The choice of the product generated by the UAV and the definition of precision depend on the purpose of the inspection:

a) Recognition actions, aimed at identifying the mining scenario, including irregular operations, aim at a swift view of the scene, not requiring data with high cartographic precision. Orthomosaics generated without the inclusion of control points and, in some situations, assessment of the real-time video alone is sufficient for decision-making;

b) Inspection actions involving quantitative analysis, such as the calculation of the volume of ore extracted by miners, and the measurement of the volume of tailings deposited into a dam or accumulated on the ground after dam breaks, require the generation of DEMs with good cartographic precision;

c) Actions aimed at assessing dam Emergency Action Plans require the generation of a DSM with good cartographic precision, allowing the generation of flood maps highly consistent with the terrain.

Sound management of the UAV operation process, from flight planning to the generation of orthomosaics, is essential for the control over accuracy and final errors.

In order to minimize orthomosaics' internal errors and to ensure imaging of the entire area of interest, the DNPM uses as a rule for flight planning a minimum lateral overlap of 60%, and longitudinal overlap up to 90%.

The use of UAVs integrated with Full Motion technology enables observation of the real-time videos directly over the georeferenced database and is a useful tool to streamline decision-making in the field.

The use of UAVs has several advantages as compared to satellite images or conventional aerophotogrammetry surveys, such as: 1) immediate availability of images; 2) aerial photographs with higher resolution than those provided by today's satellites; 3) ability to obtain images even in the presence of clouds, since the UAV flies at low altitude and; 4) low acquisition and maintenance costs.

DNPM's research on volume calculation and accuracy carried out on a tailings pile showed that the UAV-generated DEM produced better results than those obtained through GNSS and LiDAR technologies. The accuracy of the vertical equidistance of DEMs generated by the UAV, GNSS, and LiDAR was 0.3m, 1m, and 1.3m, respectively. The superior result obtained by the µUAV regarding volume calculation is assigned to the survey's aerial perspective and nadir, which generated a point cloud evenly distributed over the entire study area.

Regarding the generation of DEMs for the preparation of flood maps, the UAV shows good results in scarcely vegetated areas. In densely vegetated areas, it is necessary to obtain the DSM, which requires the joint use of other methods, notably, the P-band interferometric radar.

Due to the good results provided by the UAV, DNPM has been introducing the tool in its inspection routine.

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EVALUATION OF BULK ORE SORTING FOR PRE-CONCENTRATION OF COPPER ORE AT PANAUST PHU KHAM OPERATION

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EVALUATION OF BULK ORE SORTING FOR PRE-CONCENTRATION OF COPPER ORE AT PANAUST PHU KHAM OPERATION

ABSTRACT

As mineral deposits become more complex, difficult to process and contain lower grades, extraction of resources require costly mining and processing of increasingly larger volumes of rock per tonne of product. The aim of pre-concentration is to remove barren material as early in the mining process and at as coarse a particle size as possible. This has the main objective of increasing production of metal and reducing processing costs and consequently improving profitability. In addition, pre-concentration may significantly reduce energy consumption, water losses and greenhouse gas emissions per tonne of product. It is cheaper and more eco-efficient to separate and reject the below cut-off grade material earlier in the process, rather than crushing, grinding and separating it through the processing plant. The sorter acts as a gatekeeper – only above cut-off grade material (value-adding) reports to the processing plant; no waste dilutes the feed. Several technologies may be applicable for pre-concentration, such as screening, gravity concentration and magnetic separation. In this paper, we look specifically at sensor based bulk ore sorting - the separation of gangue from a fully loaded conveyor belt based on the grade as measured or inferred from a sensor. The technical and economic viability of bulk ore sorting has been evaluated for the PanAust Phu Kham copper deposit and operation in Lao PDR. Ore sorting technology is described briefly along with discussion of appropriate sensor and diversion systems for the Phu Kham deposit and the economic implications have been evaluated. In the absence of sampling data, ore grade variability was estimated using geostatistical tools based on mine grade control data. The sorter performance was simulated to allow economic evaluations to be conducted for different scenarios. This demonstrated that bulk ore sorting has the potential to increase production and annual inferred cash flow for the Phu Kham operation.

KEYWORDS

Bulk ore sorting, pre-concentration, copper, resource efficiency, sensors

INTRODUCTION

Phu Kham is a large open-pit copper-gold operation in Laos. The geology of the deposit is extremely heterogeneous which suggests it may be amenable to pre-concentration with bulk ore sorting. In particular, later in the mine life the ore will become harder reducing the throughput that can be achieved with the existing processing plant. Bulk ore sorting may offer an opportunity to upgrade the feed prior to the capacity constraint at the processing plant and increase production. This should have further benefits of decreasing operating costs, energy and water consumption per tonne of product due to the reduction of the amount of material needing downstream processing and handling.

In this paper we discuss the potential application of bulk ore sorting for pre-concentration at Phu Kham. An economic assessment has been conducted to evaluate if the grade variability in the sorter feed and the upgrade achieved by the sorter is likely to justify the added costs associated with ore sorting.

THE PHU KHAM OPERATION

The Phu Kham operation consists of a large open-pit copper-gold mine and processing plant located approximately 100 km northeast of the Laos capital Vientiane. Phu Kham geology is extremely heterogeneous due to weathering, alteration, faulting and folding. The deposit consists of complex horizons of copper-gold stockwork and skarn mineralisation. Such variability affects plant throughput and metallurgical performance.

The operation comprises a large conventional open-pit mine feeding ore to a process plant consisting of crushing, grinding and flotation to recover copper and precious metals. The copper is predominantly contained in chalcopyrite, but other copper minerals such as chalcocite, covellite, bornite and tennantite are also present. Expansion and optimisation projects have increased the maximum design capacity of the processing plant from 14 million tonnes per annum (Mtpa) to up to 19.5 Mtpa for most of the remaining mine life (Bennett *et al*, 2014).

Crushing is performed in a single stage with a gyratory crusher. The grinding circuit consists of a SAG mill and two parallel ball mills each in closed circuit with hydrocyclones. Following this, the flotation circuit, comprised of roughers, regrinding and several cleaning stages, produces a copper-gold concentrate containing 22 to 25 % copper, 7 grams per tonne (g/t) gold and 60 g/t silver.

ORE SORTING TECHNOLOGY

Current Technology (Individual Particle Sorters)

Ore sorting is based on the measurement of a property that is different in the valuable and waste components using some form of sensor. A variety of sensors are available; at present, those most commonly used in industrial applications include photometric, electromagnetic, radiometric and x-ray.

The currently available ore sorting technology involves the measurement and separation of individual particles. It is well documented in the literature (for example, Manouchehri, 2004; Wotruba, 2006; and Bergmann, 2009) and in publications by suppliers. Careful feed preparation is required so that individual particles can be detected and measured, and ejection of single particles is usually achieved by blasts of compressed air. Consequently, current sorters have very low capacity (up to 300 tph for larger particles and much less for smaller particles). They would not be economically viable or practical for high tonnage pre-concentration; for this type of application, sorting needs to be applied to bulk quantities of ore such as a loaded truck tray or a fully loaded conveyor belt.

Bulk Ore Sorting

The concept of bulk ore sorting involves the separation of material on a fully loaded conveyor belt based on the grade as measured or inferred from a sensor measurement. It requires a fast and penetrative **sensor** or combination of sensors that measure a property that is different in the valuable and waste components. It also requires a **control system**, to interpret the data from the sensor or sensors and make an accept or reject decision, and a **separation system** such as a diverter gate to separate the valuable "batches" of ore from waste, as shown in Figure 1 (Duffy et al., 2015):

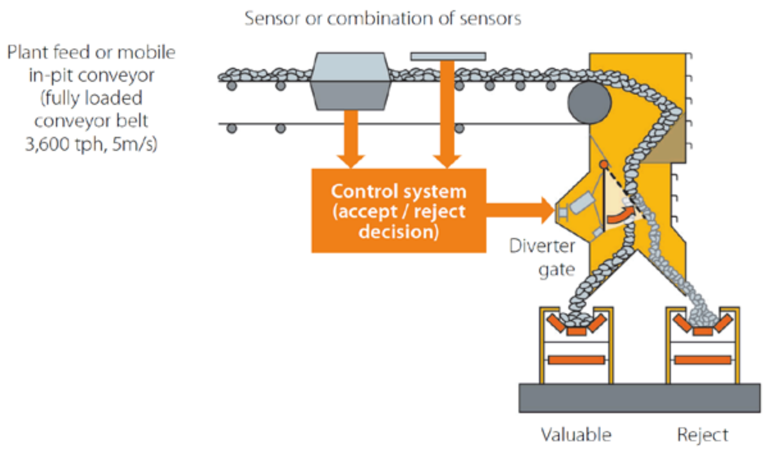


Figure 1 - Bulk ore sorting concept (Duffy et al., 2015)

Sensor Technologies for Bulk Ore Sorting

A review of existing sensor technologies indicates that most are currently not suitable for bulk ore sorting, as they are either not sufficiently penetrating or are too slow for effective separation. For example, laser-induced breakdown spectroscopy (LIBS), laser-induced fluorescence (LIF) and photometric sensors are surface only measures (not penetrating into the rock). X-ray fluorescence (XRF) has a beam size and penetration of only a few millimetres. Therefore, these sensors cannot provide a representative measure for large quantities of heterogeneous material such as required for bulk ore sorting.

Developments in X-ray Transmission (XRT) sensors allow more accurate discrimination of effective atomic density. These are applicable for dry coarse coal separation and possibly also base metals and iron ore, but penetration rates will suffer due to the higher density. While good correlation with grade may not be achieved for base metals (such as copper), the sensor may be able to detect barren material that does not contain the heavier sulphide minerals, and this may be sufficient for sorting applications.

Prompt-gamma neutron activation analysis (PGNAA) and pulsed fast and thermal neutron activation (PFTNA) sensors measure elements and can penetrate the full cross section on a loaded conveyor belt. However, currently the measurement speeds are too slow for effective bulk ore sorting, in the order of minutes rather than seconds. It may be possible adapt these sensors for bulk sorting; the trade-off would be some reduction in accuracy and increased cost.

The CSIRO has developed a sensor using magnetic resonance (MR) that has the ability to rapidly measure batches of ore on large primary production conveyors (Miljak, 2011). It is well suited to a bulk ore sorting application as it is penetrative, can measure large throughputs on fully loaded conveyor belts and the measurement time is rapid. However, the MR sensor measures an individual mineral (not element) and may have limitations measuring ores with complex mineralogy. The sensor is currently developed for chalcopyrite, a dominant copper mineral, and with further development could potentially be applied to other minerals (Heselev, 2012). The MR sensor may be suitable for application at Phu Kham as chalcopyrite is the predominant copper mineral.

The most appropriate sensor for any application will depend on the ore properties. In some cases more than one type of sensor may be incorporated to overcome the limitations of the different sensor types.

ECONOMIC EVALUATION

To evaluate the potential impact of bulk ore sorting on production, costs and inferred cash flow at Phu Kham, the actual production and cost data from 2014 was taken as the base case. The potential impact of the bulk ore sorter was determined using the following methodology:

1. The grade variability in the feed to the sorter was estimated using geostatistical tools.
2. The sorter separation was simulated based on the predicted feed variability to determine the new processing plant feed (tonnes and grade).
3. Economic calculations were conducted for several different scenarios considering additional costs for sorter operation and sorter reject handling.

Each of these steps is described in more detail in the following sections.

Estimating Ore Variability in Sorter Feed

To be amenable to pre-concentration by bulk ore sorting the sorter feed needs to have sufficient grade variability occurring in large enough batches for effective separation. This depends on the in-situ variability of the ore deposit and also the mining methods, ore handling, crushing and blending prior to delivery to the sorter. Understanding the grade variation is important to determine the required response time for measurement and separation and also the potential grade uplift which will determine if bulk ore sorting is economically viable. However, it is very difficult and costly to collect a large number of representative samples of ROM or primary crushed material. Therefore, geo-statistical tools were used to estimate grade variability in the feed to the ore sorter for this preliminary analysis.

Actual grade data from the 2014 grade control model (10 x 10 x 10 m blocks of approximately 2700 tonnes) were used for the analysis. The variation occurring within the Grade Control Model blocks (i.e. for smaller block sizes) was estimated using dispersion variance from variograms provided by Phu Kham. The dispersion variance, D^2, (or the variance of true block values) increases for smaller block sizes as the variability is averaged (or smoothed) when considering larger blocks, as illustrated in Figure 2.

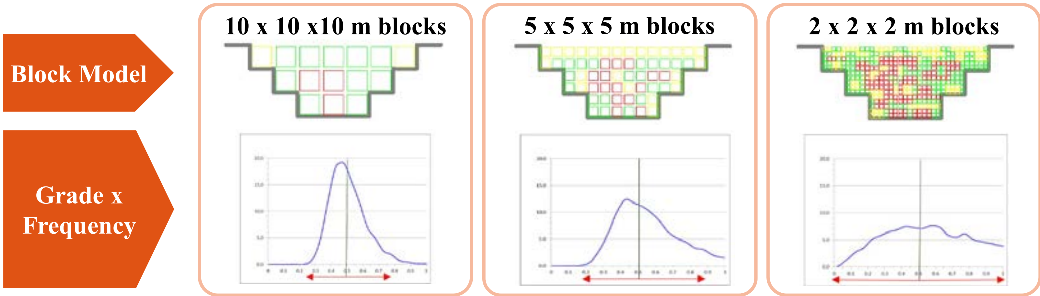


Figure 2 - Dispersion variance at different block sizes

Dispersion variance may be calculated from the semivariogram, ϒ, and the geometry of the block and region in which the blocks are situated. The dispersion variance of blocks, v, inside a bigger region, V, is the difference between the average semivariogram values calculated on V and on v (Brooker, 1991). Thus, the variance of a variable for smaller blocks, v, can be calculated using Equations 1 and 2.

$$D^2(v,V) = \Upsilon(V) - \Upsilon(v) \qquad (1)$$

$$\overline{\Upsilon}(v,V)= \frac{1}{n} \iiint \gamma(h)\ dV \qquad (2)$$

This approach was used to estimate the grade variance for 5 x 5 x 5 m blocks based on the data from the 10 x 10 x 10 m blocks in the 2014 grade control model. The block size selected (5 x 5 x 5 m) equates to approximately 340 tonnes of ore. The sorter will measure and separate much smaller batches of material; a 20 second sensor integration (reading) time equates to about 14 tonnes of material at Phu Kham. However, some mixing will occur through the mining, handling and primary crushing which means the feed to the sorter will be more homogenous than the in-situ material. Therefore, the 5 x 5 x 5 m block size was selected to be conservative and not over estimate the upgrade that may be achieved through sorting.

Sorter Separation Simulation

The dispersion variance was used to estimate the tonnes and grade recovered to the ‘accept’ stream by the sorter for the expected feed grade variation and specified cut-off grade. The ‘accept’ stream from the sorter becomes the new feed to the processing plant. An example result from one of the simulations is shown in Figure 3.

An ore sorter availability of 90% is assumed, and the results incorporate misdirected material during movement of the diverter gate. The time allowed for the gate to change position is three seconds; although, this may be decreased (reducing misplacement) with further optimization of the diverter design. The simulation was conducted using the expected sorter feed estimated from the geostatistics as described previously. The result from the simulation was the breakdown of tonnes and grade to the sorter ‘accept’ (i.e., new processing plant feed) and ‘reject’ streams for each scenario.

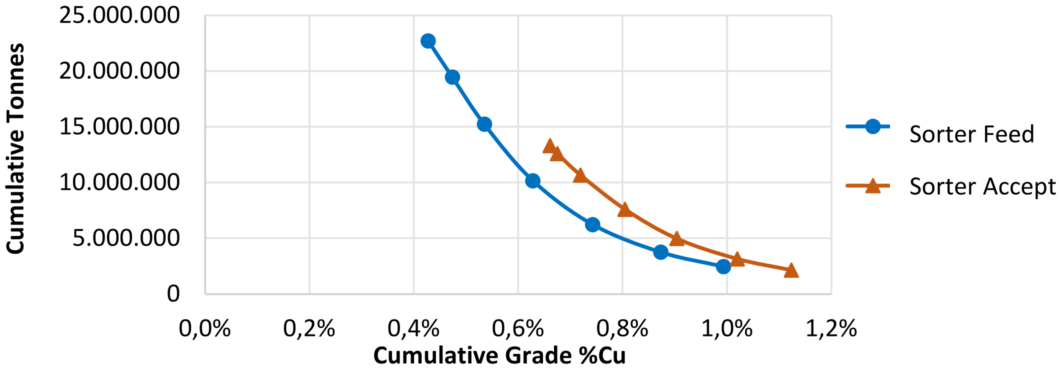


Figure 3 - Sorter separation simulation results

Economic Comparison

An economic comparison was conducted to evaluate the overall impact of competing factors such as increased mining rate, higher plant feed grade (lower processing cost per tonne of product and increased production rate) and additional operating costs associated with sorting and reject handling. The assumptions for this analysis were as follows:

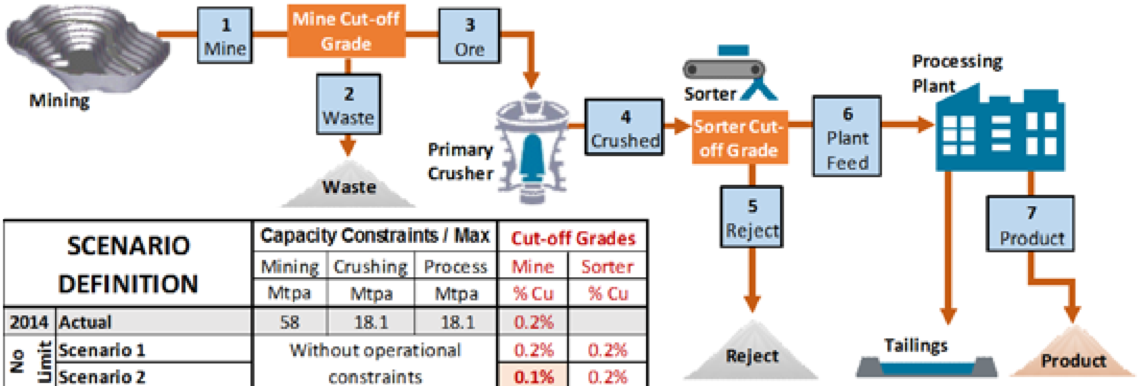
- Mining, processing and overhead costs as per 2014 per tonne costs.
- Costs for the increased mining rate were calculated at the same cost per tonne as 2014.
- Additional operating costs were included for operation of the ore sorter ($0.4 per tonne sorted) and for removal of the material rejected by the sorter ($1.6 per tonne rejected).
- Ore sorter availability of 90%.
- Ore sorter integration (reading) time of 20 seconds (equivalent to 14 tonnes).
- Processing plant recovery was calculated as a function of feed grade based on regression analysis of Phu Kham plant data from 2014.
- Copper selling price USD $5,500.00/t (Copper price USD $6,000.00/t (May 2015) – USD $500.00/t smelting charge).

A number of scenarios were evaluated covering the following situations: **no limitations** to mining and crushing capacity, **current limitations** (considering mining and crushing operational constraints) and **SAG mill limiting conditions** (expected later in the mine life). The scenarios are defined and the mass balance for each is shown in Figure 4. The results for all scenarios are summarised graphically in Figure 5.

<u>Discussion of Results</u>

The first two scenarios considered what may be possible if there were **no limitations** to mining and primary crushing rate but the processing plant throughput was kept the same as that achieved in 2014 (18.1 Mtpa). These scenarios have been compared to the 2014 actual results.

Scenario 1 considers both the mine and sorter operating at the current cut-off grade of 0.2% Cu. The ore sorter applies the cut-off grade decision to a smaller batch size than in the mine. Thus, below cut-off grade material (due to planned and unplanned dilution and variation on a smaller scale than mine block size) is removed from the plant feed. A higher mining rate is required to maintain the same plant throughput due to the rejected material. However, it delivers a significantly higher plant feed grade (33% increase) and consequently increased metal production. Inferred cash flow is increased by more than 60% due to the increased production and the fact that the cost of ore sorting and reject handling is less than treating below cut-off grade material through the processing plant. However, a 37% increase in mining rate is required, and both the required mining rate and primary crushing rate exceed the current capability of the operation.



SCENARIO DEFINITION		Capacity Constraints / Max Mining Mtpa	Crushing Mtpa	Process Mtpa	Cut-off Grades Mine % Cu	Sorter % Cu
2014	Actual	58	18.1	18.1	0.2%	
No Limit	Scenario 1	Without operational constraints			0.2%	0.2%
No Limit	Scenario 2				0.1%	0.2%
Current Limits	Current Base Case	58	19	19	0.2%	
Current Limits	Scenario 3	58	19	19	0.2%	0.2%
Current Limits	Scenario 4	58	19	19	0.2%	0.1%
SAG Limited	SAG Limited Base Case	58	19	16.5	0.2%	
SAG Limited	Scenario 5	58	19	16.5	0.2%	0.2%
SAG Limited	Scenario 6	58	19	16.5	0.2%	0.1%

MASS BALANCE		1 Mined	2 Waste	3 Ore		4 Crushed	5 Sorter Reject		6 Plant Feed		7 Product	Change from Base Case Metal in Product	Inferred cash
		Mtpa	Mtpa	Mpta	% Cu	Mtpa	Mtpa	% Cu	Mtpa	% Cu	Ktpa Cu		
2014	Actual	52	34	18.1	0.46	18.1			18.1	0.46	63		
No Limit	Scenario 1	72	47	24.7	0.47	24.7	6.7	0.10	18.1	0.61	89	40%	62%
No Limit	Scenario 2	66	39	26.5	0.43	26.5	8.4	0.10	18.1	0.58	84	32%	48%
Current Limits	Current Base Case	55	36	19	0.47	19			19.0	0.47	70		
Current Limits	Scenario 3	55	36	19	0.47	19	5.1	0.10	13.9	0.61	68	-2%	5%
Current Limits	Scenario 4	55	36	19	0.48	19	3.8	0.08	15.2	0.57	70	0%	6%
SAG Limited	SAG Limited Base Case	48	31	16.5	0.47	16.5			16.5	0.47	60		
SAG Limited	Scenario 5	55	36	19	0.47	19	5.1	0.10	13.9	0.61	68	13%	21%
SAG Limited	Scenario 6	55	36	19	0.48	19	3.8	0.08	15.2	0.57	70	15%	23%

Figure 4 – Scenario Definition and Mass Balance

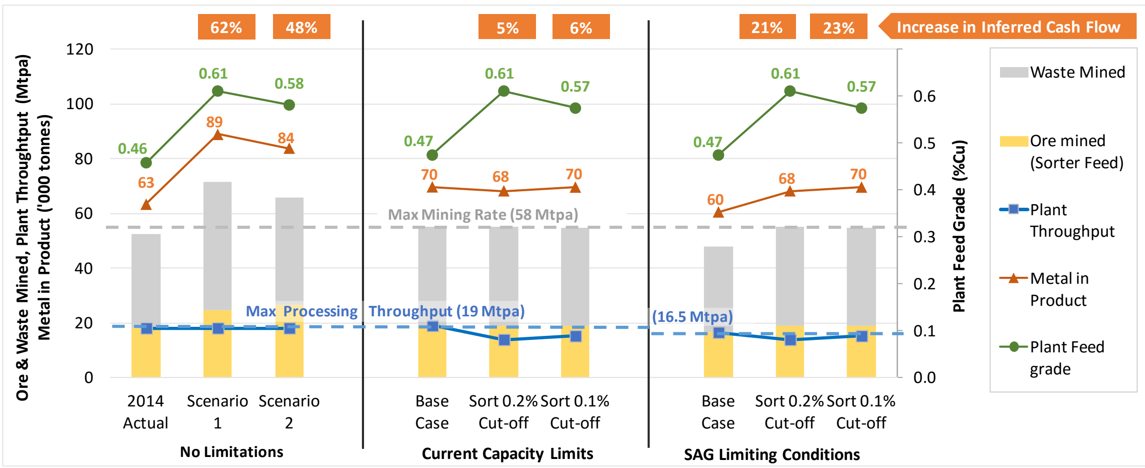


Figure 5 - Summary of Scenario Outcomes

Scenario 2 considers lowering the mine cut-off grade (0.1% Cu) but operating the sorter at the current cut-off grade (0.2% Cu) as required for the plant feed. This scenario recovers valuable mineral from sub-economic material that would have been discarded as mine waste. It also removes below cut-off grade material from the plant feed (as described for Scenario 1). The overall result is a higher plant feed grade (27% increase) and metal production than the base case (but lower than Scenario 1) while also recovering valuable mineral from mine waste. Inferred cash flow is increased by almost 50%, and this scenario requires a less significant increase in mining rate (25% increase) than Scenario 1. However, both the mining rate and primary crushing rate still exceed the current capacity of the operation.

Both Scenario 1 and 2 indicate significantly increased metal contained in product and cash flow compared to the base case. However, they exceed current operational constraints at Phu Kham. Therefore, further scenarios (3 and 4) were evaluated considering the **current operational constraints** as follows: maximum mining rate of 58 Mtpa, maximum primary crushing and sorting rate of 19 Mtpa, maximum concentrator throughput of 19 Mtpa, and maximum copper tonnes in product of 87 ktpa. These constraints allow an increased plant throughput compared to the 2014 actual throughput of 18.1 Mtpa. Therefore, scenarios 3 and 4 are compared to a new base case with plant throughput of 19 Mtpa so that any estimated benefits are associated with the effect of sorting and not the increase in processing plant throughput.

Scenario 3 considers the current constraints at Phu Kham with both the mine and sorter operating at a cut-off grade of 0.2% Cu. The primary crushing capacity is the same as the processing plant capacity. The feed to the sorter must be primary crushed; therefore, the sorter cannot treat more tonnes than the processing plant, and as a result cannot increase metal production rate. The rejection of material by the sorter decreases the processing plant throughput. Slightly less (2%) metal is contained in product compared to the base case (due to sorter losses) but the processing plant treats 27% less material which significantly reduces costs and energy consumption. The overall result is a 5% increase in cash flow due to the reduction in processing costs which outweighs the sorter operating costs and small production loss.

Scenario 4 considers the current operational constraints with the mine operating at a cut-off grade of 0.2% Cu and the sorter operating at a lower cut-off grade of 0.1% Cu. The logic for this scenario is to reduce sorter losses while still removing the lowest grade material from the plant feed. Given the primary crusher constraint, this scenario provides a balance between reducing processing plant throughput (and costs) while maintaining production rate. The net result is very similar metal production than the base case but with 20% less material treated in the processing plant. The overall result is a 6% increase in cash flow.

Throughput modelling and forecasting indicates that later in the mine life Phu Kham will become SAG mill limited rather than primary crusher limited due to changes in ore hardness (Bennett et al; 2014). Therefore, Scenarios 5 and 6 consider **SAG mill limiting conditions**; these are the same as the current operational constraints defined previously, but with processing plant throughput limited to 16.5 Mtpa. In this case, the potential benefits from bulk ore sorting would be much greater. This is due to the fact that the primary crusher, and hence sorter, can treat more tonnes than the processing plant and as a result can increase production compared to the base case. The base case for these scenarios limits the processing plant throughput to 16.5 Mtpa (due to SAG mill limitations).

Scenario 5 considers the SAG mill limiting conditions with both the mine and sorter operating at a cut-off grade of 0.2% Cu. The primary crusher capacity is not sufficient to maintain the maximum processing plant throughput of 16.5 Mtpa in this scenario. However, the bulk ore sorter achieves a combination of increased metal in processing plant feed (due to upgrading) and thus increased production (13% increase in metal in product) and reduced processing plant throughput (16% reduction) and thus operating costs. The overall result is a 21% increase in inferred cash flow.

Scenario 6 considers the SAG mill limiting conditions with the mine operating at a cut-off grade of 0.2% Cu and the sorter operating at a lower cut-off grade of 0.1% Cu. Similarly to Scenario 4, the logic for this scenario is to maximise the benefits from the sorter by removing only the lowest grade material from the plant feed (because there is sufficient plant capacity) in order to minimise sorter losses and maximise production. The net result of this scenario is a 15% increase in metal in product with an 8% reduction in processing plant throughput. The overall result is a 23% increase in inferred cash flow.

As demonstrated by these scenarios, it is cheaper in this case to sort and reject barren material contained in the plant feed than to treat this material through the processing plant. This economic analysis has indicated a small potential benefit with the current operational constraints at Phu Kham and a greater benefit when the operation becomes SAG mill limited later in the mine life. Therefore, as a first step, it is recommended to install a sensor. This would enable measurement of the variation in the sorter feed to confirm the upgrade potential of the ore at Phu Kham, and would also facilitate validation of the sensor accuracy. If the testing confirms the upgrade potential at Phu Kham a diverter and sorting solution could be implemented for when the operation becomes SAG limited or earlier if the variation is sufficient.

PRACTICAL IMPLICATIONS OF BULK ORE SORTING

The mining industry typically takes a ‘one size fits all’ approach. The extraction process is generally designed for the average ore and a blending strategy is used to deliver, as far as possible with the ore available, a stable feed to the plant. It is not unusual to have a number of different grade classifications on the ROM pad which can be fed to the process in the required proportions to achieve a desired feed grade. However, to maximise the benefit of pre-concentration, mining methods need to exploit the natural variation rather than blend it out (Duffy et al., 2015). This will require a shift in mining practices.

Furthermore, bulk ore sorting could reduce dilution and ore loss in mining operations by improving grade control. Dilution can have a very significant impact on the profitability of mining operations, as it decreases the plant feed grade. Operating costs are increased, as a higher tonnage is treated by the processing plant per tonne of product and the processing plant capacity is effectively decreased, prolonging the mine life. Additionally, ore losses occur when ore is misclassified as waste resulting in lost revenue. Measuring grade with the sorter will allow correct allocation of ore and waste improving execution of the theoretical cut-off grade. This may also reduce (but not eliminate) the quality control requirements at bench level. This has potential to increase mine productivity and reduce costs, particularly in deposits with complex mine geology and when mining near ore-waste boundaries.

A bulk ore sorter for pre-concentration could be located anywhere from the mining face to the grinding circuit feed. In general, earlier in the process is better; to exploit the natural heterogeneity of deposits and maximise the benefits. However, in most cases, bulk ore sorting would need to be implemented after primary crushing to present material at a size that can be handled by the sorter (conveyor, sensor and diverter), but any additional crushing or material handling should be avoided if possible. The best location will be affected by factors such as mine type and mining methods; location of waste dumps and processing plant; operating costs of the mine, sorter and downstream processes; impact on productivity in the mine and plant; changes to layout; space required; environmental impact; and whether it is a Greenfield operation or retrofit to an existing operation (Duffy et al., 2015).

Pre-concentration effectively upgrades the plant feed; less tonnes of ore are treated in the processing plant per tonne of product, thus reducing the costs, energy and water consumption per tonne of product. Additionally, gangue minerals are generally harder than valuable minerals, and rejecting the gangue could reduce the hardness of the plant feed and further increase the throughput and reduce energy consumption of the comminution circuits. The increased plant feed grade should also deliver improved grade-recovery performance in separation processes such as flotation. For the scenarios evaluated, bulk ore sorting could increase Phu Kham feed grade by 0.10 to 0.14% Cu (absolute). Based on regression analysis of Phu Kham historical data this increase in plant feed grade should provide an increase in flotation plant recovery of approximately 2 to 3% (absolute).

CONCLUSION

The economic evaluation suggests that bulk ore sorting at Phu Kham has the potential to increase the amount of metal in product and/or reduce processing plant throughput (and costs) to improve the annual inferred cash flow for the conditions investigated. This is because it is cheaper to sort and reject barren material contained in the plant feed than to treat this material through the processing plant.

If there were no limitations on mining and crushing rate at Phu Kham the annual metal production may be increased by 30 to 40% and inferred cash flow by 50 to 60%, but the required increase in mining and crushing rate is not viable. With the current operational constraints bulk ore sorting would deliver little change in the annual metal production; however, inferred cash flow may be increased by about 5% due to a reduction in processing plant throughput and operating costs. When the operation becomes SAG mill limited (rather than primary crusher limited) later in the mine life, the annual metal production may be increased by about 10 to 15% and inferred cash flow by more than 20%. In this case, production can be increased because barren material is rejected by the sorter prior to the bottleneck (the SAG mill).

As most ore deposits are heterogeneous, it is expected that in many cases there would be sufficient grade variation to make upgrading by bulk ore sorting possible if implemented early in the process. Certainly for Phu Kham, where the geology is complex and highly heterogeneous, geostatistical analysis has indicated that bulk ore sorting should provide sufficient upgrade to be economically viable.

A bulk ore sorter acts as a gatekeeper; only above cut-off grade material (i.e., that which adds value) reports to the plant; no waste dilutes the plant feed. Likewise, no above cut-off grade material is lost to waste; preventing lost revenue. Thus, bulk ore sorting achieves the cut-off grade with greater accuracy and optimises the extraction of the resource. This is of growing importance as mineral deposits become more complex, difficult to process and contain lower grades. Increasingly larger volumes of rock per tonne of product need to be mined and processed. However, pre-concentration reduces the amount of material requiring downstream processing and handling and has the potential to reduce processing costs, energy and water consumption, increase production, and improve resource efficiency.

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EXPERIENCES IN RECOGNITION OF GASO-GEODYNAMIC ZONES IN THE ROCK MASS OF A COPPER ORE MINE USING GEOPHYSICAL METHODS

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ABSTRACT

In this paper, geophysical tests (seismic tomography and borehole ground penetrating radar [BGPR]) along with geological engineering tests are employed to identify the zones exposed to the threat of gas and rock outburst in the rock mass of the Rudna copper ore mine in Poland. A comprehensive description of geological conditions is presented, with particular focus on the circumstances behind the origin of gas traps in the rock mass of the Rudna mine. The methodology for detecting zones of gaso-geodynamic threat consisted of three stages. In the first stage, a part of the deposit presumed to be threatened by outbursts of gases and rocks was identified on the basis of geological investigation conducted with the use of preparatory drifts. It was assumed that the fundamental and possibly useful information would be generated in the assessment of the alterations of elastic properties of a rock mass, obtained by means of seismic tomography. In the second stage, the results of the seismic tomography were used to design and make control boreholes in the zone of the seismic anomalies, for the purpose of a detailed investigation of the structure and properties of the rock mass. In the third stage, the BGPR was used to obtain detailed geological engineering information. Results showed a very strong correlation between the location of the zone containing gases and water and the anomalous S-wave velocity and field of Young's modulus in the dolomite layer and, to a lesser extent, other seismic parameters under the considered geological and mining conditions. The advantages and limitations of the applied geophysical methods have been highlighted for the use in the identification of gaso-geodynamic zones.

KEYWORDS

Copper ore mining, gases and rocks outburst, seismic tomography, borehole ground penetrating radar (BGPR)

INTRODUCTION

In Poland, since the 1990s, the threat of gas and rock outbursts has significantly diminished, owing to the limitation of the mining industry and the closing down of numerous mines, in particular, the hard coal mines in Lower Silesia (Pilecki, Laskowski, Hryciuk, Pilecka, Czarny, Wróbel, Koziarz, & Krawiec, 2013). Outbursts (82% of incidents) have occurred mostly during the drilling of preparatory excavations. The threat of gas and rock outburst in the hard coal mines was related to the tectonic disturbance zones, as well as to the low compactness of coal in the outburst zone, accompanied by a high level of CH_4 emission from the coal bed.

Presently, only the hard coal mines operated by the Jastrzębie Mining Company and the Kłodawa salt mine are exposed to the threat of gas and rock outbursts. Lately, that threat occurred in the Rudna copper ore mine, which is part of KGHM Polska Miedź SA. In September 2009, outburst of gases and rock was recorded in the Rudna copper ore mine while drilling a preparatory drift by means of explosives at a depth of approximately 1,200 m. As a result of the outburst, approximately 1,200 m^3 of crushed rock material filled the excavation for a length of approximately 70 m. At that time, no tremor was recorded by the seismic mine station. The mixture of gases released contained more than 80% nitrogen and only a few percent CH_4. Further geochemical examination of gases from that area showed that the released gases were genetically most similar to natural gas exploited in the deposits located in the reservoir rocks of basal limestone and Rotliegendes on the Polish Lowland, at a distance of 30–40 km from the mining zone of the Rudna mine (Dec, Pietsch, & Marzec, 2011). The aforementioned gas originated from an organic substance present in Carboniferous and partially in Devonian formations.

The main reason behind the gas and rock outburst in the Rudna mine was the disturbance of the balance between gas pressure and rock mass strength, the so-called gaso-geodynamic balance. We assumed that the balance was disturbed in the layer of highly porous and fissured dolomite filled with gases under high pressure because of drilling a preparatory drift by blasting. It must be stressed that until then, no similar phenomenon had been observed in the copper ore mines of KGHM Polska Miedź SA.

For the purposes of general recognition of the structure of the Zechstein formation and the top layers of the Rotliegend formations and the presence of anomalous zones at the Zechstein / Rotliegend border, a 3D seismic survey was performed on the terrain surface (Dec et al., 2011). This seismic survey did not provide sufficient information about the fracture system of the rock layers in the immediate vicinity of the excavations. Therefore, further investigation was conducted through mining excavations (Pilecki et al., 2013). Diverse research efforts were undertaken, which included geological, geophysical and geomechanical investigations, to assess the threat of an outburst in the parts of the deposit that were planned to be mined. Seismic tomography confirmed the information about the location of anomalous zones of elastic properties in the dolomite layer as indicated by the geological survey of the Rudna mine. The selected seismic anomalies have been verified through drilling. In some of boreholes, flows of gas and water of varying intensity were registered (Pilecki et al., 2013). To detail data on the location of weak zones, borehole ground penetrating radar (BGPR) was also carried out. BGPR studies have identified weak zones in the rock mass within a radius of approximately ten meters around the control borehole.

In light of the newly identified outburst threat, it was necessary to modify the technology of drilling preparatory and operating workings. In this paper, we describe the methodology used for detecting zones of gaso-geodynamic threat. This methodology is illustrated by example of the studies conducted in the area of XXVIII/1 field in the Rudna copper mine at a depth of approximately 1,200 m. We present experiences in identification of gaso-geodynamic zones in the rock mass with the help of geophysical methods. It was assumed that the fundamental and possibly useful information would be produced in the assessment of the alterations of elastic properties of a rock mass, obtained by means of seismic tomography.

Two series of seismic tomography were made in June 2013 and October 2014. The results were used to design and drill control boreholes and further to drill preparatory and operational workings in the XXVIII/1 field in the Rudna mine.

This study also presents a wider description of geological conditions, with particular focus on the circumstances behind the origin of gas traps in the rock mass of the Rudna mine. The summary highlights the most significant findings and advantages and limitations of the applied geophysical methods when used for the identification of gaso-geodynamic zones.

GEOLOGICAL CONDITIONS ASSOCIATED WITH THE OCCURRENCE OF GAS TRAPS

The copper ore deposit in the Rudna mine belongs to the category of bedded deposits (Pilecki et al., 2013). It occurs in three lithological types: in the sandstone of the Weissliegendes formation (top part of the Rotliegend formation), cupriferous clay slates and dolomites of basal limestone (top part of the Zechstein dolomite series). The lithological series is marked by non-uniform lithological structure and diverse mineralization; hence, the level of drilling exploratory excavations varies. Changes in the lithological structure of rock types are clearly related to the varying morphology of the topmost sandstone (Błaszczyk, 1981).

Over the entire area of the copper ore deposit, morphological elevations and depressions occur on the topmost limestone. Those forms are the relics of sand dunes, which originated in desert conditions during the period of the Rotliegend. The marine transgression in the Zechstein period brought about the flattening of the dune elevations and the resedimentation of sand in the lowering of the interdune areas (Kaczmarek, 2006). To date, in the mining zone of the Rudna mine, five elevations have been detected. They are 15–35 m high forms, with a maximum length of 25 km and width of approximately 1 km; the distance between the subsequent axes of elevations ranges from 1.5–3.5 km. The northeast slopes of the elevations are very steep, and the southwest slopes are weakly inclined, largely due to the inclination of the main tectonic unit of the Fore-Sudetian Monocline. The morphology of the upper part of carbonic formations and slates is not strictly bound up with the structure of the elevations. The older formations of the Zechstein dolomite series exhibit lesser tectonic discontinuity than the incumbent formations of that series. The Zechstein dolomite series was subject to tension and compression during the period of the formation of folded structures, overthrusts and faults (Jerzykiewicz, Kijewski, Mroczkowski, & Teisseyre, 1976). The fissures thus created were filled with material transported in water.

Within the Zechstein dolomite series, in the areas of fissuring isolated from above by the layers of anhydrite, gas traps are observed. Both fissures, as well as the porous spaces of folded structures, may serve as gas collectors. During the formation of the Fore-Sudetian Monocline, it is likely the gases flowed in from the Carboniferous period strata; afterwards, they accumulated in the layers of the dolomite series (Błaszczyk, 1981). The characteristics of the gas traps indicate that they are most likely to occur in the slopes of morphological elevations in the zones of tensile stress. They may also be present in the weakly inclined top portion of the Zechstein formations, if in the dolomite series, some larger zones of fissures and faults were formed in the area of a drag fold structure or an overthrust (Błaszczyk, 1981). The phenomena observed in the fissuring zone often include water outflow and sometimes fine crystalline white salt. It was assumed that the gas traps exist along the direction of major tectonic structures (Dec et al., 2011).

METHODOLOGY FOR DETECTING ZONES OF GASO-GEODYNAMIC THREAT

The methodology for detecting zones of gaso-geodynamic threat consists of three stages:

A – Performing seismic tomography in a part of the rock mass presumed to be threatened by outbursts of gases and rocks identified on the basis of geological investigation.

In the first stage A, a part of the deposit presumed to be threatened by outbursts of gases and rocks was identified on the basis of geological investigation. The gaso-geodynamic threat in the Rudna mine is related to the accumulation of gases, presumably in the porous and fissured rock mass of the dolomite series, in the sloped areas of the morphological elevations of the Weissliegendes limestone. Bearing that in mind, the aim of geophysical and geological investigation is to identify the weak zones associated with the fracturing of the dolomite layer in the area of morphological zones, in the immediate roof of the excavations.

The seismic measurements are taken from drifts, due to the way in which the deposit had been cut. At least 96 geophones of 100 Hz natural frequency are used in the measurements. They are installed at every 5. m in the sidewall of the drift in its roof part. The waves are generated by setting off a small explosive of 150–300 g placed in short 2 m boreholes located approximately every 15 m.

The basic results are presented in the form of the map of velocity field changes of the P-wave and the S-wave. On the basis of the P-wave and the S-wave values, other seismic parameters are calculated: the ratio of the P-wave velocity to the S-wave velocity (Vp/Vs), the dynamic Poisson's ratio and the dynamic Young's modulus of elasticity. In the maps, special attention is devoted to seismic anomalies that can correlate with zones of gaso-geodynamic threat in the rock mass.

B – Drilling control boreholes in the zone of seismic anomalies.

In the second stage, the results of the seismic tomography are used to design and drill a control borehole in the zone of the seismic anomalies, for the purpose of a more detailed investigation of the structure and properties of the rock mass. The control borehole is approximately 0.075 m in diameter and is inclined in the direction of the dolomite layer in the excavation roof. The borehole is drilled up to the bottom part of the anhydrite layer located above the dolomite layer. In the process of drilling, special attention is devoted to recording the outflow of gases and water, as well as to the fracturing of the rocks and other effects related with drilling. On the basis of the core, the lithological description is verified, and the *rock quality designation* (RQD) index (Deere, 1964) is calculated.

C – Performing BGPR measurements to obtain more detailed geological information.

In the third stage, the BGPR was used to obtain detailed geological information. In the mine conditions under consideration, a BGPR with antennas at a frequency of 100 MHz made it possible to collect information about the rock mass around the borehole at a distance of approximately 12 m. The specific BGPR measurement methodology applied is presented in the work of Łątka, Czarny, Krawiec, Kudyk, & Pilecki, (2010). As a result, the final radargram is obtained. The radargram was correlated with the lithology, changes in the RQD index and the water and gases present that were gathered during drilling.

AN EXAMPLE OF RESULTS IN THE XXVIII/1 FIELD IN THE RUDNA COPPER ORE MINE

In the cross-section of the excavations where the geophysical tests were performed, sandstone was present. Above, a clay slate of a minor thickness (amounting to several dozen centimeters) existed, which locally disappeared, and a layer of dolomite with the thickness up to approximately 12 m was observed (Figure 1). It must be stressed that the dolomite layer possessed the most favorable seismic properties in comparison with the adjacent rock types: sandstone, clay slate and anhydrite (Figure 1).

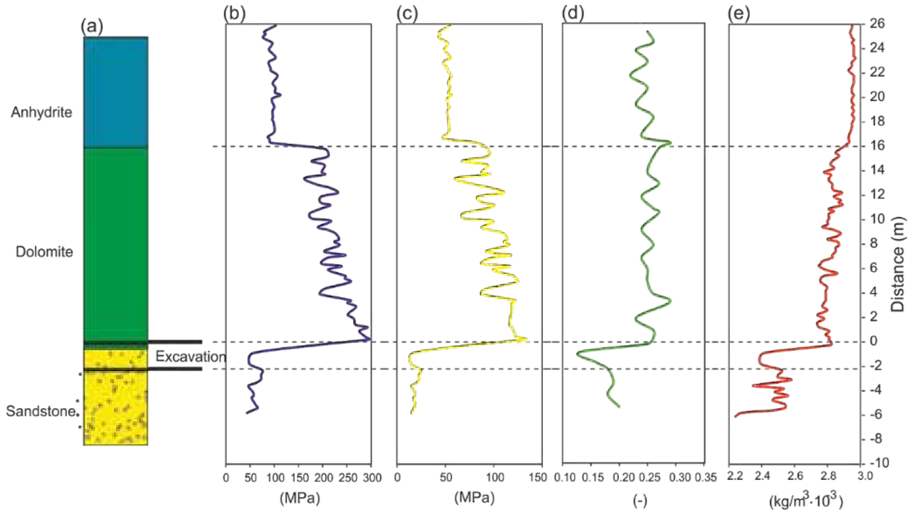

Figure 1 – a) Geological profile from the area under investigation; b) uniaxial compressive strength; c) Young's modulus; d) Poisson ratio and e) the rock's volumetric density on the basis of KGHM Cuprum data

Seismic tomography in the XXVIII / 1 field of the Rudna mine was made in two series of measurement in June 2013 and in October 2014.

To improve the identification of waves in the rock mass, numerical modeling was conducted using a program developed by Bohlen (2002). We analyzed the propagation of waves in a simplified three-layer rock mass consisting of sandstone, dolomite and anhydrite. For the assumed physical and mechanical parameters characteristic for the study area, wave field images were calculated (Figure 2). In the case of the Y component characterizing the development of a P-wave, one can assume that it can be associated with the direct wave propagating through the layer of dolomite. The S-wave (component X) develops later relative to the P wave, but, similarly, the first picks of this wave are related to the wave propagated in dolomite. In practice, due to the distance of several hundred meters, the identification of different types of waves is less complicated. Therefore, the identification of the waves in the dolomite layer presents no considerable difficulties.

In one survey, approximately several dozen hundreds of seismic rays were obtained (Figure 3). On the basis of the first survey, the poorly constrained zones are located mostly in the edges of the area under investigation, in the neighborhood of sources and receivers. The smallest number of rays that covered the mesh of the grid (25 × 25 m) was 12 for the P-wave and 17 for the S-wave. A simultaneous iterative reconstruction technique algorithm was applied in the calculations, with curvilinear reconstruction of the seismic rays (Gilbert, 1972). The maximum mismatch between calculated and measured times for the resulting model for a single seismic ray equaled 2.08 ms for the P-wave and 2.24 ms for the S-wave; therefore, the errors in wave velocity calculations are approximately 110 m/s and 50 m/s for the P-wave and S-wave, respectively.

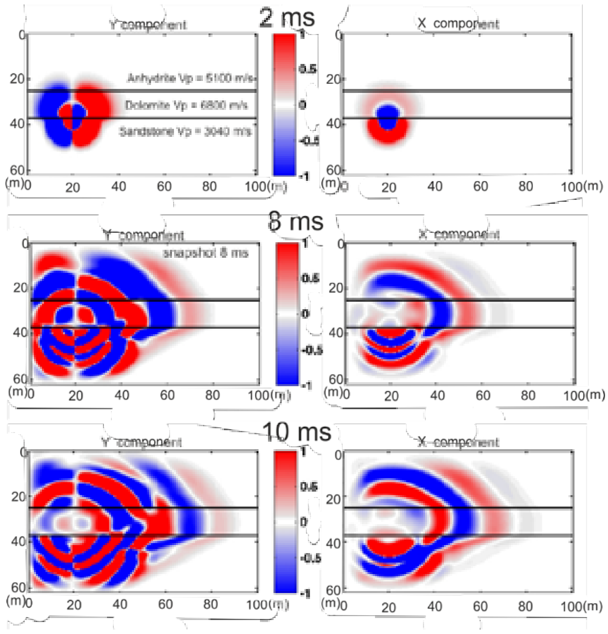

Figure 2 – The development of the wave field numerically modeled in a three-layer rock mass of anhydrite - dolomite - sandstone after 2, 8 and 10 ms. Images depict the component Y (left side) and the component X (right side) (Pilecki, Laskowski, Porębski, Pilecka, Jach, Czarny, …Harba, 2015)

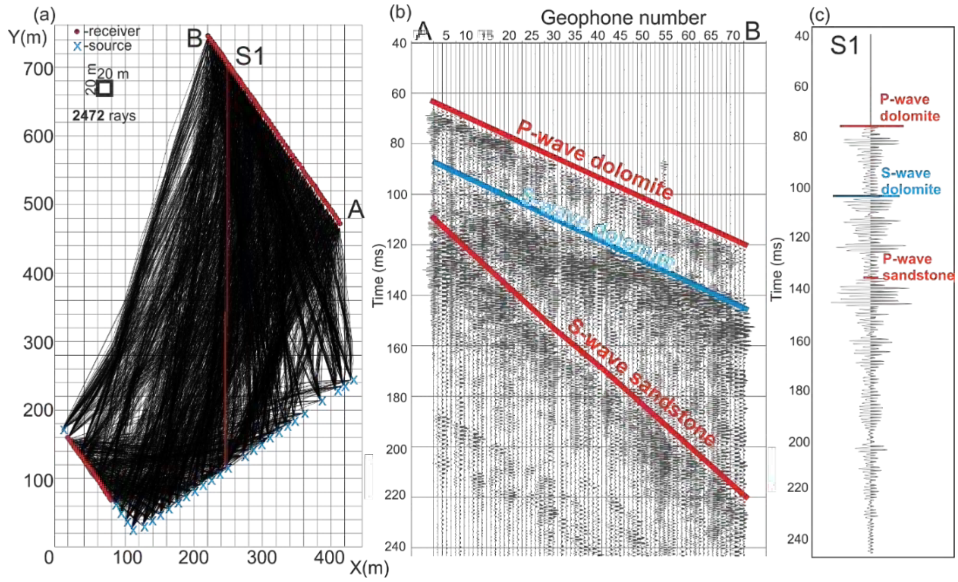

Figure 3 – a) Map of P-wave rays path in local coordinates from the area under investigation; b) an example of seismic recording in section A-B; c) an example of a trace in point S1

Comparison of the results of both measurements series (Figures 4 - 6) clearly indicates that contour lines of seismic parameter values cannot be connected because of the different influence of the secondary stress field associated with the progress of the exploitation. In the front zone of the longwall, one can observe clear anomalies associated with the additional load from the roof layers bended above the goaf. In particular, the impact is visible in the north-east corner of the study area due to the greater intensity of exploitation. Identified positive anomalies, with higher values of velocity, are affected by the stiffness increase of the layer of dolomite. It can be explained by the tightening of existing pores and cracks. In the developing failure conditions, such an anomaly can be negative due to the creation of additional fractures. The identification of the location of seismic anomalies made it possible to design control boreholes.

On the map of the P-wave velocity field (Figure 4), a negative anomaly no. 1 is indicated for the first measurement. In the control borehole drilled in this area, the presence of gases was revealed, but the outflow was low. The core obtained from the control borehole showed that the zone of intense fracturing had been properly identified.

In the case of series 2, for the anomaly no. 1 velocity values slightly deviate from the values for the undisturbed field and are below the level of measurement error. One of two possible control boreholes C1 and C2 were proposed to drill into the anomaly area. In the C2 borehole, there was no presence of gas, but only the small outflow of water at 3 l/min in the top part of the layer of dolomite. In both series, anomaly no. 2 is associated with an increase in seismic velocity under the influence of the exploitation because of the additional load from the bended roof layers.

On the maps of other parameters, similar seismic anomalies were interpreted. The map of the velocity field of an S-wave (Figure 5) indicated two anomalies: designated as no. 1 - low velocity and no. 2 - the higher velocity. Both anomalies are similarly situated as anomalies on the map of the P- wave velocity field. On the right side of the longwall, a small velocity increase was marked, comparable to a measurement error. The borehole H-35 in the area of this anomaly showed no evident gaso-geodynamic threat. The genesis of anomalies on this map is the same as for the P-wave anomalies.

On the map of the dynamic Young's modulus of elasticity changes, marked anomaly no. 1, which represents the zone of lesser elasticity in the rock mass, and no. 2, the anomaly of greater elasticity. The image of dynamic E modulus changes is the most similar to the image of the S-wave velocity. A similar system of seismic anomalies was obtained for the other seismic parameters: dynamic Poisson's ratio and the ratio of P-wave velocity to S-wave velocity (Vp/Vs).

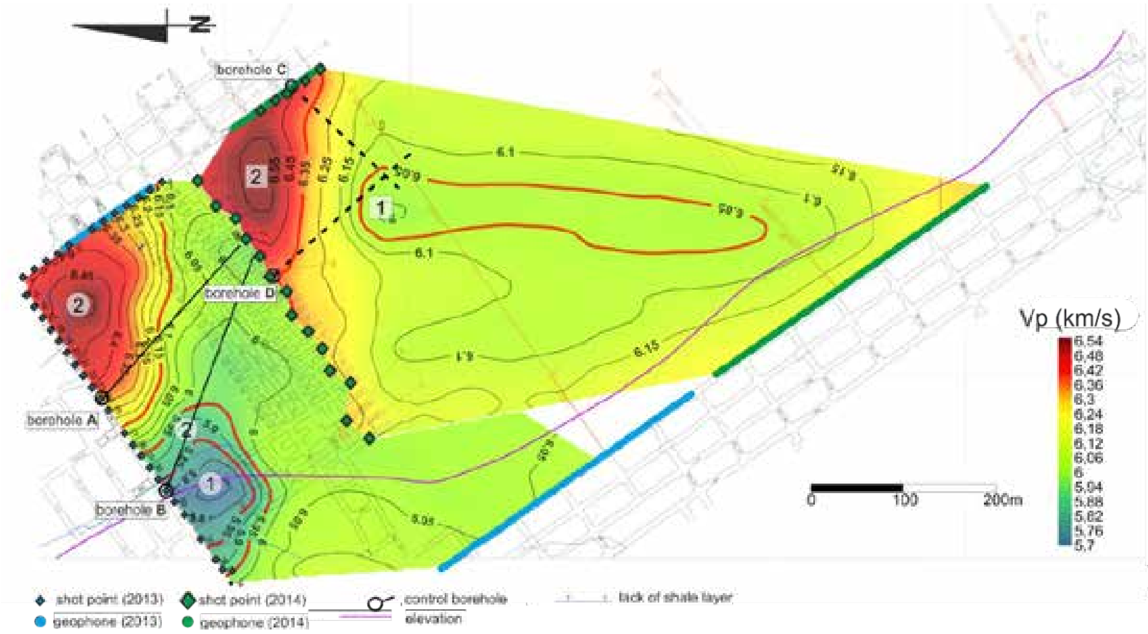

Figure 4 – Map of P-wave velocity changes for two measurement series in June 2013 and October 2014 in the XXVIII/1 Rudna mine (on the basis Pilecki et al., 2015)

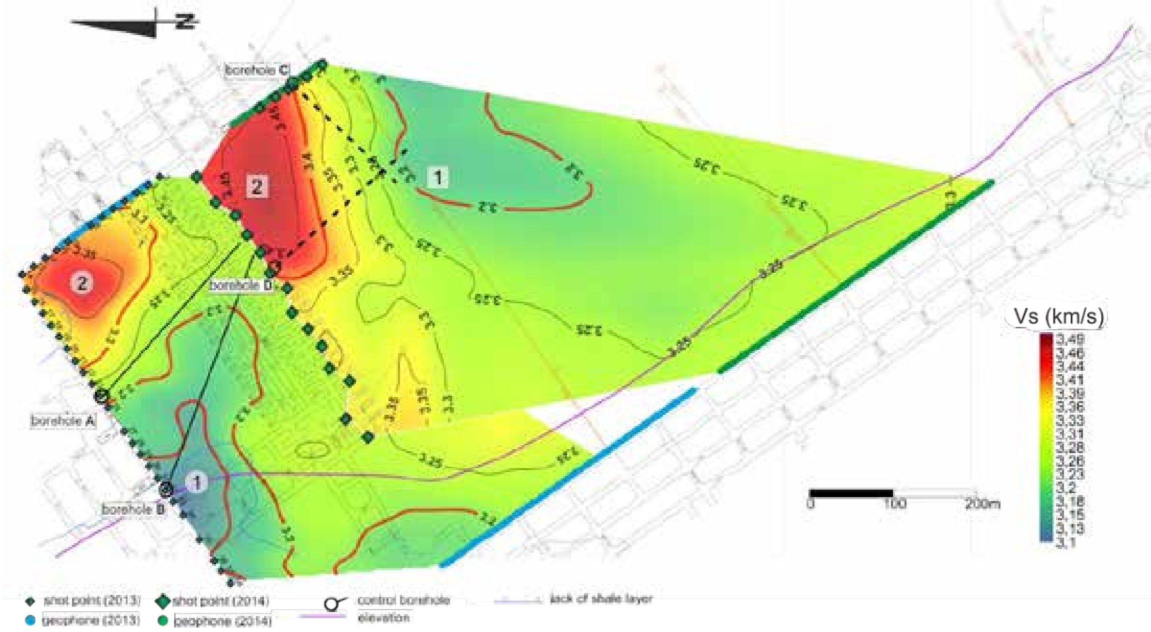

Figure 5 – Map of S-wave velocity changes for two measurement series in June 2013 and October 2014 in the XXVIII/1 Rudna mine (on the basis Pilecki et al., 2015)

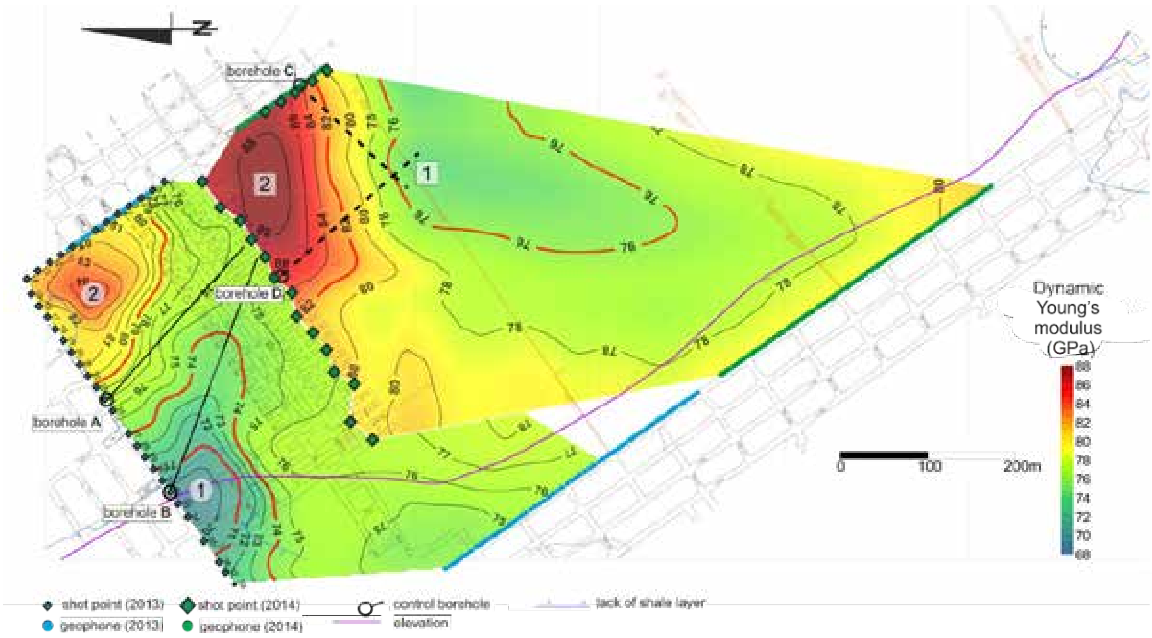

Figure 6 – Map of dynamic Young's modulus of elasticity changes for two measurement series in June 2013 and October 2014 in the XXVIII/1 Rudna mine (on the basis Pilecki et al., 2015)

Recognized anomalous seismic parameter values indicate small changes in the stiffness of the layer of dolomite undisturbed by exploitation of the rock mass, with the exception of anomalies no. 1 of first measurement. The control borehole drilled in the area of this anomaly showed the greatest presence of gases, but does not pose much of a gaso-geodynamic threat. Other anomalies of natural origin were small, comparable to the measurement error.

The obtained results of the drilling control boreholes prove the results of the seismic surveys. Due to the small level of gaso-geodynamic threat there was no need to perform BGPR surveys.

CONCLUSIONS

This work presents experiences in identification of gaso-geodynamic zones in the rock mass of the Rudna copper ore mine. The methodology used for detecting zones of gaso-geodynamic threat is a result of several years of geological, geomechanical and geophysical research. This methodology is illustrated by the studies conducted in the area of the XXVIII/1 field in the Rudna mine at a depth of approximately 1200 m. We presented experiences in identification of gaso-geodynamic zones in the rock mass with the help of geophysical methods. In the proposed methodology, it was assumed that the fundamental and possibly useful information would be produced in the assessment of the changes of elastic properties of a rock mass, obtained by means of seismic tomography. In the Rudna mine conditions, seismic tests concentrated on the recognition of the structure and properties of the dolomite layer located in the immediate roof of the excavations. Seismic tomography results served as the data necessary to design a control borehole to verify the seismic information. In earlier cases (Pilecki et al., 2013), not presented in this text, BGPR with 100 MHz antennas was used. The BGPR tests made it possible to identify more precisely the properties of rock mass around the control borehole. The radar data were used in correlation with information on lithology and the degree of fracturing of the rock mass, including the RQD index, as well as additional information on the presence of water and gases, mechanic symptoms and other phenomena observed in the process of drilling.

The present experiences in the use of the proposed methodology made it possible to formulate the following conclusions:

- As a result of the verification of the seismic anomalies by means of the borehole tests, the strongest relationship existed for the S-wave velocity and the dynamic Young's modulus changes. It should be assumed that the other negative and positive anomalies, which were not controlled in the borehole tests, have analogical reasons as the anomalies that were verified.
    - Small changes of seismic parameters indicate the relatively favorable level of gaso-geodynamic threat in rock mass, characterized by a weak variation of fracture intensity in the dolomite layer.
    - In the vicinity of the front line of the longwall, observed seismic anomalies are related to the influence of mining operations. The nature of the anomalies indicates that they occur due to tightening of existing pores and cracks. In consequence, the increase of the stiffness of the dolomite layer is observed. The main reason is the additional load from bending of the roof strata.

The test results complement the existing knowledge about the occurrence of gaso-geodynamic phenomena in terms of changes of geophysical parameters. Taking into consideration the advantages and limitations of the geophysical methods (seismic tomography and BGPR), it must be stressed that the results supplement the geological engineering investigation in an interesting and revealing way. An important limitation is the workload of seismic surveys in difficult conditions at the depth of approximately 1,200 m; thus, they must be carefully designed. It is also important to achieve a good technical condition of the control borehole for the BGPR surveys.

The presented methods do not solve the major problems related to the threat of an outburst of gases and rock, but they may be useful for monitoring the rock mass in the course of mining operations. However, the results presented in this study need further investigation as to their usefulness in varying conditions of an outburst threat.

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EXPERIMENTAL STUDY OF SUSTAINABLE EXTRACTION OF GEOTHERMAL HEAT FROM UNDERGROUND MINES IN CANADA

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Experimental study of sustainable extraction of geothermal heat from underground mines in Canada

ABSTRACT

Underground mines are usually perceived as mineral resources with short-to-medium life expectancy. Though the life cycle of a mine could change depending on the characteristics of its reserves and operations as well as market dynamics, decommissioning of the mine is an inevitable fact. This unsustainable ending casts socio-economic shadows on every community in which mining is the main business regardless of its size. However, every underground mining operation creates extensive galleries and excavations with access to deep ground temperatures which make them fantastic geothermal reservoirs. This paper studies the possibility of harvesting geothermal heat from underground mines in Canada with a focus on its economic and environmental benefits. It includes in-situ data gathered at more than twenty underground mines in Canada based on which geothermal capacity of each mine is assessed. During mineral extraction, underground mines are continuously dewatered to prevent mine flooding due to ground water infiltration. This study shows that these mine waters contain considerable geothermal heat energy which can be sustainably provided to the mining facilities as well as any neighboring community. Upon ore depletion, swamp pumps are removed allowing the flooding of the mine. However, the provision of geothermal energy will not cease with depletion of mineral reserves of a mine. In fact, this study shows that the accumulation of these underground waters creates a massive heat storage capacity and increases the geothermal capacity of the decommissioned mine. This clean, inexpensive and renewable source of geothermal heat helps mining industry save energy, cut operating costs, reduce its carbon footprint and eventually collect revenues by cashing the carbon credit to be regulated through international environment accords. Last but not the least, it will contribute to the sustainability of mining communities by attracting businesses such as greenhouse farms and warehouses.

KEYWORDS

Geothermal energy, mine water, sustainable energy, renewable energy

INTRODUCTION

Energy intensity imposes a real risk to future stability of mining operations. It is mainly constituted by two major factors. The first issue is the insatiable demand of mining industry for all forms of energy including electricity, heat or cooling [1]. In fact, mining is categorized amongst the most energy intensive industries along with chemical, petroleum and based metals [2]. This enormous energy need is accompanied by the continuously raising trend of fossil fuel prices in the long run (i.e. short term prices changes damp out). Therefore, energy costs are making up a bigger piece of the total operating costs of mining operations. This issue will be bolded even more by introduction of the pending carbon taxes by major industrial countries [3]. The second issue rises from the mere impossibility of predicting fossil fuel prices which is caused by numerous political and economic players involved. This unreliable nature of energy market will deem operating costs essentially unpredictable and therefore a fundamental risk source. These major issues have motivated mining industry to be on a nonstop search for new measures and technologies that can help them improve their energy efficiency in order to reduce their energy demand and carbon footprint [4]. Geothermal energy is a safe and reliable source of energy. It causes far less environmental impacts relative to conventional fossil fuels, and has the advantage of providing a continuous and uninterrupted supply of energy, unlike several other renewable sources such as wind, tidal and solar.

EMERG (Earth/Mine Energy Research Group), at the Department of Mining and Materials Engineering of McGill University, is proposing the idea of utilizing the mine water found in active mines, as a geothermal source to provide heat for onsite operations. A large proportion of the capital costs for geothermal energy are associated with drilling boreholes and pumping water to the surface. For an active mine with a fully functioning infrastructure, such costs are already covered by the operating expenses. Our design proposes the installation of heat pumps as a step in the dewatering network, before the water discharges to the surface collecting ponds. Moreover, the various design parameters required for the design of our system, such as mine water and ambient temperature values, relative humidity, information on local geology and dewatering activities, can be readily obtained for an active mine. All of this greatly facilitates the development of the proposed design, while also greatly reducing the associated capital costs and expenses. Overall, there are immense benefits to replacing electric or natural gas heating systems with geothermal heating system within an active mine.

RESULTS & DISCUSSIONS

The present study includes data from about 13 active Canadian mines that were visited and that includes mines operated by Agnico Eagle, Barrick, Glencore, KGHM, and Vale. From Agnico Eagle Goldex, Lapa and Laronde underground mining operations were visited. Moreover, Hemlo, Kidd Mine, and Levack underground mining operations from Barrick, Goldcorp, and KGHM (respectively) were visited. Furthermore, underground mines operated by Vale were visited and these include Birtchtree, Coleman, Copper Cliff North, Copper Cliff South, Creighton, Garson, Stobie, Thompson T1 and Thompson T3.

The study gives an estimate of the energy assessment with relation to a ranging water flow (GPM) and the varying temperature difference of the water due to the heat pump. As a result, the heat capacity of the potential geothermal system can be derived and is provided in Kilowatts (kW). After calculating the potential energy a heat pump unit was selected for each mining operation. The annual energy savings as well as the reduction in carbon dioxide emission were calculated. Since the type of energy used to heat the mine buildings and facilities is not currently known, two comparison calculation models were included. One is made for a scenario were the geothermal heating system replaces natural gas or propane heating system while the other comparison is made for a scenario were the geothermal heating system replaces electrical heating system.

Table 1 is a summary of the resulted calculations. It contains information about the pumped water, the possible heat gain, and the estimated annual savings and CO2 reduction. These calculations are based on utilizing FHP's WW420 heat pump(s). It should be noted that Canada has been moving to greener sources of energy in order to decrease carbon dioxide emissions. Therefore, the amount of energy generated from high carbon emitting sources such as coal and oil will shift toward less polluting energy sources such as hydroelectric and nuclear.

The findings of this study suggest that geothermal energy extraction via heat exchangers is a viable opportunity. The summary table provided illustrates the economic and environmental benefits from extracting geothermal energy. In addition, the environmental benefits through the reduction of carbon emissions can result in a high public relation value as well as savings from the pending carbon taxes. It is important to note that the conclusion in this preliminary study is based on conservative calculations so as not to be misinterpreted.

The next phase of the project requires the implementation of the geothermal heating system into an active mine site. A heat pump or a chiller can be placed in an active mine and be monitored for 9 months in order to assess the field data and compare it to the calculated. This would be the first time ever to have a geothermal heating system implemented in an active mine. The following two possible scenarios can be devised for implementation phase.

The first scenario includes the implementation of a large scale geothermal chiller at mine site. The chiller will be used to heat mine intake air. In this case, the geothermal energy of the mine will replace the natural gas or propane consumed for preheating mine intake air in the cold season. The chiller can also be used for mine cooling purposes in summer time. A schematic representation of a chiller application is presented in Figure 1.

Table 1. Summary of data gathered for assessing the geothermal potential of active mines

Mine #	Water Flow (GPM)	Water Temperature (^{0}C)	Heat Gain (kW)	Annual Savings ($/yr)		CO_2 Reduction (t/yr)	
				Natural Gas	Electricity	Natural Gas	Electricity
1	1000	16.5	1,160	-	305,360	1427	252
2	375	16.7	430	-	115,023	537	95
3	320	22	430	2,805	113,772	513	91
4	398	16.7	465	125,930	188,092	554	305
5	950	20.9	1,578	167,173	415,116	1,897	2,692
6	290	18.9	360	21,000	205,000	437	620
7	133	16.6	160	8,000	88,000	190	270
8	515	16.7	600	31,000	342,000	738	1,048
9	675	14	730	30,000	414,000	913	1,296
10	560	15.8	640	30,500	362,000	787	1,116
11	800	11.5	800	24,500	455,540	1,025	1,455
12	719	13.8	760	288,000	136,000	1,102	209
13	116	13.6	125	46,220	21,800	177	34

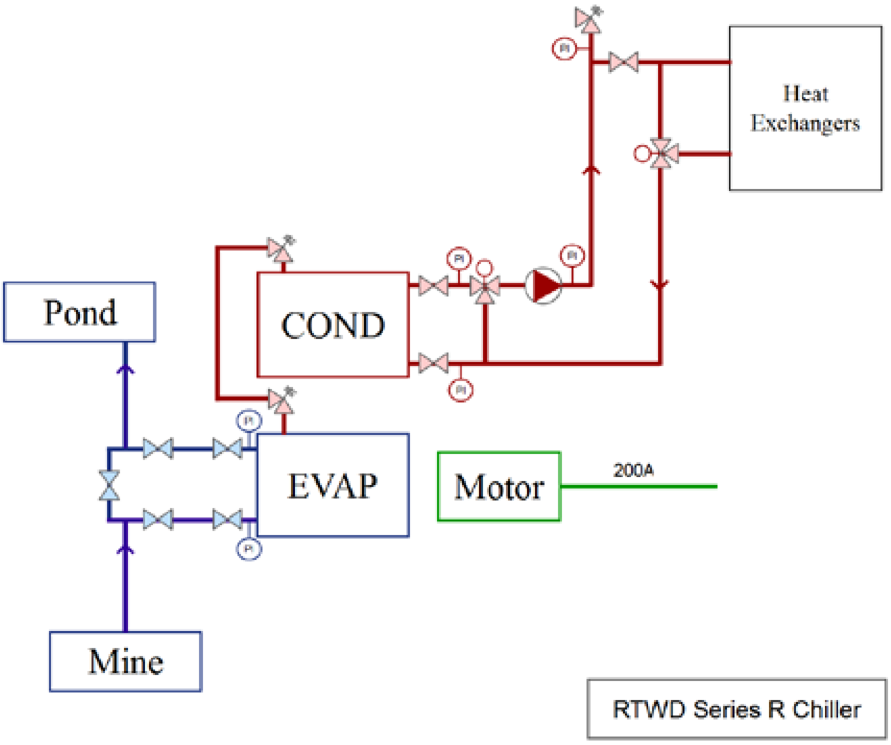

Figure 1: Schematics of geothermal chiller application

The capital cost for a chiller from Trane is $125,000 per unit. Installation costs will depend on the specific electrical and piping works required at each mine site. The chiller can harvest considerable amount of heat energy and so that is reflected on the payback period. Since a chiller would have a relatively high heating capacity, it is suggested to be applied into the mine ventilation system. Therefore, it can augment the natural gas heating system that is installed in a mine in order to save the amount of natural gas heating used.

Another scenario is to install heat pump(s) at mine site to provide mine surface facilities with geothermal energy. In the case of heat pumps (Figure 2), it is possible to utilize various number of units depending on the needed heating requirement. The benefit of a heat pump system is that the price per unit is lower when compared to a chiller. In addition, the size of the unit is smaller than the chiller. On the other hand, a chiller can handle more water flow per unit and so one unit can be suffice to handle all the water flow produced by the mine. In addition, due to the larger size, it can produce more heat per unit.

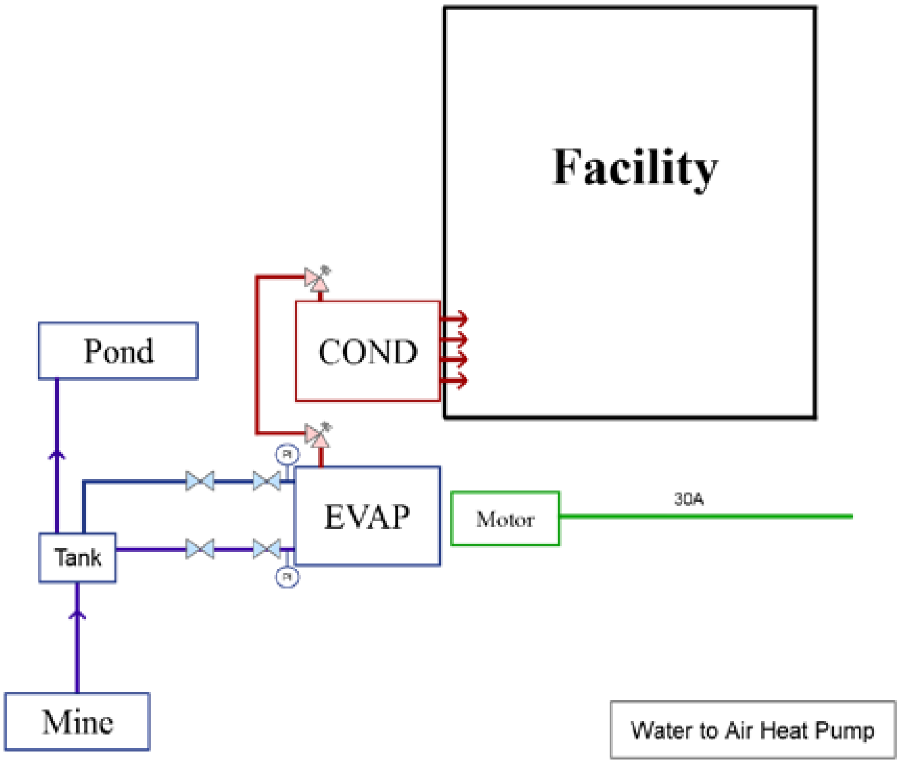

Figure 2: Schematics of geothermal heat pump application

When it comes to the economics of each scenario it can be observed that there is a significant difference. As for the case of the utilizing heat pumps, the cost of the unit from Trane is lower when compared to a chiller ($20,000 per unit). Each heat pump unit is capable of extracting heat from 51 GPM of geothermal water. For example, if a mine provides 560 GPM of water, 11 geothermal heat pumps can be used. Thus the total capital costs for heat pump application will be higher than Chiller application. Installation costs will depend on the specific electrical and piping works required at each mine site. The heat pump can be placed at any facility within the mine site and it can replace either a natural gas heating system or an electricity heating system. Since natural gas prices are lower compared to electricity prices it is expected that the annual savings are more significant when replacing an electrical heating system with a geothermal system.

CONCLUSIONS

From this study it can be concluded that geothermal energy extraction via heat exchangers is a viable opportunity. The summary table provided illustrates the economic and environmental benefits from extracting geothermal energy. In addition, the environmental benefits through the reduction of carbon emissions can result in a high public relation value as well as savings from the pending carbon taxes. It is important to note that the conclusion in this preliminary study is based on conservative calculations so as not to be misinterpreted. Geothermal heat capacity of an active mine helps with saving energy, cutting operating costs, reducing carbon footprint and eventually collecting revenues by cashing the carbon credits. Eventually, it will contribute to the sustainability of mining communities by attracting businesses such as greenhouse farms and warehouses.

ACKNOWLEDGEMENT

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FEATURING THE NANOBUBBLES SCENARIO IN MINING AND MINERAL FLOTATION

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FEATURING THE NANOBUBBLES SCENARIO IN MINING AND MINERAL FLOTATION

ABSTRACT

Nanobubbles (NBs, with mean diameter of 150 – 200 nm) have special properties which may be employed as ancillary tool to enhance the flotation of minerals, pollutants removal and water reuse. Microbubbles (MBs: 30-100 µm diameter) and NBs are generated simultaneously after depressurization of a flux of water containing dissolved air and forced through flow constrictors, such as a venturi tube or a needle valve. NBs rapidly attach to hydrophobic particles, colloids, mineral or residual ultrafines, precipitates and oil droplets; decreasing the bubble-particle density and increasing the flotation efficiency. After coating the particles units, the NBs serve as nuclei for the MBs and/or MaBs (macrobubbles: 700 µm-2 mm diameter) to adhere, assisting the flotation. Based on the discovery of these phenomena, we studied forms of generation, at high concentrations, and explored several features and applications of these NBs which we summarize in this contribution. In the mineral separation, the main advantages are the formation of particles aggregates by the NBs and the rapid flotation by the consortium NBs (billions of them!) and the coarse bubbles, reduced amount of reagents needed and lower energy consumption. Herein, we studied the flotation of low feed grade gold and copper-gold ores, at laboratory scale, as case-studies and the removal of pollutants removal. We obtained higher flotation recoveries of gold and copper sulfides after injection of NBs and complete removal of xanthate (feed = 60 mg.L^{-1}) by precipitation of the xanthate with copper ions (copper sulfate) at pH 9, followed by flotation with bubbles generated by depressurization of saturated air-water (2.5-5 bars). The removal was very rapid with flotation constants rates of 8.5-10 min^{-1}. Equally, we readily removed mineral particles (sand, silicates, and quartz) coated with ether amines, from process water (thickeners overflows of iron flotation tailings), reducing the turbidity of this wastewater from 400 NTU to 21 NTU. We were able to carry out this research only with modern techniques and special equipment to fully characterize the NBs (concentration; interfacial properties; aggregation power). Further, we invented a new route to generate the NBs, at high concentration (> 10^{9} NBs.mL^{-1}). We concluded that the use of tiny bubbles - NBs and MBs, will lead to cleaner and more efficient mineral flotation, also improving process water quality, enhancing the water reuse in dams and in thickener polluted overflows.

KEYWORDS

Nanobubbles generation and applications, mineral flotation, pollutants removal

INTRODUCTION

Froth flotation is the most efficient and cost-effective fine particle unit operation in mineral processing (Fuerstenau, M.C. et al., 2007). In modern mineral sustainable processing, flotation is employed not only to recover difficult-to-treat particles, but also on water reuse and wastewater treatment, nowadays the major issues to solve because they constitute a significant environmental concern and a technical challenge (Azevedo et al., 2016; Calgaroto et al., 2014; Rubio et al., 2002).

In mineral flotation, the separation efficiency decreases sharply with ultrafine particles. Studies have shown that this drawback is mainly due to the low probability of bubble-particle collision and attachment. Thus, improvements are needed to optimize particles-bubbles attachment, reducing the detachment probability (Fan et al., 2010; Sobhy and Tao, 2013).

Recent articles have indicated that the process efficiency can be significantly enhanced with microbubbles (MBs) and/or nanobubbles (NBs) by increasing coarse bubbles attachment onto the adhered NBs and by aggregating the problematic fine particles (Azevedo et al., 2016; Calgaroto et al., 2015). More, fine bubbles have great impacts on gas holdup, essential in the froth flotation of mineral based process industries (Gontijo et al., 2008; Jameson et al., 2007).

Furthermore, mineral processing plants are known to have serious problems with wastewater and thus there is a need for a sustainable emission of the process water to the environment or reusing the water back to the process. For example, thickener overflows bearing residual reagents, colloidal or fine solids have to be treated should water has to be reused. Yet, flotation feeds desliming using hydrocyclones is a common practice when slimes affect mineral values collection and/or the froth.

Usually, this overflow is disposed in dams and when the suspensions are diluted (watery), stabilization problems may occur. Again, a solid/liquid operation is necessary and the main technical dilemma is which process would be a better option. The fineness of the solids poses problems if filtration is to be used due to the blockage of the porous filter materials by the particles.

The flotation with MBs and NBs generated by depressurization of dissolved air in water, so-called dissolved air flotation (DAF), appears to be a promising option, in both before-mentioned cases. Unfortunately, only few examples of these applications are available in the mining area (Calgaroto et al., 2016; Rubio, et al, 2002). Flotation with MBs has a high potential for removing colloids, precipitates, flocs, oily drops, fines and ultrafines solids (Edzwald, 2010; Oliveira and Rubio, 2009; Rodrigues and Rubio, 2007; Rubio et al., 2002).

The generation of MBs and NBs in hydrodynamic cavitation tubes and by depressurization of air-saturated water has been reported, with many improvements in the flotation processes (Calgaroto et al., 2016, 2015; Fan et al., 2010; Sobhy and Tao, 2013). The main properties of NBs are their high stability, longevity, rapid attachment to hydrophobic surfaces serving as nuclei for coarser bubbles adhesion. Thus, these particularities broaden the potential applications of NBs in modern mining processes.

In the flotation of mineral particles, a number of recent studies have shown advantages in mineral flotation in the presence of NBs (Ahmadi et al., 2014; Calgaroto et al., 2015; Fan and Tao, 2008; Fan et al., 2012, 2010; Sobhy and Tao, 2013). The main claims are that the NBs increase the contact angles and, subsequently, enhance the probability of flotation (coal, phosphates), mainly the bubbles-particle attachment and stability. Yet, the enhancement of particle flotation rate is another benefit (Fan et al., 2013).

In the environmental area, technologies involving NBs have been applied in the removal of amine collector from wastewaters (Calgaroto et al., 2016), bioremediation of groundwater, degradation of surfactants and industrial wastewater treatment (Agarwal et al., 2011; Li et al., 2014; Tsai et al., 2007).

Flotation collectors and frothers are organic compounds widely used in the mining industry, such as in the copper, iron and potassium mining. High levels of organic collectors (xanthates and amines) can be harmful and cannot be directly disposed into aquatic bodies (Newsome et al., 1991). Their toxicity, degradation and byproducts formed are issues which are not completely understood. Usually, the organic collectors are either solubilized in water or adsorbed onto particle surfaces, which sometimes are removed from the system as a mineralized froth and dumped in tailings dams. Millions of tons of these harmful derivatives per year are used in flotation processes, volumes that will continue to increase in the next few years, due to the more complex (disseminated-locked particles) and low feed grade of values in future ores (Araujo et al., 2010).

Herein, the innovation is to employ NBs together (or not) with the MBs in different systems to investigate their effects on the flotation of mineral particles and/or process water treatment. It is believed that further research involving NBs is important to understand the fundamentals and broaden their technological applications. There is a demand for advances in terms of sustainable (workable) generation, mineral particles aggregation, hydrophobizing power and flotation (minerals fines and wastewater treatment and reuse). This study is a continuation of a series of articles regarding NBs and focuses on their generation as highly loaded aqueous dispersions and their role in mineral and effluent processing. Flotation of particles and residual mining reagents were summarized, and their potential in sustainable mining is envisaged.

EXPERIMENTAL

Materials

NBs generation

We used deionized (DI) water (product of a reverse-osmosis, ion-exchange resins and activated carbon) with a conductivity of 3 $\mu S.cm^{-1}$, a surface tension of 72.5 $mN.m^{-1}$ and pH 5.5 to produce NBs aqueous dispersions. NaOH and HCl solutions from Vetec® (Rio de Janeiro - RJ, Brazil) were used for medium pH adjustments. Pine oil (terpene alcohol - CH3–C6H9–(OH)–C3H5 -, supplied by Química Maragno®; Turvo – SC, Brazil) was used to reduce the water surface tension during NBs generation.

Minerals

We used metal sulfide ores (Cu, Au), from different Brazilian deposits, which were crushed, grounded (P80: 74 µm), homogenized and quartered, for chemical analysis and flotation studies. Sample 1 (Au associated with Cu sulfide ore) had a content of 0.3 $g.t^{-1}$ Au and 0.6 % Cu and Sample 2 (free Au and associated with sulfide ore) had a content of 0.8 $g.t^{-1}$ in feed. Galena (PbS) and quartz (SiO_2) lumps from South Brazil were used to investigate mineral-bubbles interactions, in the presence and absence of flotation collectors (isopropyl xanthate, supplied by Flomin Co, and decil-ether- amine, from Clariant, respectively).

Flotation reagents

For Sample 1 (Cu-Au sulfide), we used the collector PAX (Potassium amyl xanthate); the promoter Aero-MX 7020 (Aliphatic Alcohol Mixture); and a frother mixture (Flotanol D25-polypropylene glycol and polypropylene glycol methyl ether + Flomin F650-Glycol Ether). For Sample 2 (Au sulfide), we used Na_2S, as activator; collector PAX; promoter AP3473 (sodium diisobutyl dithiophosphate) and another frother mixture (MIBC-methyl isobutyl carbinol + Flomin F650).

Aggregation studies by NBs

We used spherical silica particles (d = 1 µm, AngströmSphere®) as a model of hydrophilic particulate system, and the hydrofobizing agent decil-ether-amine (Flotigam® EDA 3B, from Clariant).

Residual solids removal by flotation with MBs and NBs (DAF-dissolved air flotation)

We used process water from the overflows of a tailings thickener of an iron ore concentrate. Ferric chloride ($FeCl_3$) was used for coagulation followed by injection of bubbles generated after water depressurization at 4 bar.

Residual soluble reagents

For the preparation of xanthate ions solutions, example of residual ore collector, we utilized amyl xanthate (from Flomin@) and precipitate the anions with copper sulfate ($CuSO_4$, from Reagen@). For the preparation of iron ions solutions (Fe^{+3}), as example of residual ions, we used anhydrous ferric chloride. A sodium hydroxide (NaOH, from Vetec@) solution was utilized for pH adjustment.

Methods

NBs aqueous dispersion generation

Bubbles were formed by depressurization of DI water, previously saturated with air at various saturation pressures (P_{sat}) during 30 min in a steel saturation vessel, equipped with an internal container made of glass (40 cm high, 10 cm in diameter, and 0.5 cm thick; 0.7 L effective capacity). The depressurization-cavitation stage was performed at a 2-mm internal diameter needle valve (Globo 012-Santi®, São Paulo – SP, Brazil) and MBs and NBs were generated in a glass column, as described by Calgaroto et al. (2014).

Then, we separated the MBs from the NBs exploring the fact that the MBs rise (during 5 min) and collapse in the liquid surface, abandoning the glass column (Azevedo et al., 2016; Calgaroto et al., 2015). This procedure takes advantages of the high stability of NBs in aqueous systems and their lack of buoyancy.

Size and concentration of NBs

We measured the concentration and size of NBs in quadruplicates in a Nanoparticle Tracking Analysis (NTA) instrument (NanoSight LM10 & NTA 2.0 Analytical Software, Malvern Instrument Ltd, Salisbury, UK) at room temperature, after using different P_{sat} values (2.5 - 6 bar) for generating bubbles at various surface tension values (pine oil solutions of 10; 20; 50; 70 and 100 mg.L^{-1}).The NBs aqueous dispersion samples (approximately 0.1 mL) were injected into the sample chamber with sterile syringes (5 mL of capacity) until the liquid reached the tip of the nozzle. All samples were measured without dissolution, and each video clip of the particles was captured during 10 seconds with a manual shutter and gain adjustments. The NTA analytical software was used for capture and calculations. The obtained values of the mean size, size distribution and number of NBs are the arithmetically calculated values for a triplicate experiment.

<u>The effect of NBs on the attachment of MaBs at mineral surfaces</u>

The experimental setup shown in Fig. 1 consisted of a system whereby the MaBs were injected into a flat glass cell and microphotographs taken with a stereomicroscope (Zeiss Stemi SV11) coupled to a digital camera (Sony NEX-3). The cell was rectangular, 1 cm deep, 30 cm high and 10 cm long. A glass tubing (0.5 cm in diameter and 15 cm long) containing galena or quartz at its end was inserted within this cell. The cell contained 2 orifices for the injection of: i. NBs and flotation collector solutions; ii. MaBs. The minerals particles and materials were cleaned using a solution of hydrochloric acid (10% HCl), followed by immersion in a sulfochromic solution for 0.5 h and rinsing thoroughly with DI water. NBs aqueous dispersions were generated as described before.

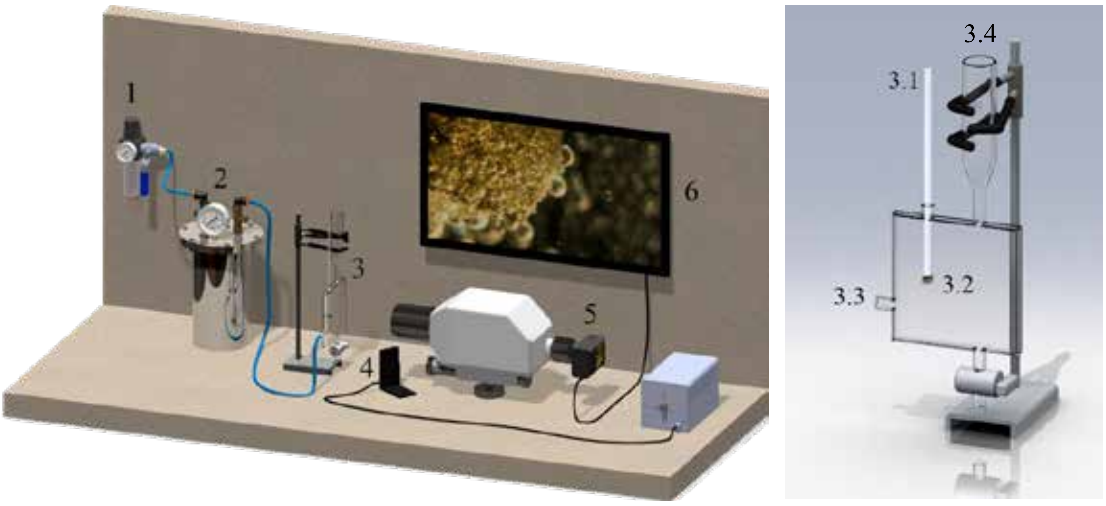


Figure 1. Experimental set-up for image acquisition of the MaBs-mineral particle interaction in the absence and presence of NBs. Left side: (1) Compressed-air filter; (2) Saturator vessel; (3) Flat glass cell; (4) White-light source; (5) Stereomicroscope coupled to a digital camera; (6) Monitor for image reproduction. Right side: Details of the flat glass cell - (3.1) Mineral grain holder; (3.2) Mineral particle; (3.3) Inlet for MaBs injection, (3.4) Inlet for NBs and flotation collector solutions injection.

<u>Aggregation of ultrafine particles</u>

We investigated the aggregation of spherical silica particles (0.1% w/w) by NBs using microscope imaging (Olympus, model BX41) and Photometric Dispersion Analysis (PDA 2000, RankBrothers®). We conditioned the particles (0.35 g) in two glass beakers (500 mL) with decil-ether-amine (1 mg amine.g^{-1} silica) and 50 mL of either DI water (control) or NBs aqueous dispersion, under magnetic agitation, for 3 min. Then, 300 mL of NBs aqueous dispersion (pH 7) was added in one of the beakers and the same volume of DI water was added in the other beaker (control). Then, these samples flowed at 40 ml.min^{-1} rate with a peristaltic pump through the optical sensor of the dispersion analyzer equipment and the aggregation index (R output) was monitored. Aliquots of both silica suspensions were collected for microscope imaging and microphotographs were taken with 1000 x magnification.

<u>Bench mineral flotation studies</u>

We performed comparative mineral flotation studies assays in a Denver cell, Model D12 (3 L) with and without MBs and NBs. For the standard (STD) trials, the conditioning of the pulp was performed under stirring (1000 rpm), for 3 min in the same flotation cell (3 L). Then, we injected the air (5-7 L.min^{-1}), during 9 min and collected the flotation concentrates using automatic scraper aid (10 collections per min), keeping the cell volume constant, with continuous water injection. We carried out the tests with MBs and NBs following the STD test procedure, but with bubbles injection by depressurization of air-saturated water at a P_{sat} of 2.5 bar, as described before. In the experiments with isolated NBs, an elapsed time of 3 min was required for the MBs to abandon the suspension, leaving isolated the NBs.

Removal of pollutants from mining wastewaters

We conducted three studies to investigate the NBs in the removal of pollutants from mining wastewaters by flotation assisted by NBs. Firstly, we investigated the removal of iron ions in the form of ferric hydroxide ($Fe(OH)_3$) precipitates. These experiments were performed in 500 mL solutions containing Fe^{+3}. The separation was carried out though the precipitation of Fe^{+3} ions at pH 7, followed by DAF at different P_{sat} values (2 – 4 bar). The flotation was carried out by the depressurization of 150 mL of air-saturated water (23% recycle rate) through a flow constrictor (needle valve).

Secondly, we studied the removal of xanthate ions by precipitation with copper sulfate at pH 9, followed by flotation with MBs and NBs. We employed rapid mixing ($G = 2264\ s^{-1}$) during 5 min and slow mixing ($G = 27\ s^{-1}$) for more 5 min. We collected a sample (5 mL) for analyzing xanthate (corresponding to point “0”). Then, after the flotation stage, the treated water samples were collected, for xanthate and residual copper analyses. The effect of the Psat on flotation rates was studied at 2.5 and 5 bar and the results were expressed as xanthate removal percentages. The flotation kinetics were calculated by fitting the data to a first-order kinetics model for batch flotation (Zuñiga, 1935). We employed an UV-VIS spectrophotometer (ThermSpectromic®, model Genesys 10) for the analyses of xanthate and for copper atomic absorption, utilizing a Varian® SpectrAA 110.

Finally, we treated a sample of thickeners overflows of an iron flotation (north Brazil) to evaluate, at bench scale, the potential of flotation with MBs and NBs. The turbidity of the samples of wastewater was about 400 NTU. We used $FeCl_3$ ($30\ mg.L^{-1}$) as coagulant and pH 8; mixing (400 rpm, 1 min) and slow mixing (60 rpm, 4 min). Then, MBs and NBs (P_{sat} of 5 bar) were injected by the bottom of a flotation cell for separation of the pollutants (gangue particles coated with amines).

RESULTS AND DISCUSSION

NBs generation and main factors involved

Figure 2 shows the effect of P_{sat} on the generation and concentration of NBs and we observed a substantial decrease in the concentration of MBs and a high concentration of NBs, at low P_{sat} 2.5 bar, especially at the lower air/water interfacial tension. The MBs formed down-stream of the flow constriction is a direct function of the amount of dissolved air in water and increases with increasing P_{sat}, when more air is available (Rodrigues and Rubio, 2007). Consequently, some NBs disappear because they are entrapped by the MBs. Because the MBs at $P_{sat} < 3$ bar were notably lower, their interaction with the bulk-phase NBs was significantly reduced, and the NBs concentration increased. Based on this, there are two forms of using these tiny bubbles, in conjunction or separated.

Figure 2 shows that the NBs increased by about 5 times from 3×10^8 to 1.6×10^9 $NBs.mL^{-1}$ upon a decrease in water/air interfacial tension from $72.4\ mN.m^{-1}$ to $49\ mNm^{-1}$ at a P_{sat} of 2.5 bar.

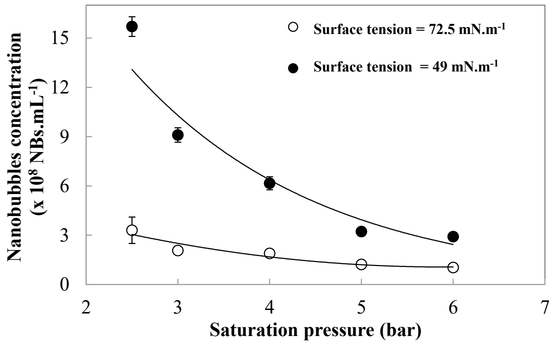


Figure 2. NBs concentration (density) as a function of saturation pressure at two aqueous surface tension values. Conditions: pH 7; surface tension of $49\ mN.m^{-1}$ ($100\ mg.L^{-1}$ pine oil); DI water surface tension of $72.5\ mN.m^{-1}$.

This phenomenon was observed in the generation of MBs by Takahashi et al. (1979) on the minimum "energy" ΔF required for bubble formation (Equation 1),

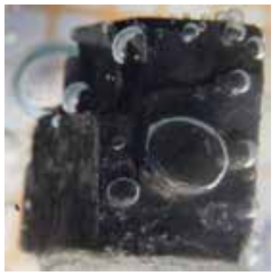
$$\Delta F = \frac{16\pi\gamma^3}{3(P_{sat} - P_o)^2} \quad (1)$$

where γ is the surface tension of the liquid, and Po is the atmospheric pressure.

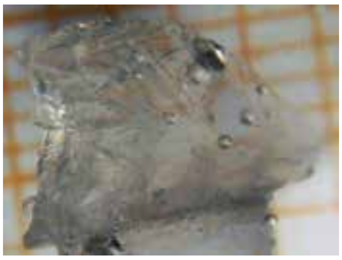
The effect of NBs on the attachment of MaBs at mineral surfaces

Figure 3 shows photographs of two mineral surfaces whereby the population of MaBs, main responsible for flotation, increases after conditioning with NBs. This finding validates results obtained by authors from our research group that the NBs have hydrophobizing (“collector”) power (Azevedo et al., 2016; Calgaroto et al., 2015). These results are extremely important for mineral flotation because NBs greatly enhance the number of attached bubbles and decrease significantly the particles-bubbles density.

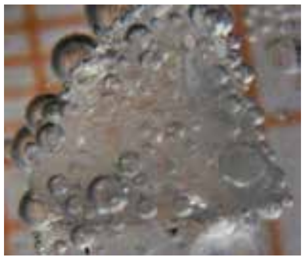
Galena in DI water - MaBs, pH 6.5



Galena in DI water- NBs+MaBs, pH 6.5



Quartz conditioned with amine [10^{-6}M] – MaBs, pH 6.0



Quartz conditioned with amine [10^{-6}M] - NBs+MaBs, pH 6.0

Figure 3. MaBs population at Galena and Quartz mineral surfaces in the absence of NBs (at the left); and in the presence of NBs (at the right).

Aggregation of ultrafine particles

In the last years, many authors (Fan et al., 2012, 2010; Sobhy and Tao, 2013) have reported the enhancement of fine flotation systems efficiency with the injection of NBs. Although these fines recovery gains have been attributed to the aggregation power of NBs, this property has not been experimentally demonstrated yet. Herein we employed photometric dispersion analysis and optical microscope imaging to validate this aggregation effect. The output signal of the R value, so-called aggregation index (AI), increased when the flow of sample was switched from the blank to the sample containing NBs (Figure 4). This result indicates that in the presence of NBs the particles increase in size and decrease in number, demonstrating the aggregation effect of the ultra-fine particles.

More, in Figure 5, we show a microphotograph of suspensions of silica with and without NBs conditioning, demonstrating the formation of clusters in the presence of NBs.

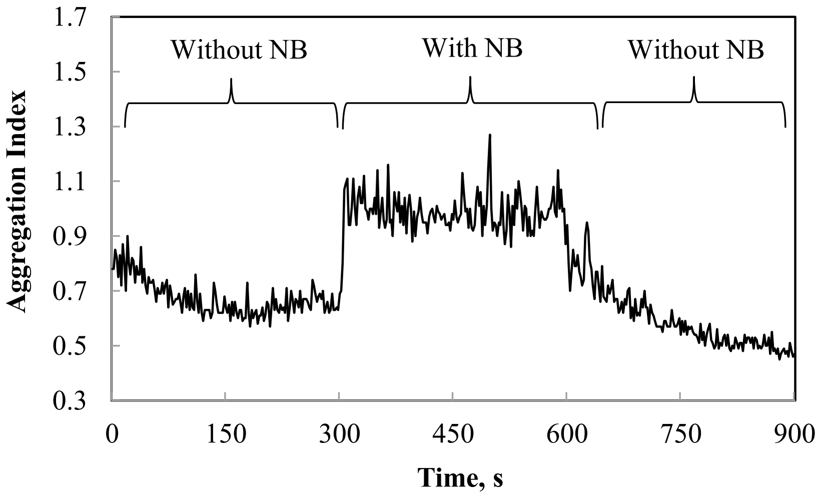


Figure 4. Aggregation index (dimensionless R parameter) of suspensions of spherical silica particles (1 µm) with and without NBs. Conditions: solids concentration = 0.1 %; [amine] = 1 mg.g^{-1} silica.

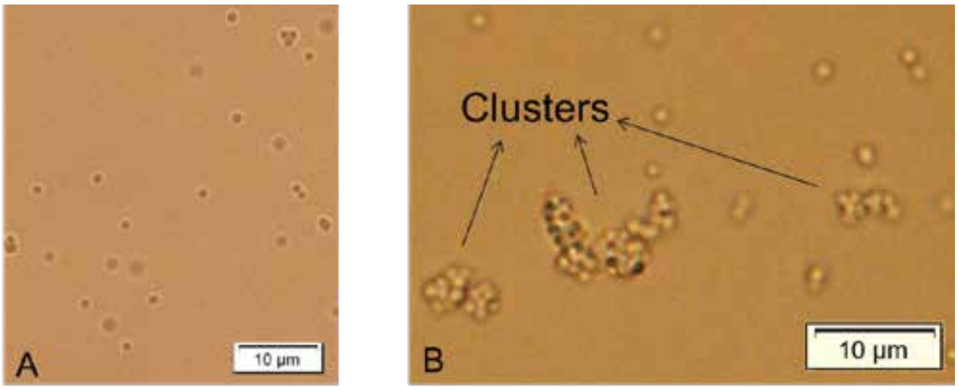


Figure 5. Microscope images of silica suspensions (0.1 % of solids) conditioned with decil-ether-amine (1 mg amine.g^{-1} silica). A) Without NBs; B) With NBs.

Flotation of minerals with MBs ad NBs

Results in Tables 1 and 2 show higher flotation recoveries efficiencies, at similar concentrated grades, when the pulp was conditioned with MBs and NBs. Results proved the afore-mentioned results of enhancement in the number of MaBs attached to the mineral surfaces and validate the potential of NBs and MBs in mineral flotation.

Table 1. Comparative flotation studies in a Copper-Au ore. Conditions: 32 % w/w; pH: 8.5; [PAX]: 18 g.t^{-1}; [AERO-MX 7020]: 10 g.t^{-1}; [Flotanol-Flomin D25 + F650]: 30 g.t^{-1}; Conditioning time = 2 min; Speed agitation = 1000 rpm; Feed Grade Au: 0.3 g.t^{-1}; Feed Grade Copper: 0.6 %.

Separation parameters	STD	MBs + NBs	NBs
Mass Recoveries (%)	3.0	3.5	3.2
Copper Recoveries (%)	69	73	69
Gold Recoveries (%)	50	56	52
Copper grade (%)	12	11	11
Gold grade (g.t^{-1})	6.6	6.3	4.7

Table 2. Comparative flotation studies in a Gold ore. Conditions: 35 % w/w; pH natural; [PAX]: 50 $g.t^{-1}$; [AP3473]: 30 $g.t^{-1}$; [MIBC]: 30 $g.t^{-1}$; [F650]: 3 $g.t^{-1}$. Conditioning time = 2 min; 1000 rpm; Gold Feed grade: 0.8 $g.t^{-1}$.

Separation parameters		STD	NBs	NBs+CRF+HIC+ extra addition of reagents *
Mass Recoveries (%)		14.3	12.2	16.4
Gold grade ($g.t^{-1}$)	Concentrates	2.7	3.4	6.0
	Tailings	0.04	0.05	0.04
Gold recoveries (%)		92	91	98

* Combined flotation: NBs plus CRF = concentrate recycling flotation; HIC = high intensity conditioning and with an extra addition (50 %) of the initial collector (in the 4^{th} min).

Removal of contaminants

Figure 6 shows the kinetics of DAF for 2.5 and 5 bar of P_{sat}. Flotation (and thereby xanthate ion removal) with P_{sat} of 5 bar was very fast (40 s), reducing the xanthate in the feed to 1 $mg.L^{-1}$. Further, the flotation rate followed a first-order model with kinetics constants of 4.3 and 5.7 min^{-1} for 2.5 and 5 bar, respectively.

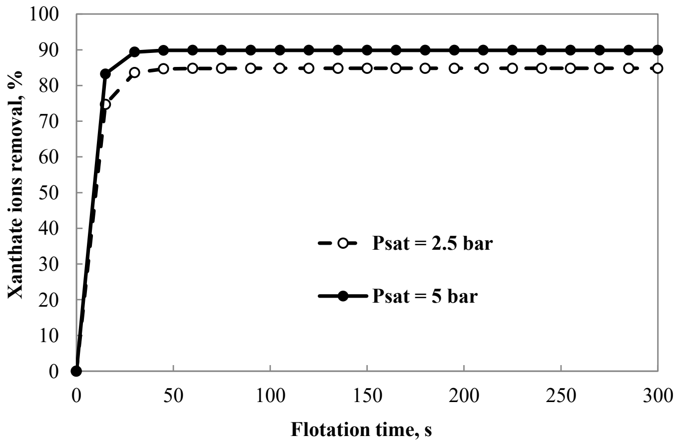


Figure 6. Flotation kinetics of xanthates precipitates with MBs and NBs: Conditions: pH 9; xanthate in feed = 60 $mg.L^{-1}$; DAF recycle ratio = 23%; conditioning time = 10 min (equimolar Cu^{+2}/xanthate).

In the case of $Fe(OH)_3$, results showed that with a higher NBs concentration, greater was the Fe removal efficiency (Table 3). The higher concentration of MBs may cause some flocs breakage due to the higher rising rate to the surface of the liquid. With more concentrated NBs, the latter's attach and adhere onto the hydrophobic colloidal precipitates; aggregate them and entrain inside before rising by flotation. These mechanisms operate simultaneously and help to improve the separation with MBs + NBs.

Table 3. Iron ion removal, in the form of precipitates by flotation with MBs and NBs.

Iron, feed concentration, $mg.L^{-1}$ (values ± ½ standard deviation)	Iron final concentration, $mg.L^{-1}$ (values ± ½ standard deviation)	Treatment method details
30 (0.7)	1.1 (0.02)	[1]Precipitation at pH 7; Flotation at 4 bar (less concentrated NBs)
	0.9 (0.05)	[2]Precipitation at pH 7; Flotation at 2 bar (more concentrated NBs)

The application of coagulation-flotation with MBs-NBs, for the treatment of water overflows of the iron flotation thickeners, resulted in the reduction of turbidity of this water of 95% (Table 4). By removing the suspended solids, the content of amine and other compounds is reduced. Visually, we observed a significant lower foaming (amine presence) compared to that of the process water.

Table 4. Removal of turbidity and fines solids from the process water of the overflow of flotation thickeners by flotation with MBs and NBs

Feed turbidity (values ± ½ standard deviation)	Turbidity (values ± ½ standard deviation), after flotation	Treatment method details
400 NTU	21 NTU	Coagulation with $FeCl_3$ (30 mg.L^{-1}) at pH (8.5) followed by flotation at a P_{sat} of 5 bar

In addition, our group have published successful examples of removal of ions (sulfate-which generate mining acid drainages) by flotation with MBs and NBs (Amaral Filho et al., 2016) and residual amine (Calgaroto et al., 2016), a toxic reagent used in the iron industry and dumped after used as collector for quartz (reverse ore flotation).

FINAL REMARKS AND CONCLUSIONS

Millions of tons of ores are lost in mineral flotation operations whereby the grinding is intense to liberate the valuable minerals, generating the difficult-to-treat fines (<74 µm) and the ultrafines (< 10 µm). Many theoretical studies and alternatives have been studied for decades and the problem continues (Gontijo et al., 2008; Jameson et al., 2007; Rubio, 2003; Sivamohan, 1990). The actual tendency to overcome this problem is to employ fine bubbles and as in this work, we focused on the flotation metal sulfides, in the presence of MBs and NBs. These bubbles quickly adhered to hydrophobic particles; aggregated the fine/ultrafine fractions and allowed the attachment of MaBs. Results obtained validated the theory behind the use of small bubbles, mechanisms were discussed and results proved the potential of the NBs in flotation.

On the other hand, mining and mineral processing activities consume intensively water resources; and reducing this usage is an important strategy towards a more sustainable mining industry. Values of water consumption vary from ore to ore and in the metal sulfide area is around 0.8 m^3.t^{-1} of ore processed. There is a growing attention to this issue and many researchers and companies have identified forms of water reuse options; even some have been already implemented around the world (Bahrami et al., 2007; Wessman et al., 2014). Saving strategies include evaporation reduction, paste tailings disposal, filtered tailings disposal, treatment of overflows of thickeners and hydrocyclones and tailings dams. In addition, water losses occur when they contain residual flotation reagents and ultrafines solids, which usually affect flotation circuits due to undesirable effects, decreasing the process efficiency. We believe that treating and reusing this process water is a need, especially in arid regions. In our group, we were able to show successful removal of precipitates, colloidal dispersions, ions and residual reagents (Amaral Filho et al., 2016; Calgaroto et al., 2016).

Research needs for a future scenario, involving NBs (and MBs) are the following:

1. Treatment, by flotation, of effluents generated by the mining and mineral processing operations, namely AMD-acid mining drainage or ARD-acid rock drainage (Silva and Rubio, 2009), SBW-sulfate bearing waters (Amaral Filho et al., 2016), Molybdate ions in copper-moly sites (Rubio et al., 2002);

2. Wastewater treatment and reuse, to decrease water consumption. This would imply in the removal of oils from machineries, maintenance services and the like. We studied and provided similar examples in Zaneti et al. (2013);

3. Removal by flotation of colloidal precipitates, collectors (xanthates and derivatives, amines, fatty acids), ions, iron, copper, lead, zinc, fosfates, arsenic, frothers (Rubio et al., 2002; Santander et al., 2011).

Figure 7 depicts how to inject MBs or NBs, in a sustainable manner in future column flotation operations.

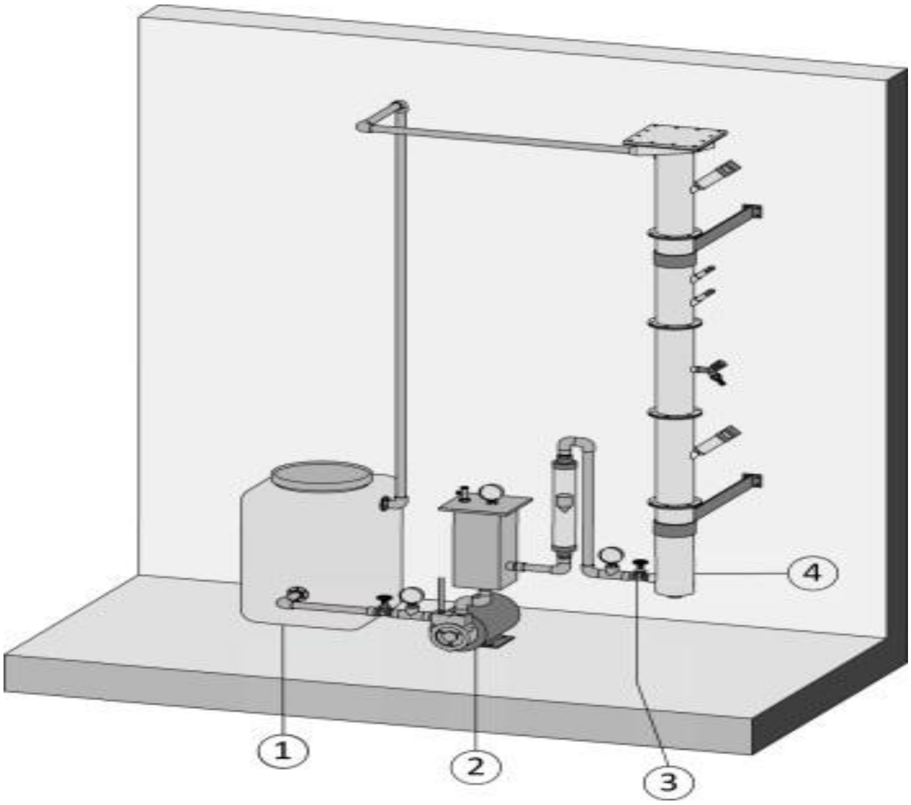


Figure 7. Flotation column with MBs and NBs generation by a centrifugal multiphasic pump and flow constrictor. 1- Feed water; 2 – Centrifugal multiphasic pump with air injection; 3 – Flow constrictor (needle valve); 4 – Column.

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FOSTERING INTERNATIONAL COOPERATION IN THE FIELD OF RAW MATERIALS – THE INTRAW PROJECT AND THE EUROPEAN INTERNATIONAL OBSERVATORY FOR RAW MATERIALS

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FOSTERING INTERNATIONAL COOPERATION IN THE FIELD OF RAW MATERIALS – THE INTRAW PROJECT AND THE EUROPEAN INTERNATIONAL OBSERVATORY FOR RAW MATERIALS.

ABSTRACT

In the last decade, a structural change has taken place in the world's mineral markets and the global demand for raw materials stands at the bottom of a new growth curve. In this perspective, safeguarding the domestic minerals supply in a sustainable way will be challenging not only to Europe but also to other countries such as Australia, Brazil or the United States. As part of the European Commission's Horizon 2020, the International Cooperation on Raw Materials (INTRAW) project was launched in 2015 under the coordination of the European Federation of Geologists.

Since then and in line with the European Raw Materials Initiative and the Strategic Implementation Plan (SIP) of the European Innovation Partnership on Raw Materials (EIP-RM), the INTRAW project has been working on mapping national best practices, policies and strategies aiming to maintain a competitive and functioning economy and non-energy minerals industry (primary and secondary raw materials). INTRAW´s work has been focused in three domains: Research and Innovation, Education and Outreach, and Industry and Trade of five technologically advanced partner countries: Australia, Canada, Japan, South Africa, and the United States of America. Findings during the first year of our research show the leading role of Australia in long-term economic growth and of Western Australia in automation, of Japan in deep sea mining, substitution and raw materials diplomacy or of Canada in setting the pace of exploration.

The outcome of the ongoing mapping and knowledge transfer activities will be used as a baseline to set up and launch the European Union's International Observatory for Raw Materials as a definitive raw materials knowledge management infrastructure. The Observatory will be a permanent international body that will remain operational after the end of the project aiming at the establishment of strong long-term relationships with the world's key players in raw materials technology and scientific developments.

KEYWORDS

Raw materials management, best practices, international comparison, mineral policies, mineral strategies

INTRODUCTION

In the last decade a structural change has taken place in global mineral markets. The old rule of thumb – 20 percent of the world population in Europe, United States of America (hereinafter USA) and Japan consuming more than 80 percent of the total minerals production – is not valid any more. With the integration of India, the People's Republic of China and other populous emerging countries like Brazil and Russia into the world economy, today more than half of the world's population claims an increasing share in raw materials. Thus the global demand for raw materials stands at the bottom of a new growth curve. It is assumed that by 2030 the worldwide need for raw materials will have doubled. A doubling of raw materials demand for the important industrial metals is plausible in view of the relevant predictions, e.g. of the global motor vehicle stock. Some projects estimate that it will rise from approx. 800 million in 2002 to more than 2 billion light duty vehicles by 2030 (IEA, 2015), whereas in non-OECD countries an increase by 195 million to 1,172 million is expected, i.e. almost all growth is in developing countries. Estimates by the OECD Development Centre suggest that today's development and emerging economies are likely to account for nearly 60% of world GDP by 2030 (OECD, 2010). Raw materials technology developments are of increasing importance as a motor of growth in the emerging countries.

Access to raw materials on global markets is one of the European Commission's priorities. Over the last decade the European Union (hereinafter EU) has become much aware that securing a reliable, fair and sustainable supply of raw materials is important for sustaining its industrial base, an essential building block of the EU's growth and competitiveness. This was triggered by the increasing demand for unprocessed minerals and metals, volatility in the prices of certain raw materials, as well as market distortions and supply risks imposed by a number of countries involved in the trade of raw material commodities (e.g., resource nationalism in China for rare earths). Given the global nature of raw materials value chains and the afore-mentioned global challenges, finding solutions to ensure a level playing field and a fair and unrestricted access to raw materials for the EU and for non-EU countries requires international cooperation on raw materials.

Efforts to foster international cooperation are being made by the EU framed under the Strategic Implementation Plan (SIP) of the European Innovation Partnership on Raw Materials (EIP-RM) (https://ec.europa.eu/growth/tools-databases/eip-raw-materials/en/content/european-innovation-partnership-eip-raw-materials). The third pillar of the SIP refers to International Cooperation and promotes Global Raw Material Governance Dialogues and Raw Materials Diplomacy, e.g. the on-going dialogues between Europe and Latin America. The International Cooperation on Raw Materials (INTRAW) project, running during the period 2015-2018, has been formulated with the objective of mapping best practices and boosting cooperation opportunities on raw materials with technologically advanced non-EU reference countries (Australia, Canada, Japan, South Africa, and the United States) in response to similar global challenges. The ultimate goal of the project to set up and launch the European Union's International Observatory for Raw Materials as a definitive raw materials knowledge management infrastructure.

In this paper we present summarized results of the contextual analysis conducted for each of the reference countries and available in five Country Reports (Falck, Akhouri, & Murguía, 2015; Murguía, 2015a, 2015b, 2015c, 2015d). The information in the Country Reports was collected across three pillars: Research and Innovation (R&I), Education and Outreach, and Industry & Trade; in this paper we present insights of these three. The last pillar also encompasses an analysis of the key drivers of mining success across the reference countries (except Japan). The analysis is based partially on literature such as the Fraser Institute´s reports (Jackson & Green, 2015) or the annual ranking by the Behre Dolbear group (Wyatt & McCurdy, 2014). Results highlight for each of these pillars which good practices each country has conducted in order to be economically successful at a general level, and in relation to the management of minerals. Finally, a summary of the plans for the launch of the European International Observatory for Raw Materials are presented with the feedback on the country reviews received by a Panel of Experts.

Based on the comparison of results across countries, we aim to provide guidelines for the both EU and non-EU mineral-rich countries (such as Brazil) as to what are the historical and current key aspects that should be carefully looked into if a successful and sustainable mining industry is to be achieved.

METHODOLOGY

This technical paper arises from the research conducted for the INTRAW project on a country-level for 49 indicators with the aim of understanding the historical context and identifying the key drivers enabling the success of the mining industry, all of which are presented in detail in the "Country Reports" previously mentioned. These reports were created in an iterative way by the Consortium Partners, based on literature and local knowledge of the country by each expert and supporting teams. Preliminary results of the research were discussed with a Panel of International Experts during the Bled Workshop held in September 2015 in the city of Bled, Slovenia. This paper is built upon the afore-mentioned Country Reports, collects selected information, re-organizes it, and presents a comparison of main results.

RESULTS

Research and Innovation

Since the steel-based industrial revolution of the late 1890s, the USA has joined the ranks of world leaders in innovation. Government and industry-funded institutions have been developing throughout the 20th century which has given the country a solid R&D infrastructure, including government-funded labs, high-tech profile innovation clusters like Silicon Valley, and many others. Econometric studies strongly suggest that R&D spending has a positive influence on productivity, with a rate of return that is likely to exceed that on conventional investments. After the USA, Japan ranks 2nd in the world in terms of total expenditure on R&D with a 3.3% of the GDP. The USA also leads the world rank in business expenditure on R&D, followed by Japan in 2nd position. The USA features as a highly attractive destination for researchers and scientists, ranking 2nd in the world after Switzerland. Japan ranks 2nd position in the world after China with regards to the total amount of personnel working on R&D. All of the above show that the knowledge and resource base (infrastructure) in the USA and Japan has been of high importance in their transition towards a knowledge-based economy. Canada and Australia have a well-established science, technology and industry systems, even though they do not rank amongst the top ten countries in terms of R&D personnel, expenditure on R&D and attractiveness to researchers & scientists. South Africa also has a well-developed science system, which has been developed in relative isolation due to sanctions and since the end of apartheid has been reconnected to the world's developments.

The USA, Japan and South Africa have strong R&D cultures. Compared to many nations, the USA has a highly developed and successful industry-research institute collaboration system. USA companies are highly sophisticated and innovative, supported by an excellent university system that collaborates admirably with the business sector in R&D. Japan´s R&D culture developed during the 20th century and was led by technology transfer process from the West to Japan during the catch-up period and afterwards when Japan took the lead in innovation. R&D in Japan is mostly (70%) financed by the private sector. Knowledge transfer between companies and universities is low in Japan, and start-ups are still low in comparison to other peer countries. Japan´s national innovation system is now in a transition towards science-based industries and the R&D culture is still maintained as a key to resume economic growth. Despite South Africa's strong R&D tradition, it's Gross Expenditure on Research and Development (GERD) of 0.8% is below that of other emerging economies with much of the research being business-driven. Australia and Canada rank poorly internationally in innovation and do not have strong R&D cultures. Canadian companies are thus rarely at the leading edge of new technology and too often find themselves a generation or more behind the productivity growth achieved by global industry leaders. University-industry collaborations are not well-developed. However, unlike the general trend, the Canadian mining industry is a global leader in capital investment, financing and innovation. Australia has a moderate to low performance on innovation due to a poor business innovation culture. Yet its innovations in the mining sector have been and remain of significant importance (as driver of productivity increase).

The USA has an excellent track record at continuously investing in geoscientific data and related research as they are considered critical factors enabling the development and growth of the mining industry. The U.S. Geological Survey was established in 1879 to determine the natural wealth of the country, and has continued to serve that role. Likewise, the U.S. Geological Survey and the U.S. Census Bureau closely monitor production and consumption of an extensive range of commodities, which provides critical economic intelligence regarding the viability of operations. The information acquired and published by the USGS is internationally considered reliable and their data and publications are freely available and are amongst the most widely used around the world for mineral statistics. The availability of excellent geoscientific data enables small and medium sized mining companies to compete with larger enterprises, who tend to create and maintain their own knowledge bases in addition to the publicly available data.

Raw materials-related R&D is conducted in South Africa by public and private partners. Mintek is one of the world's leading technology organisations specialised in mineral processing, extractive metallurgy and related areas. The Council for Geoscience (CGS) and the Council for Scientific and Industrial Research (CSIR) are two of the National Science Councils of South Africa. One of the mandates of the CGS is to develop and publish world-class geoscience knowledge products and to render geoscience-related services to the South African public and industry. The CSIR mining research is focused on mineral extraction, particularly underground mining of gold and platinum: breaking and moving rock (drill and blast) safely and efficiently, and in increasing the safety of operations for workers to a maximum. Besides public agencies, much R&D is conducted in South Africa in the mining field via private sector involvement. For instance South Africa's gold and platinum mining industry works at greater depths and under more difficult conditions than any other mining industry in the world.

Canada is also heavily invested in geoscientific data but unlike the USGS (central body), the data acquisition and related research is done by provincial geological surveys, with the data made publicly available. The mining industry has continuously invested in R&I, even during bust times and high risk, like in the 1990s when the sector invested in new, automated technologies that enhanced the economic viability of projects by lowering production costs. The sector has undergone a profound change to a high-tech industry, and it has become a driving force in Canada´s new knowledge-based economy. New technologies in mining have created a circle of growth and innovation that circulates through two-way linkages between mining and the rest of the economy. A large part of innovations in the mining industry takes place in the exploration sector. Vancouver is the global centre of expertise for mineral exploration, with some 1,200 exploration companies located in the British Columbia, most of which are in the greater Vancouver area. Also in exploitation, innovation plays a central role. For instance in the 1960s R&I in reliable transport infrastructure (e.g. development of the ice roads technology) granted access for exploration first, and development later, of diamond orebodies in the Northern Territories. A more recent example is given nowadays by much of Canada's remaining base metals, which are likely located two kilometres or more beneath the surface. This situation presents cost and operational challenges. In response, the industry is investing in remote-operated equipment, automated loading and transportation systems, robotics and seismic mapping. Many of these innovations have been funded by the industry, whereas government-funded research mechanisms are not aligned and it is claimed that they do not provide support commensurate with the industry's innovation needs, priorities and contributions (Marshall, 2014).

Australia has traditionally maintained a high level of investment in R&I in the mining sector. One of the most significant Australian innovations was the development of flotation (the most effective method of separating minerals from the gangue) in Broken Hill in 1903, and is a method widely used in the international metal mining industry. Similarly influential was the development of heap leaching of low grade ore bearing gold in Western Australia. Nowadays Australia keeps pioneering R&I to increase productivity and cost control and the Pilbara region acts as the main focus for innovation activity in mining. In this region several companies, including some from the mining equipment, technology and services sector, are testing and running automation technologies such as driverless haul trucks, automated wheel changers for haul trucks, remote train and ship loading, remotely operated drill and smart blast activities, as well as the development of a new class of tunnelling machines for underground mines. All such innovations will continue to provide Australia with a competitive advantage in the field of mining.

Education and Outreach

The INTRAW project undertook an analysis of mining & raw material supply education provision and skills availability in Canada, USA, South Africa, Australia and Japan, their national workforce demands, perceptions of skill levels/qualities, and funding. These technologically advanced non-EU reference countries were chosen for mapping best practices and boosting cooperation opportunities between the EU and other technologically advanced countries. During the last decade, skills shortages have been arguably the mining industry's most significant problem. The analysis examined formal programmes of adult education leading to diploma, certificate, degrees, further and higher education qualifications, or vocational education programmes including apprenticeships, all of which lead to a formal award. Outreach consisted of a range of initiatives, courses, organisations and schemes specifically aimed at certain sections of society, especially those from historically disadvantaged, ethnic, or native populations who wish to learn more about mining and gain access to jobs in the mining industry.

Reviews were performed for each target country to provide data on courses, student numbers, centres of excellence, delivery mechanisms and, where possible, staff numbers. The internationalization of education and barriers to access were also analysed. Important educational and human resource issues that the analysis identified included gender imbalances, minority groups, local community skills development support, university recruitment and company strategy. The analysis revealed a number of key elements that characterise mining education across these advanced raw material producing countries.

Mining is undertaken in over 100 countries and it is estimated that on a global basis the formal mining sector employs more than 3.7 million workers. Although every mine is different, the organisational structure of a mining company generally includes senior corporate staff, managers, university-educated specialists, supervisors, and operational staff and skilled and semi-skilled staff and associated contractors.

Today’s mining industry relies on highly skilled workers with a diverse skill set, the ability to use sophisticated technology and operate in challenging environments. It typically seeks skilled operators, graduates and technical specialists with not just mining knowledge but also digital literacy, problem solving ability and good interpersonal skills, who can work safely in both a team and individual capacity. Staff require initial training and must also continuously upskill as technology advances. Benefits to staff include operating in an environment where safety is paramount and enjoying pay and conditions that are highly attractive.

Mining education encompasses a wide range of education and training options that can be accessed by students seeking to enter the industry, mature entrants reskilling, in-work employees’ upskilling and even those taking courses purely for interest. Universities offer a range of mining focused undergraduate degree options around applied geology, mining engineering, mineral processing and metallurgy, as well as a raft of generic but relevant subjects in engineering, business, environment, etc. At postgraduate level, more vocational Masters courses are offered, focused on mineral exploration, mining geology, geotechnical engineering, geometallurgy, construction, minerals engineering, minerals processing, metallurgy, and environmental impact. Advanced study through research Masters and PhDs are an important provision in many universities and are typically based on industry-related scientific questions, undertaken in collaboration with or sponsored by individual companies, or as part of larger collaborative research programmes. Universities may also have a range of continuing professional development (CPD) short courses, professional programmes and some distance learning provision accessed by those in the industry to address specific skill needs or for professional development.

Training for technician and administrator levels in mining related areas are usually delivered by technical colleges and training centres. These usually involve a combination of conventional teaching with placements and work-based learning, and include technical, commercial and clerical provision. Vocational training courses including artisan and trade skills (e.g., welders, drillers, maintenance mechanics, electricians, loader operators etc.) usually have many more training routes through both mining and other sector training organisations and are typically delivered by apprenticeships involving block or day release, combined with workplace skill development. Mining investment in efficiency, mechanisation and automation will push up the required skills levels and reduce the opportunities for low skill jobs.

Volatility in the industry and increased resource nationalism, as well as demand of producer countries for local staff to take over the more senior roles, is leading to a need for rapid upskilling and loss of experienced international staff. The cyclical nature of the industry has caused endemic skills shortages followed by oversupply, which lags the industry cycles and results in elevated costs and loss of experience from the industry. Employers need to consider funding, retaining and upskilling staff through the downturns and this may require new models of employment.

Real-time skills and employment data are not easily accessible and new methods are needed if prediction through the cycle is to be realistic. There appear to be few absolute skills shortages, perhaps in mining engineering and mineral processing but not critically so. Criticality is caused by timing of availability as the training duration lag time reinforces the skills shortage as the upturn develops and compounds the oversupply at the next downturn. Training needs to be more aligned with industry cycles –good practice evidence is available but there is a need for more creative solutions to in-work education and industry-education partnership arrangements.

In aiming to address educational issues in the mining industry, in addition to the key findings outlined above, the analysis has produced a list of possible metrics to benchmark and compare EU countries against and form the basis for action plans:

- Number of universities teaching mining/minerals geoscience
- Length of programs and quality of curriculum (including staff: student ratio)
- Number of students and demographics
- Amount of mining/minerals geoscience in secondary school curriculum
- Number of mining education organizations and membership
- Training data and workforce shortages
- Qualification requirements

- Others metrics identified in reports by the Mining Industry Human Resources Council - Canada (MiHR), Society of Mining Professors (SOMP) and national workforce planning exercises

Industry and Trade

Trade inter-dependencies

During the second half of the 20th century the international trade of raw materials, importantly of metals and minerals, expanded remarkably and it consolidated as a principal driver behind the economic growth of industrial economies such as the USA or Japan. Dramatic decreases in transport and communication costs coupled with reductions in trade barriers have been the driving forces behind today's global trading system. Nowadays, by weight, coal and iron ore are by far the most extensively traded minerals. The USA and Japan were key players shaping the international trade in mineral raw materials and their economic success was and is closely interrelated with those supply partners such as Canada, Australia and South Africa, amongst others. Special mineral trade bilateral partnerships during the 20th century and which still today remain very important were those of USA-Canada and Japan-Australia.

- ***USA-Canada partnership***: due to geographic proximity and similar historical cultural characteristics, the USA and Canada share a history of economic development based on the domestic use and bilateral trade of mineral resources. Actually, for many years, the USA-Canada border has been the longest in the world and both countries have been sharing the world's most comprehensive bilateral trade relationship. In 2013 U.S. exports to Canada accounted for 19% of overall U.S. exports, in turn, Canada's exports to the U.S. accounted for a 75% of all exports (2014). In 2013 and concerning mineral commodities (ores, concentrates, and semi- and final-fabricated mineral products), 50.2% of mineral Canadian exports were sent to the USA and 50.8% of mineral imports came from the USA.

- ***Japan-Australia partnership***: main enablers of strong raw material trade relationships between both countries was the creation of bulk transport vessels lowering transportation costs, the adaptation of port facilities, and the bilateral agreements on commerce and business, with the Commerce Agreement signed between both countries in 1957 (the first trade agreement of Japan after WWII). More recently it has been boosted with the Japan-Australia Economic Partnership Agreement (JAEPA) entering into force in January 2015. The main historical drivers of the boom were coal and iron ore exports enabled by discoveries in the Pilbara region in Western Australia and by other metals. Long-term contracts were here key enablers, e.g., by guaranteeing Japanese demand and providing a secure base for investment by the Australian partners. Notably since the 1960s (Siddique, 2011), Japan has traditionally imported strategic resources (iron ore, coal, manganese) and exported vehicles and machinery to Australia. Australia´s mining sector has heavily relied on foreign investments, and Japan has been injecting capital in investments for over half a century. Nowadays Japan depends on Australia for around 60% of its imports of coal and iron (Australia´s first and second most valuable exports), and Australia is Japan´s main supplier of natural gas and uranium. Australia's economy has grown strongly in tandem with trade with Japan and its mineral demand is synchronized to the business cycle of Japan, China and India (Liew, 2012). Japan buys more than any other country of five of Australia's most valuable merchandise exports: coal, aluminium, natural gas, bovine meat and copper ores (Australian Government Department of Foreign Affairs and Trade, 2008).

In a globalized market economy, all of the partner countries have based their minerals supply and demand on other multiple partners with smaller but significant shares. The USA is a good example of changing and multiple sources for supply combining domestic extraction and imports. The domestic endowment of natural and mineral resources (e.g., coal, wood) was highly important in the early phases of the industrialization, but then the economy began a transformation process towards a knowledge and services-based economy in which the availability of domestic resources became less important, i.e. supply could be covered to a large extent by mineral imports. Around 1950 the USA became a net importer of minerals (Lindert, 2000), rubber and forest products (Palo, Uusivuori, & Mery, 2012) and in 1958 the USA turned from a net exporter of fossil energy carriers to a net importer, and by 1973 already 20% of all fossil energy carriers and one-quarter of all petroleum and natural gas was imported. Net imports of ores and metals began to increase in the late 1940s with imports of non-metallic minerals rapidly rising in the 1970s. Currently the USA, which is inhabited by 5% of the world´s population, is the world´s largest economy and consumer of natural resources using roughly 20% of the global primary energy supply and 15% of all extracted materials (Gierlinger & Krausmann, 2012).

In Japan, the government and the mining industry have been historically closely interrelated. Japan's post-WWII high growth era and its sustained economic and industrial development was enabled by a dynamic mineral resources policy which ensured that the Japanese industry secured a stable supply of raw materials to overcome its extreme import dependency of minerals (Japan's import dependency in 2012 was 99.6% for oil, 97% for natural gas, 100% for copper ore, and 99.3% for coal). The latter encompassed not only securing the supply of primary raw materials via agreements with countries and direct investments by private capitals in overseas mines and in exploration in Japanese offshore resources. Private Japanese capitals have invested in over 40 iron, nickel, copper, zinc and gold mines in Southeast Asia, Australia, North and South America, and Africa. For instance, in Brazil, owner of 80% of the world´s niobium reserves, JOGMEC, Japanese steel companies and Korean companies have invested USD 1.95 billion to acquire 15% of Cia. Brasileira de Metalurgia e Mineração, the world's dominant producer of niobium.

Nowadays the government administrative agency Japanese Oil, Gas and Metals National Cooperation (JOGMEC) is a key actor in the Japanese resources policy. With a worldwide network of 13 offices, JOGMEC leads a multi-faceted strategy and permanently supports the domestic and overseas development of the minerals industry, both primary and secondary, fostering innovation and cooperation. Such a strategy encompasses joint operations, training for experts, providing equity capital and loans and liability guarantees for metal exploration and development by Japanese companies, conducts overseas geological surveys to help Japanese companies secure mineral interests and to support their exploration projects, among others. Also, Japan is investing much in offshore exploration for energy and minerals; for instance in 2014 JOGMEC pioneered the signing of the world's first cobalt-rich ferromanganese crust exploration area contract with the International Seabed Authority and secured exclusive interests for Japan.

South Africa has also been closely related in minerals trade with Japan as supplier of essential minerals such as chromium, manganese, cobalt, vanadium, and PGMs of which South Africa hosts 95% of world reserves (USGS, 2016). Japan is, in fact, South Africa's third largest trading partner and over 100 Japanese companies have a presence there. Japanese private investments have been led towards the metals sector, and recently JOGMEC has invested in a number of exploration joint ventures in South Africa particularly in nickel and the PGMs (including a joint venture with Platinum Group Metals Ltd in Waterberg and the joint venture in the Stellex North Project) and a rare-earth joint venture in Malawi.

Selected key drivers of mining success

Another aim of the contextual INTRAW Country Reports was the analysis of *key drivers* behind the success of the domestic non-energy extractive industry in all mineral resource-rich countries, with the exception of Japan which is often considered a mineral resources-poor country (excluding in-use stocks) and does not host a domestic mining industry. Even though the historical circumstances of each country differ and the mining industry evolved adapting to internal and external situations, the INTRAW project identified a series of common drivers. The most important are summarized below.

Exploration phase:

It is widely recognized that mineral endowments are one necessary but not sufficient condition for the extractive industry to flourish. The mineral endowment is closely related to prospecting and exploration activities and investments which render mineral occurrences into discovered deposits. During exploratory phases, key drivers (not exclusive of the exploration phase) are:

- ***Availability of public and reliable geoscience data***: Canada´s federal government (and the provinces too) have traditionally provided funding for public geoscience information on the premise that good economic government policy requires a sound knowledge of Canada's mineral potential. The public availability of up to date bedrock geological maps, regional geochemical, geophysical and geospatial data (in a repository) and reports, reduce the cost and risk of exploration by allowing companies to identify areas of high mineral potential, reducing the need to spend time and money exploring less prospective ground. In addition, geoscience information also informs government policy decisions in respect of land use planning, infrastructure development and environmental protection. This information is critical to reduce the financial risk associated with exploration decisions. In Australia, the Australian (Commonwealth), State and Territory governments undertake various geoscience programmes to support mineral and petroleum exploration. These programmes acquire and make available precompetitive geoscience information and datasets, particularly covering important areas, as a basis for exploration.

- ***Well-developed and dynamic exploration cluster***: Canada slipped from the world's top destination for exploration spending in 2012 to the second spot behind Australia in 2013. Canada has been particularly successful in fostering the development of junior mining firms that focus on exploration, and this contingent of junior firms (based in Toronto or Vancouver) has created a number of key advantages for the Canadian mining sector as a whole. Canada has also been successful in the collection of a cluster of around 2,000 junior mining exploration firms. A key reason for the establishment of this cluster has been the Canadian tax and finance system which has provided junior firms (via the Canadian Mineral Exploration Credit and the above mentioned flow-through shares), with capital that they could otherwise not obtain from banks, which tend to be averse to exploration given that it represents high-risk investment.

<u>Exploitation phase</u>

- ***Politically and institutionally stable framework***: this involves a high respect for the rule of law and security of tenure, attractive for mining investments from domestic and international sources. It has been and remains a key factor for the USA, Canada and Australia, less for South Africa which has faced more instability. However, in spite of the significant social problems, South Africa remains one of the most stable countries on the African continent with indices indicating a favourable business and investment environment. The USA has had stable mineral laws for over 100 years and a well-defined protection of property rights. Likewise Canada and Australia have traditionally been considered low (political) risk environments for mining investments. Canada as a country and most of its provinces have consistently been placed very high in the Fraser Institute´s ranking for the most attractive locations for mining investment; Australia has been consistently considered the world´s most secure location for mining investment in the Behre Dohlbear ranking during the period 2005-2013 (Wyatt & McCurdy, 2014).

- ***Access to land, energy and water***: in Australia access to land for exploration and development has been enabled by the ownership rights scheme (mineral rights are held by the state, making ownership and access negotiations easier). By engaging in negotiations with Indigenous Australians to secure land access agreements sufficient energy and water infrastructure is now available. In South Africa, at present, with increased population numbers and as a result of urbanisation the supply of energy and water are constrained. This challenge is recognised by the South African government, and efforts are being made to increase the energy output.

- ***Efficient permitting procedures***: the Canadian permitting procedure for mining is considered stringent but very effective with a permitting delay of around two years, similar to Australia, and lower than that for the USA (average of ten years) (SNL Metals & Mining, 2015). Also permitting times in South Africa have acted as an incentive, i.e. permitting takes on average currently 12 months for exploration licenses, and the conversion between the exploration and the mining permit is straightforward, providing security of tenure. In Australia fast and low cost permitting has been a key driver of the mining industry; recent best practice examples are given by the Nova (nickel-copper) or the DeGrussa (copper-gold) projects (Murguía, 2015a). It is also important to note the role of the government, proactively working with exploration companies to ensure that documents are appropriately prepared for lease and permit applications. Usually the government assigns case managers (or project officers) to complex projects (a service called comprehensive case management services in the Department of Mines and Petroleum, WA, or a "one stop shop" in South Australia). The case manager works closely with the company to assist in the resolution of bottlenecks and to negotiate agreed approval timelines across government (Tyne, 2015).

- ***Granting of the social licence:*** Canada´s mineral sector has seriously considered the issue of the social licence and understands that dealing with this issue requires ongoing dialogue between the agencies in government that issue the mineral tenure and control the land use management regime and industry and local communities and other agencies of government to ensure all stakeholders understand at all stages the nature of the work and possible outcomes of the mining investment. Australia has had a different situation. On the one hand, Australia-based companies have seriously engaged in leading best available practices and mitigating the social and environmental impacts of the mining industry. On the other hand, Australia has also benefited from the location of most major mining centres in remote areas, away from major population centres and areas of intensive agriculture. This has been influential in reducing community objections to mining. The social licence issue remains a challenge in many states of the USA. Although this differs per state, the nation-wide overall perception is that the USA does not view itself as a mining country anymore and the public view of mine operations is generally negative, mostly because of ongoing impacts from abandoned mines from the 19th and early 20th centuries.

- ***Skilled workforce***: all four considered countries have had and still have sound educational institutions training the workforce. The USA, for instance, has a solid, well-educated general workforce, and a large (though ageing) workforce in the geosciences (Gonzales & Keane, 2010). The mining specific workforce is much more limited and has been declining for generations as the size of the necessary mining labour pool has shrunk, but this is expected to be solved with hiring of non-U.S. professionals. In Australia the lack of skilled workforce in the country has been traditionally resolved via employment-based immigration, and local shortages have been overcome using fly-in fly-out or drive-in drive-out schemes. For more than a decade, the Canadian mining sector has been involved in a skills shortage process with fierce competition from companies in other countries. The industry, educational institutions and governments are coordinating efforts to address this challenge (e.g., Canada Job Grant Programme by the federal government, the introduction of a Federal Skilled Worker category to recruit trained workers from abroad, etc.) but it is not yet clear whether efforts will successfully bridge the increasing shortage.

These key factors are general and may not be representative of all the states or regions in each country. However, they have been agreed by a large number of experts and represent a good guidance point for investors and governments seeking to improve the framework conditions of the industry alike. Other key factors have been identified and can be consulted in the Country Reports.

European International Observatory for Raw Materials

The Observatory will be launched before 2018 aiming at the establishment and maintenance of strong long-term relationships with world key players in raw materials technology and scientific developments. It will be part of the EC raw materials infrastructure that will remain operational after the project completion. In terms of functions the Observatory is relatively broadly defined in the INTRAW Grant Agreement as "*a permanent body that will ensure improved co-ordination of an effective research and innovation programmes, funded research activities, and synergies with international research and innovation programmes for the EU raw materials sector*."

This relatively broad definition provides sufficient room to fine-tune the Observatory concept to match present and expected future requirements. In doing so the project relies extensively on the suggestions and recommendations from external experts, who contribute to the work of the project's Advisory Panel. During the Workshop held in September 2015 in the city of Bled, Slovenia, several ideas were collected as to the desired functions and immediate impact of the Observatory.

The experts felt that the immediate support to "small-scale" cooperation, exchanges, networking and scholarships would likely generate substantial interest, whilst it would not result in resistance from any of the global key players. Better harmonisation of national programmes, added value generated from international cooperation, providing information on EU raw materials research for a better uptake of results, and an opportunity to bring together existing research groups by merging resources (and avoiding duplication of research efforts) were some of the key benefits emphasized. The Observatory could play an important role in the standardisation of best practices such as reporting codes in H&S, mineral reserve/resource classification, harmonisation of terminology relevant for raw materials research, etc.

In terms of potential negative feedback loops experts warned of the danger of adding to bureaucracy instead of making things easier and the possible birth of a new type of resource colonialism, with the EU taking the lead, that can expect resistance from other global players. Others warned that maintaining funding for the Observatory could be difficult and if it comes from EU sources then the EU may be aiming to enforce its own agendas that can lead to a loss of flexibility, which the decisive majority of the experts considered an added value at the first place. The need to develop new, innovative funding models was emphasized on several occasions together with the need to continuously develop the Observatory's services in order to avoid the creation of yet another passive "repository-type" web platform.

In the longer timeframe the Observatory was seen as an important catalyst in aggregating international research, resulting in a better availability of data, including data for basic/academic research. Elimination of overlapping research activities on a global level was mentioned as a long-term benefit that could result in improved technological solutions and the speeding up of the deployment of sustainable raw materials technologies. Industry participation in the work of the Observatory was seen beneficial as it could lead to investment in applied research and raw materials exploration, exploitation and recycling.

In terms of project vision it has been agreed that the Observatory will not only continuously monitor cooperation possibilities but will also actively promote these via facilitating the establishment of dedicated bilateral and multilateral funding schemes and incentives for raw materials cooperation between EU and technologically advanced countries outside the EU. In addition the Observatory could act as a flexible advisory service, providing an overview of global market opportunities for trade and well-documented investment and/or cooperation opportunities for each country within the EU.

CONCLUSIONS

The global challenges being faced by the non-energy minerals industry such as skill shortages, price volatility, market distortions and supply risks, lack of social licence to operate, and others, need to be approached by means of international cooperation, and not only via competition mechanisms. The historical analysis of the five reference countries has shown that bilateral trade partnerships can be a long-term source of mutual benefits for countries or regions, allowing stable economic growth and a politically and institutional stable environment attractive for investments. The key drivers of mining success have shown that countries face similar challenges but are resolved in different ways and can learn from each other. Results have shown that cooperation should not only be among governments, but also between governments and the industry. The close relationship of the government, its agencies and the industry in Japan is a good example. Another example is given by the constant support of the Canadian and Australian governments to the exploration sector by financing the public availability of digital data on exploration or by assigning case managers to projects in order to ensure the smooth approval of necessary permits.

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HAULAGE OPERATIONAL VARIABILITY IMPACT ON MINE PLANNING EXECUTION

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HAULAGE OPERATIONAL VARIABILITY IMPACT ON MINE PLANNING EXECUTION

ABSTRACT

One of the main challenges for engineering in general is always seeking the best results at the lowest cost. Mining engineering is no different. The mining engineer in mine planning area should realize their projects taking into account various operational variabilities of the mine in order to get the best representation of reality. The mine operation is performed with average productivity, for example, medium-distance transport, average speed of transport, loading time, etc However, the mining operation is subject to productivity changes due to the availability of equipment by stops (for maintenance or not).

KEYWORDS

Fleet Selection, Modeling, Mining planning

INTRODUCTION

GOALS

The goal of this study is to model and evaluate a mining operation, aimed at finding the best scenario that makes their operations feasible through the design and management of equipment fleet. The specific objective of this work is to scale the truck fleet to ensure that the premises of the project, as total material handling and feeding the plant, be constant. However, this result should be achieved in the best possible way, that is, causing the number of trucks not suffer fluctuation during the life of the mine, which would decrease the cost of operations.

Tool

The Deswik is a planning tool and integrated mine sequencing that allows the achievement of various scenarios, aiming at maximizing the net present value and internal rate of return of the project or any other desired strategy.

The software uses techniques of linear programming and dynamic programming which allow solving more complex mining problems such as mine sequencing with cut off variable involving the formation and recovery of stocks piles and assessing the mine equipment resources, to achieve the required quality and production targets.

Deswik.Landform & Haulage(LHS) has flexibility and accuracy to deliver the haulage solutions requireds. Covering all variables in the material movement equation. Deswik.LHS includes haul road analysis, detailed truck modeling, fixed and mobile conveying and cost modeling. Offering numerous haulage strategies from minimizing dumping height to reducing haulage distance, the easy-to-follow wizard generates multiple scenarios with ease.

A comprehensive reporting suite reveals the crucial data behind your material movement schedule including detailed haulage paths, cycle time analysis and stage plans. Environmental reporting includes disturbance and rehabilitation forecasting, wet weather simulation and final landform analysis.

Deswik.LHS is equally applicable in open pit and underground environments incorporating mining schedules at any planning resolution.

4- Development, Results and Discussion

This work is based in a hypothetical drill hole data and then create a fictitious deposit of copper. The block model is a simplified representation of reality, was prepared using the following dimensions, 25 m as X, 25 m as Y, and 10 m as Z. These dimensions are in correspondence to the grid and mineralization characteristics. The stages of the work were as follows: Definition of Push Backs; Definition of mining strategy; Sequencing of mining and finally Design and study of the transport fleet.

4.1- Definition of Pushbacks

The final pit used to work was defined using some economic parameters listed in following table.

Table 01 - Economic Parameters

Economic Parameters		
Parameters	Unit	Value
Metal Price	\$/lb	x
SRS	\$/lb	0.24
Process costs	\$/lb	1.68
Mine Cost	\$/lb	0.5
G&A	\$/lb	0.24
Plant Rehab	%	88.6

Once defined the optimal cava, corresponding to the price of 3.0 \$/lb Cu, are defined in long-term sequencing stages of the pit development, or Push Backs, intent, during the whole process of extraction, maintain regularity of the mined material.

With these parameters, the mine cut-off grade was estimated, using the following equation:

$$cutoffgradeCu = \frac{\left[\left(\left(minecosts(\frac{\$}{st}\right) + \left(processcosts(\frac{\$}{st}\right) + \left(G\&A(\frac{\$}{st}\right)\right)x1000\right]}{\left[\left(Price\left(\frac{\$}{lb}\right) - SRS\left(\frac{\$}{lb}\right)\right)\ x\ 2000\ x\ recovery(\%)\right]}$$

Two Push Backs cases were chosen, one corresponding to the price of \$ 2.1/lb Cu and another for a price of \$ 1.5/lb Cu.

Table 02 shows the contents of each stage based on the metal prices.

Table 02 – Grade on each stage according to price metal

Stages	Metal Price ($/lb)	Cut off Grade
1	1.5	0.11
2	2.1	0.07
3	3	0.05

The optimum pits calculated were operationalized, under the geometric and operational parameters, which are showed in the table 03

Table 03 – Pit Geometric Parameters

Parameters	Unit	Value
Bench	m	10
Berm	m	6.4
Face angle	o	70
Ramp Slope	%	10
Ramp Width	m	20

The sequence of images below are the pits generated at each stage.

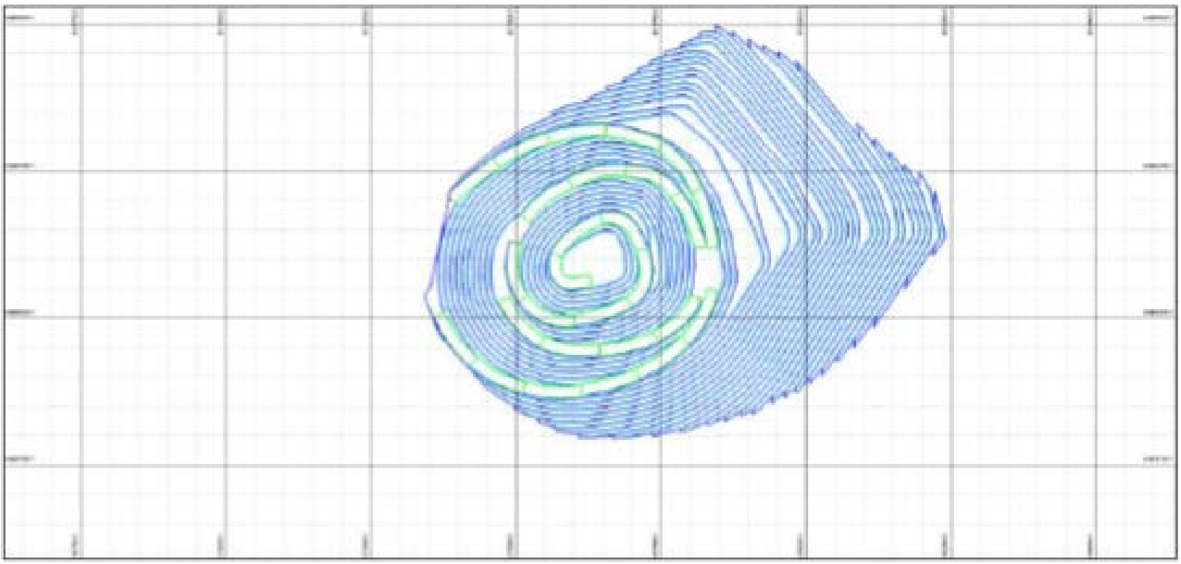


Figure 01 - stage 1 – 1,5 $/lb Cu.

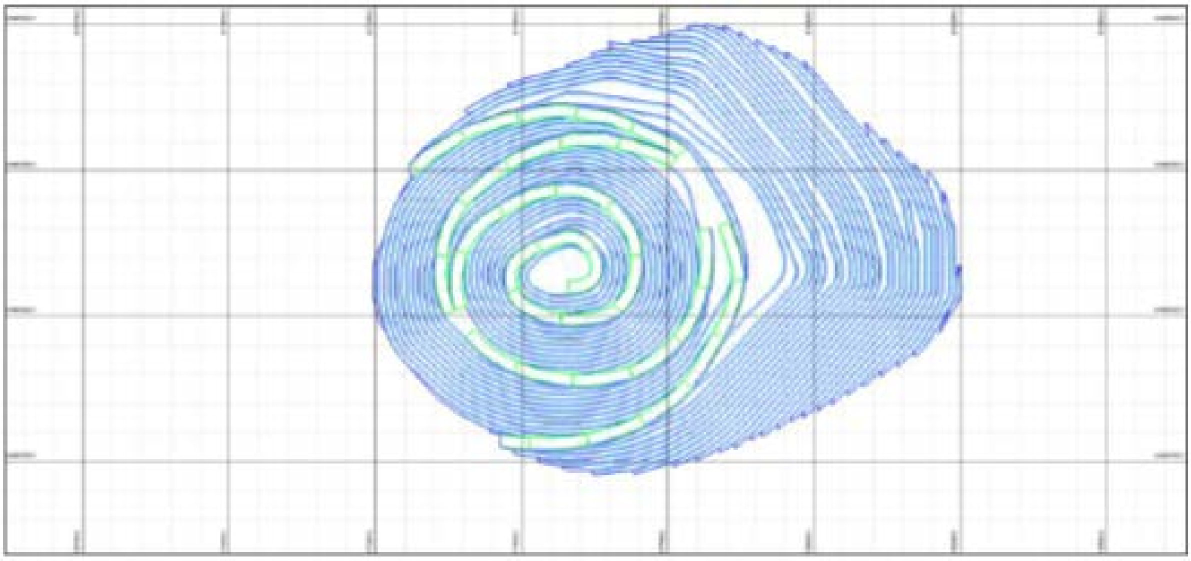


Figure 02 - stage 2 – 2,1 $/lb Cu.

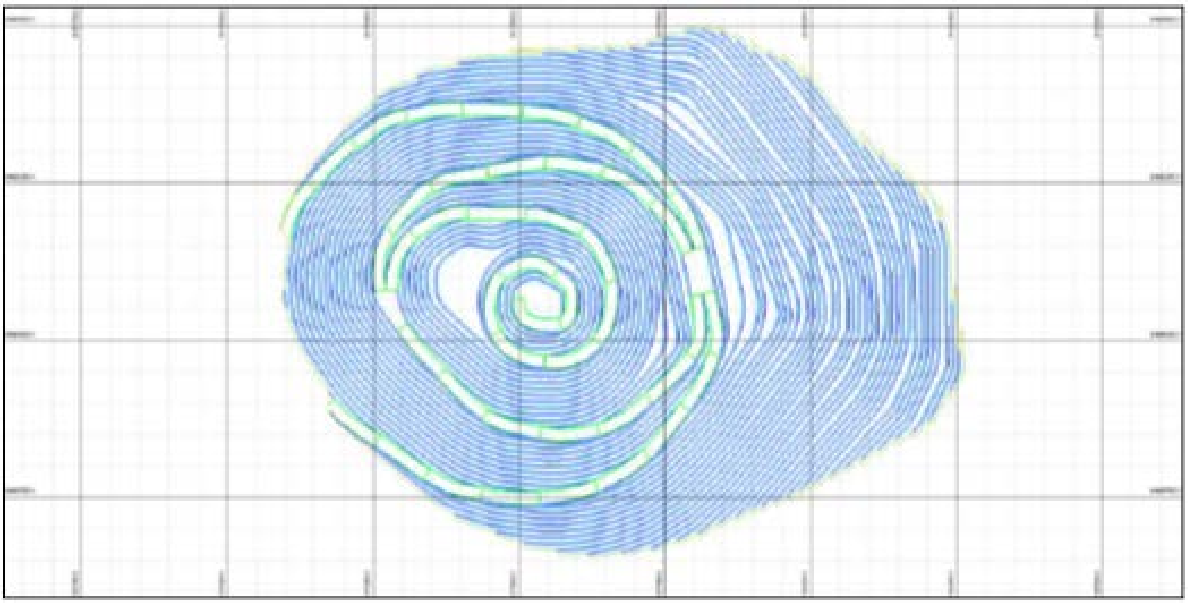


Figure 03 - stage 3 – 3,0 $/lb Cu.

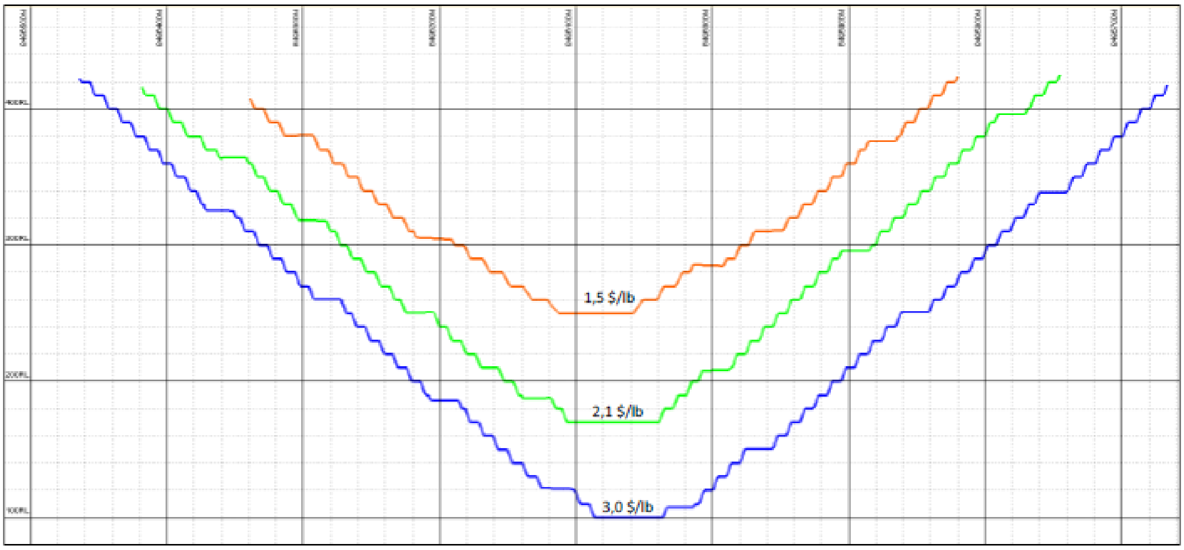


Figure 05 – Vertical Cross Section of the 3 generated pits

4.2- Mining Strategy Definition

The mining sequence was shows the gradually evolution of ore handling, which feeds the plant, wast, which is the waste dump, and also the total material handling, ore and waste. For practical effects of studies were considered only the first 4 years of mine operation, not your entire life.

In this way, it was estimated an annual plant feeding, with approximately 15 M tons and an overall movement of approximately 21 M tonnes, taking into account operating constraints. Assuming a Ramp-Up (operation beginning with successive increase in power plant until it reaches its nominal capacity) 2 years, with 40% in the first and 80% in the second.

4.3- Open Pit Sequencing

The mining sequence in Deswik was performed using the Scheduler module.

The steps taken were:

1) Definition of mine operation schedule, which was defined the mine days and hours of operation.

2) Definition of uses and physical availability of equipment used at the mine operations in order to achieve a more believable result with reality.

3) Definition of mining equipment, such as bulldozers, as production rate and periods of use in order to obtain the premises of the project, such as ore production and full material handling, established in mining strategy.

4) Definition of some necessary inputs, such determinations more detailed production targets, priorities in mining operations, among others.

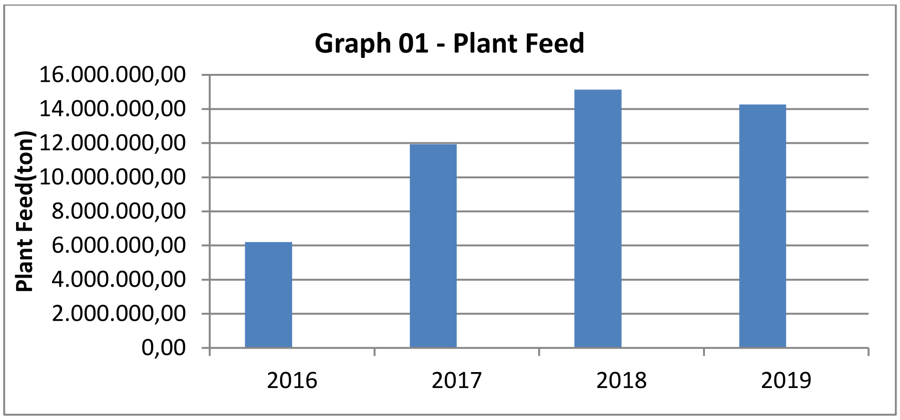
In order to achieve the project goals were made several simulations by varying some parameters and yielding different scenarios.

The result is shown in the table below.

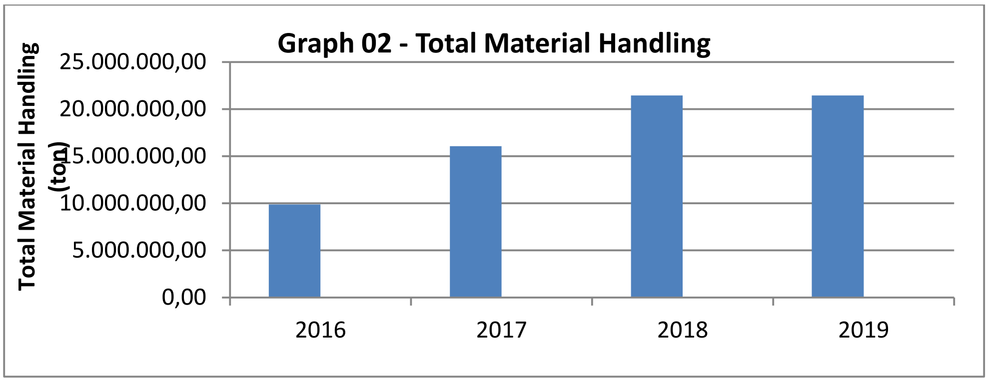
Table 04 – Sequencing Results

PLANT FEED	2016	2017	2018	2019
(T)	6.182.254,2	12.161.156,6	15.872.236,8	14.612.254,8
WASTE DUMP				
(T)	3.807.829,5	3.922.210,0	5.572.257,9	6.832.236,1
TOTAL HANDLING				
(T)	9.890.084	16.083.367	21.444.495	21.444.491
	EQUIPEMENT UTILIZATION			
BULLDOZER 01	1,0	1,0	1,0	1,0
BULLDOZER 02	0,8	1,0	1,0	1,1)
BULLDOZER 03	0,0	1,0	1,0	1,0
BULLDOZER 04	0,0	0,0	1,0	1,0

The graph 01 shows the evolution of ore handling feeding the plant. From the third year note to ore handling regularity in approximately 15 M tonnes, where the plant comes into nominal production system.



Graph 02 shows the evolution of total material handling. From the third year note the regularity of the total movement in approximately 21 M tonnes, which contributes to a more regular sizing of equipment and labor-work project.



4.4- Design and Study of Transport Fleet

The design of the truck fleet required for the assumptions of the project was carried out using the LHS module.

The steps taken were:

1) Definition of all possible pathways in all stages of the mine to ensure that all material has reached a final site, for the plant feed or the waste dump disposal.

2) Choose the truck used in the transport operations. In this work chosen was the 777G Caterpillar (Figure 06).

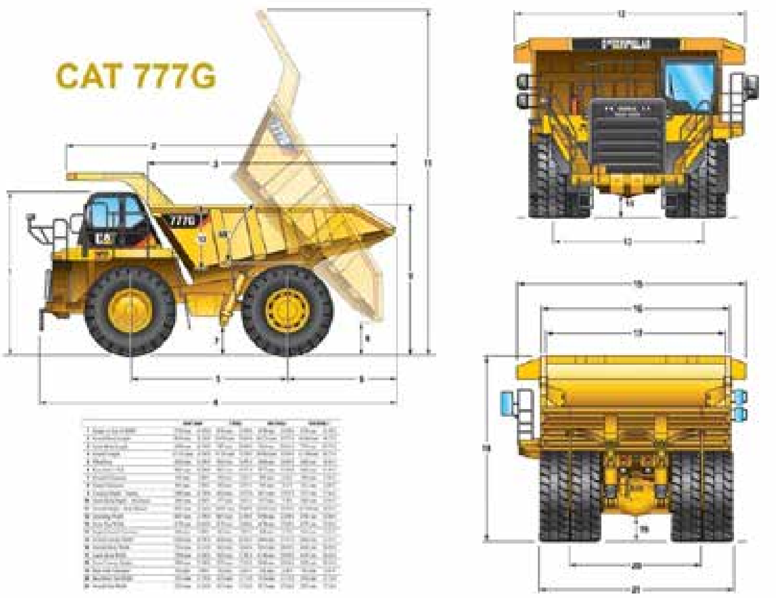


Figure 06 - Caterpillar 777G

3) Generation of the desired scenario and study report containing the data of material handling programming, including detailed transport paths, cycle among other time analysis.

With this report, it calculated the number of hours required for any material to be transported by trucks during the 12 months of the first four years of mine operation. The tables below summarize this procedure.

Table 05 – 2016

MONTH	YEAR	TH	HOURS/MONTH	# TRUCKS	#TRUCKS	AVERAGE
1	2016	4.155,24	474,32	8,76	9,00	10,00
2	2016	4.429,82	443,72	9,98	10,00	
3	2016	4.776,53	474,32	10,07	11,00	
4	2016	4.928,77	459,02	10,74	11,00	
5	2016	4.953,20	474,32	10,44	11,00	
6	2016	4.582,34	459,02	9,98	10,00	

7	2016	5.397,35	474,32	11,38	12,00	
8	2016	5.322,65	474,32	11,22	12,00	
9	2016	4.923,23	459,02	10,73	11,00	
10	2016	4.896,45	474,32	10,32	11,00	
11	2016	2.543,94	459,02	5,54	6,00	
12	2016	2.689,47	474,32	5,67	6,00	
TOTAL		53.598,99	-	-	-	

Table 06 – 2017

MONTH	YEAR	TH	HOURS/MONTH	# TRUCKS	#TRUCKS	AVERAGE
1	2017	8.350,86	475,62	17,56	18,00	
2	2017	6.415,91	429,59	14,93	15,00	
3	2017	7.559,54	475,62	15,89	16,00	
4	2017	7.333,00	460,27	15,93	16,00	
5	2017	7.174,47	475,62	15,08	16,00	
6	2017	7.144,10	460,27	15,52	16,00	
7	2017	7.566,92	475,62	15,91	16,00	15
8	2017	7.416,70	475,62	15,59	16,00	
9	2017	6.678,99	460,27	14,51	15,00	
10	2017	7.191,85	475,62	15,12	16,00	
11	2017	6.847,17	460,27	14,88	15,00	
12	2017	6.606,39	475,62	13,89	14,00	
TOTAL		86.285,90	-	-	-	

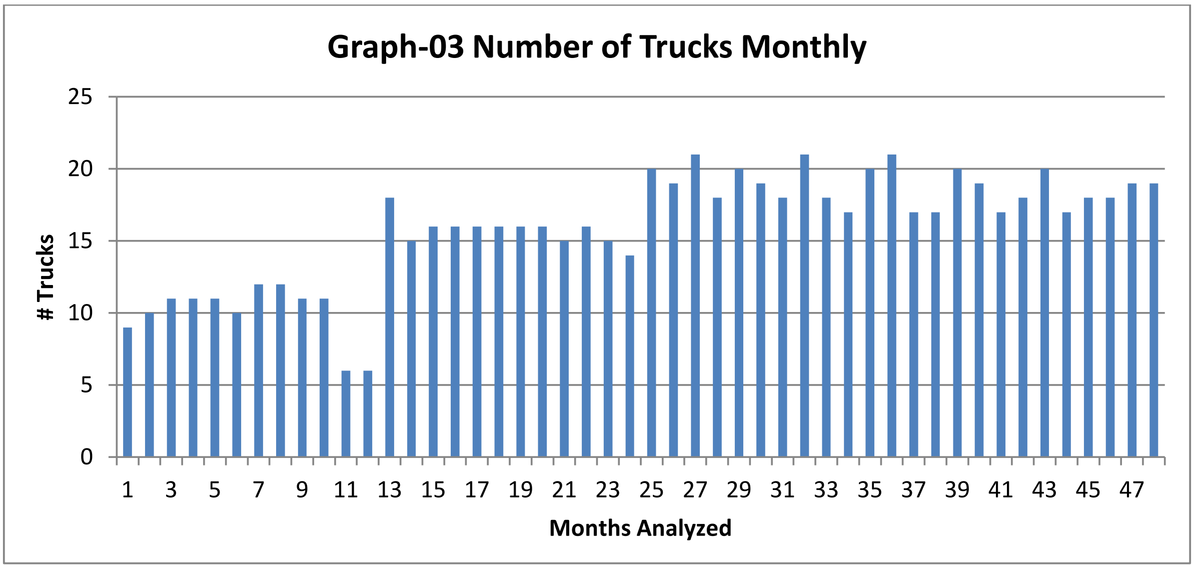
Table 07 – 2018

MONTH	YEAR	TH	HOURS/MONTH	# TRUCKS	#TRUCKS	AVERAGE
1	2018	9.398,09	475,62	19,76	20,00	
2	2018	7.935,36	429,59	18,47	19,00	
3	2018	9.660,20	475,62	20,31	21,00	
4	2018	7.878,77	460,27	17,12	18,00	
5	2018	9.495,07	475,62	19,96	20,00	
6	2018	8.507,38	460,27	18,48	19,00	
7	2018	8.382,35	475,62	17,62	18,00	18
8	2018	9.860,03	475,62	20,73	21,00	
9	2018	8.105,83	460,27	17,61	18,00	
10	2018	7.842,42	475,62	16,49	17,00	
11	2018	8.930,42	460,27	19,40	20,00	
12	2018	9.536,86	475,62	20,05	21,00	
TOTAL		105.532,78	-	-	-	

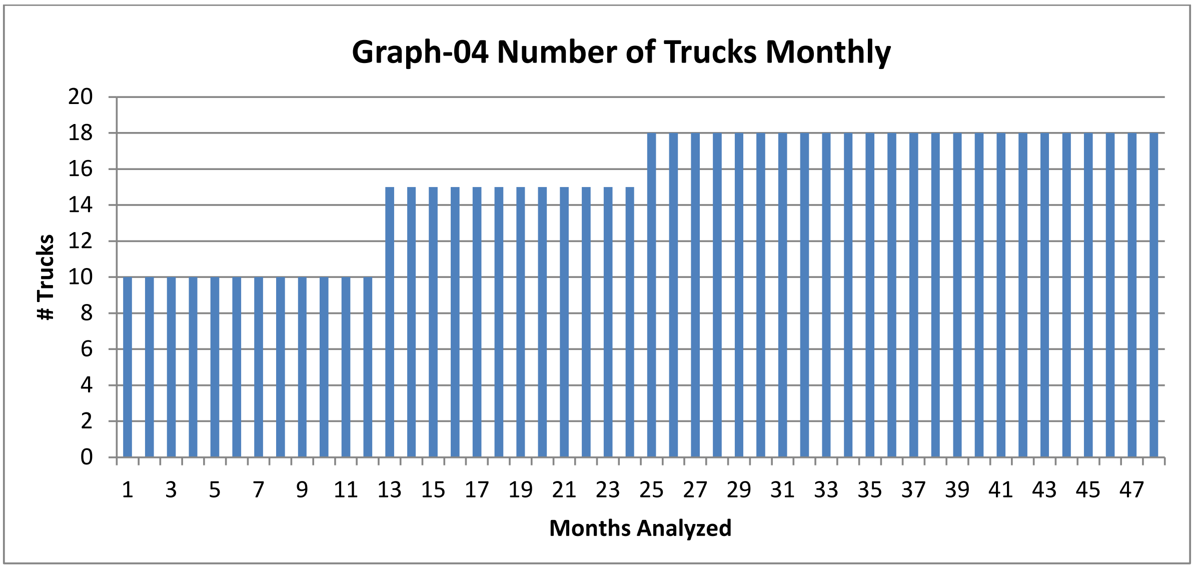
Table 08 – 2019

MONTH	YEAR	TH	HOURS/MONTH	# TRUCKS	#TRUCKS	AVERAGE
1	2019	7.964,80	475,62	16,75	17,00	
2	2019	7.148,24	429,59	16,64	17,00	
3	2019	9.306,63	475,62	19,57	20,00	
4	2019	8.374,25	460,27	18,19	19,00	
5	2019	7.948,40	475,62	16,71	17,00	
6	2019	8.255,24	460,27	17,94	18,00	
7	2019	9.478,38	475,62	19,93	20,00	18
8	2019	7.948,43	475,62	16,71	17,00	
9	2019	7.992,57	460,27	17,36	18,00	
10	2019	8.250,57	475,62	17,35	18,00	
11	2019	8.488,81	460,27	18,44	19,00	
12	2019	8.966,09	475,62	18,85	19,00	
TOTAL		100.122,41	-	-	-	

From these tables it was built a graph showing the minimum number of trucks needed to meet the project assumptions.



The graph above shows that during the period of study, 2016 to 2019 the minimum number of trucks ranged widely, from a minimum of 6 trucks, up to a maximum of 21. This result is not operational because of this fluctuation in the number of trucks during the life of the mine, what is not feasible. So we set up a fixed number of trucks each year, gradually increasing up according to project needs. On 2016, it was used only 10 trucks, and in later years, 15, 18 and 18 respectively (Graph 04).

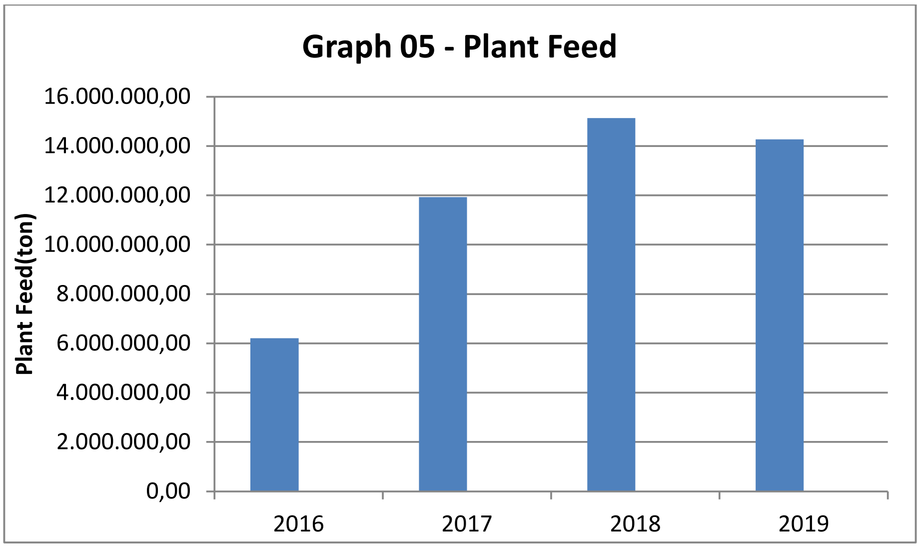


A new scenario was then generated. The result is shown in the table below.

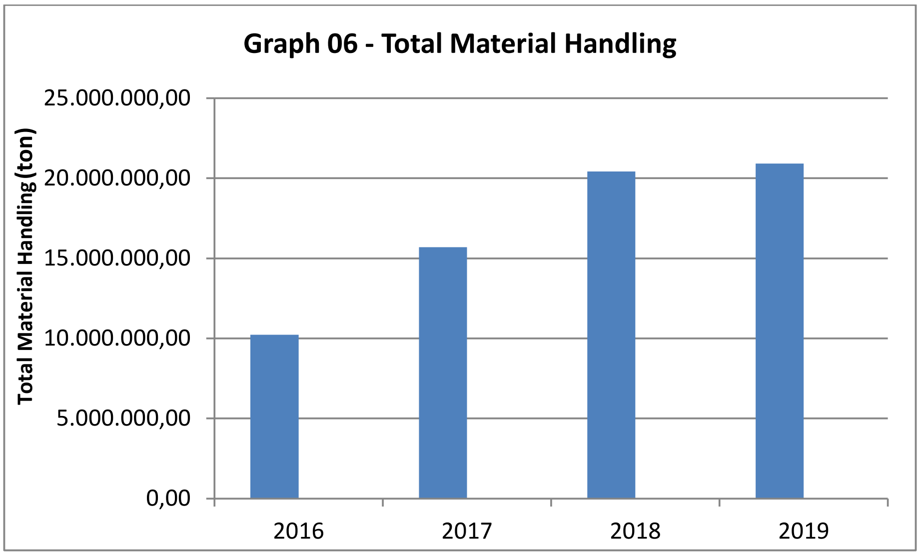
Table 09 - New Sequencing Result

PLANT FEED	2016	2017	2018	2019
(T)	6.206.134,7	11.923.986,5	15.138.461,1	14.264.700,4
WASTE DUMP				
(T)	4.022.208,2	3.765.352,1	5.279.396,8	6.662.833,0
TOTAL HANDLING				
(T)	10.228.343	15.689.339	20.417.858	20.927.533
	EQUIPEMENT UTILIZATION			
BULLDOZER 01	0,9	1,0	0,9	1,0
BULLDOZER 02	0,0	1,0	1,0	1,0
BULLDOZER 03	0,0	0,0	0,9	1,0
BULLDOZER 04	0,0	0,0	0,9	1,0

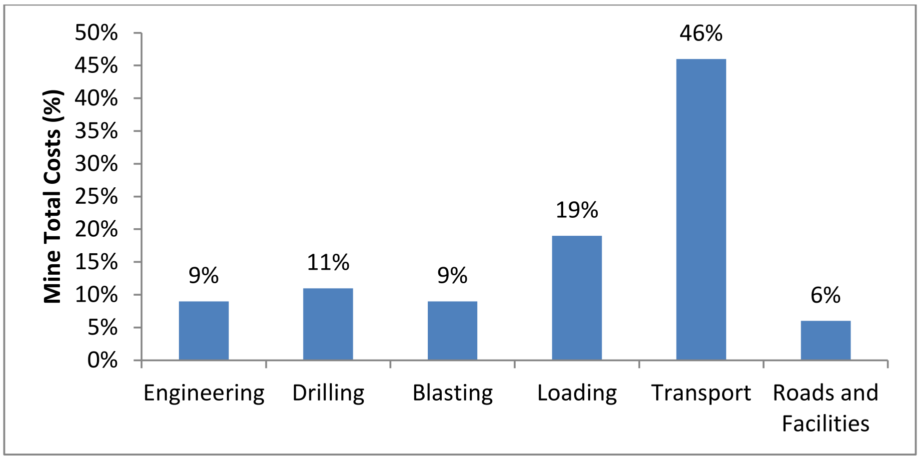
The graph 05 shows the evolution of ore handling feeding the plant. From the third year is possible to note the ore handling regularity of approximately 15 M tonnes, where the plant comes into nominal production system. The result was very similar to the first case, where the number of trucks was variable.



Graph 06 shows the evolution of total material handling. From the third year note the regularity of the total movement in approximately 21 M tonnes, the new scenario with the fixed number of trucks is also similar to the first case.



With this new scenario, it can be seen that there was a slight delay in the mine production, since the process plant and overall handling material were lower. However, the reduction of the costs in the purchase of trucks and also costs related to these costs in the mine life are very significant, as the percentage represented by the transport portion of the total cost of the mine operation is very high (average of 46%). This fact is clear observing the graph below which shows the relative percentage of costs in a mine.



Gráph 07 – Total cost Percentage per stage
Source: (BOZORGEBRAHIMI; et al, 2003)

CONCLUSION

The detailed case study in this work showed the impact of operational variability of transportation equipment during the mining plan. It follows then, that the exact design of the truck fleet in a mining project, although it is the one that best meets the premises of the project, it is often not the most feasible. This study is important in any mining project, as I said before, because of the huge costs related the purchase of trucks and also costs related to the maintenance of the equipment, manpower required for its implementation, large amount of fuel consumed , among others.

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INCORPORATING MINERALOGICAL AND DENSITY PARAMETERS IN FERROUS RESOURCE EVALUATION USING MINERALOGICAL NORM CALCULATION – MNC

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INCORPORATING MINERALOGICAL AND DENSITY PARAMETERS IN FERROUS RESOURCE EVALUATION USING MINERALOGICAL NORM CALCULATION – MNC

ABSTRACT

In order to improve the mass estimation of iron-ore resources and become more precise in the prediction of metallurgical recovery of iron in treatment plants, a new methodology to evaluate density and mineralogy of iron-ore was developed by the team of ferrous resource evaluation of Vale SA. This approach is based on Mineralogical Norm Calculation (MNC) that uses chemical analysis to indirectly calculate the absolute density (ρ_0) which is compared to in situ or natural density (ρ_n) of banded iron formation (BIF), measured by direct density tests. The MNC consists in two steps: 1- definition of a main mineral assemblage of BIF, which in this situation is composed by hematite, goethite, magnetite, kaolinite, quartz, pyrolusite, talc, wavellite, apatite and gibbsite, and 2- calculation of the proportion of minerals for each sample considering the chemical oxides results (Fe_2O_3, FeO, Al_2O_3, SiO_2, P_2O_5, MgO and MnO) and lost on ignition (LOI) which are transformed to mineral proportions. The absolute density (ρ_0) is obtained from the mineral proportions and from their respective specific gravity. Analysis of linear regressions among absolute density and in situ density indicated good correlation between them. The porosity (ϕ) of the samples is calculated by the relation of absolute and in situ density. The MNC and regression equations were applied in two ferrous mines. The results were reconciled with production data, between the years 2011-2014. The results showed more precision in mass estimation for both deposits, considering the mass error in terms of percentage of production. At the considered period, there was a better reconciliation among production data and mass estimation, comparing models with traditional density tests against CNM. The differences of reconciliation were reduced from -13.4% to -4.8% for deposit A, and from -6.5% to -2.8% for deposit B. The use of MNC will help to better define both geological and mineralogical domains.

INTRODUCTION

Due to high costs, the iron mining industry generally considers a limited number of samples to evaluate the mass of the deposit and their mineralogical variability which is necessary to define the mineral processing parameters. Mindful of continuous improvement of resource evaluation process and considering the intrinsic characteristics of iron-ore, the Vale Ferrous Resources Evaluation team proposed a low cost methodology to minimize the problem of low representativeness of density and mineralogy datasets.

The Brazilian iron deposits occur mainly in three provinces: Quadrilátero Ferrífero, Carajás and Corumbá. These deposits were affected by weathering during the Cenozoic Age and all of them are characterized by simple mineralogical assemblage containing a few numbers of mineral species. The good correlation between chemical variables, transformed to mineral proportion by Mineralogical Normative Calculation (MNC), versus mineral proportion, calculated by quadrant counting using Optical Microscope (OMC), allows the use of chemical data from drilling database. The increase of indirect information about mineralogy and density of rocks improves the estimation of BIF resources' mass.

This study presents in detail the proposed methodology and its application in two iron-ore minesof Quadrilátero Ferrífero, which presented good results in terms of costs reduction with sampling and mineral analysis. Both studies also improved the mass prediction of the resource models, as can be proved with mass reconciliation of both models during four years of explotation.

This new approach helps knowledge increase in terms of local and global mineralogical and density variability for the deposits and opens new possibilities of new ore processing studies, allowing also the stochastic simulation of mineral proportions.

GEOLOGY AND MINERAL ASSEMBLAGE OF BANDED IRON FORMATION

The most exploited Banded Iron Formation (BIF) in Brazil is located in Minas Gerais state, region known as Quadrilátero Ferrífero (QF, figure 1) which comprises a variety of metamorphic paleoproterozoic rocks (Minas Supergroup) which BIF types are found in Cauê Formation from Itabira Group. The enrichment in iron is a product of combination of supergene and structural events.

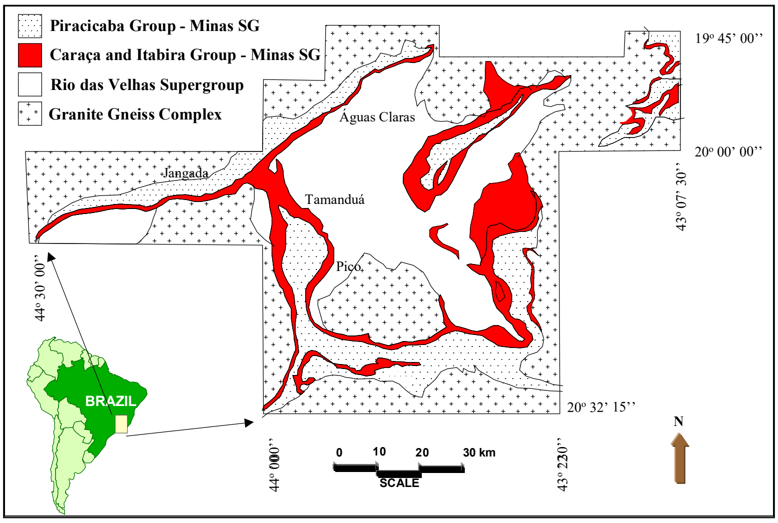

Figure 1 - Quadrilátero Ferrífero Province with Minas SG outcrops in highlight (Dorr, 1964)

The BIF from the QF is called itabirite and its mineralogical composition is restricted in terms of number of species: magnetite, hematite and goethite (ferrous minerals), and quartz, carbonates, kaolinite, gibbsite, pyrolusite, talc, wavellite, apatite and amphiboles (gangue materials). Weathering is responsible for the iron rich ranges surrounded by areas of subtle relief dominated by phyllites, dolomites and schists (Ribeiro and Carvalho, 2000). Weathering in itabirites has been reported as deep as 500 m, still an average of about 300 m is more commonly found. Important soft and enriched ore are located in weathering zones. Leaching of carbonate, silica and silicate minerals from the itabirite as well as the structural control may account for the origin of soft iron-ores by a dissolution process (Viel et al., 1987; Amorim, 1992). The dissolution of carbonate and siliceous bands starts from the fractures and pursues along the band contacts decreasing the cohesion between grains and bands (Ribeiro and Carvalho, 2000 – Figure 2).

The microscopic images from figure 3 show three main ferrous and gangue crystals present in BIF from QF (Gonçalves, 2015). Magnetite crystal frequently alters to hematite in the borders of iron bands, fractures and cleavage plans of crystals (martitization). Their occurrence is mainly on the unweathered deep zones of the deposits. Hematite has two different textural groups: martite (alteration from magnetite) and compact hematite (hematite recrystallization over stress) and it is predominant in every high and intermediate iron grade zones of most economic deposits. Goethite crystals occur on the upper zones and are associated to weathering which leaches the iron 2+ from magnetite, amphiboles and some iron rich carbonates forming iron hydroxides in alternating porous bands. Strong variation of porosity has been observed in the weathered BIF.

Figure 2 - Unweathered (left) and weathered (right) BIF contact - siliceous itabirite (Ribeiro and Carvalho, 2000)

Figure 3 - Main minerals from BIF of QF and their chemical composition. (Gonçalves, 2015)

DENSITY EVALUATION OF BIF DERIVED ROCKS

As can be seen in the previous images, the porosity and mineralogy are variable and their influence in the final rock density is significative. Two main methods are applied to measure *in situ* density of BIF: filling volume (FV), for friable materials, and hydrostatic balance (HB), for compact samples (Santos 2006 and Santos 2007). These tests have, in general, different supports: the HB method uses a cylinder sample from diamond drilling cores approximately 20 cm long and 7 cm in diameter, while the FV method uses an excavated sample with approximately 0.027 m^3 of volume. Both methods need homogeneous fields in order to have samples with regular shapes, reducing measurements errors of the volume. The relation between sample mass (M) and its volume (V) is considered as natural or *in situ* density, wet basis (equation 1).

$$\rho_n = \frac{M}{V} \quad (1)$$

Only friable samples, FV method, have moisture measurement and the water presence depends on the porosity of rock matrix and also on mineral content and its texture. Porous (goethite and martite) and argillaceous minerals (kaolinite and gibbsite) have high specific surface and the probability to retain water on the crystal surface is higher than in compact crystals (quartz and recrystallized hematite). The density in dry basis (ρ_d) is given by the relation between dried mass (M_d) and volume (equation 2). The moisture (u) is given by the proportion of water compared to the total mass, M (equation 3).

$$\rho_d = \frac{M_d}{V} \quad (2) \qquad u = \frac{(M - M_d)}{M} \quad (3)$$

Most part of weathered iron deposits have no regularity in terms of compactness as can be seen in figure 2. Drilling samples of weathered ore are fractured and their friability favors disaggregation of the rock, which reduces the cohesion between particles of samples, changing the original shape of the sampled material.
The presence of alternation of compact and friable materials makes impossible the use of both tests, BH and FV, to evaluate the density of this kind of material which represents most part of weathered deposits. This work proposes the use of screening tests to which all drilling samples are submitted, to separate coarse and fine materials. The oversize of 6mm or 8mm screen mesh is considered as grains of compact BIF or proportions of coarse materials (G_c), and finer grains, undersize, as friable BIF or proportions of fine materials (G_f).

MINERALOGICAL NORM CALCULATION

Normative calculation of mineral mass proportion was originally applied to igneous rocks, considering the chemical differentiation of magma. Similarly to igneous rocks, but considering the sequence of weathering attack, Voicu (1997) proposes a normative method to evaluate the mineral assemblage contained in a laterite soil profile from sub-tropical region in Africa. Ribeiro, Pires and Carvalho (2002) adapted the Voicu´s algorithm to be applied on the definition of BIF mineral assemblage (MNC). Motta (2014) demonstrated that there are good correlations among mineral proportions calculated by MNC and by Optical Microscopic Counting (OMC) methods. The OMC method uses polished sections of very small samples, reduced after homogenization and grinding process which breaks 95% of mass particles under 0.15 mm size. The proportions of minerals are obtained by counting of particles inside a different standard viewing field of the microscope. The OMC also uses the chemistry of minerals (Figure 3) and theirs specific gravities to calculate the final proportions of the mineral assemblage. A simplified version of MNC can be summarized as:

i. Magnetite content from FeO grade – residual Fe_i;
ii. Gibbsite content from LOI and Al2O3 grades – residuals LOI_i and Al2O3_i;
iii. Kaolinite content from LOI_i, Al2O3_i and SiO2 grades – residuals SiO2_i and LOI_ii;
iv. Quartz content from SiO2_i;
v. Pyrolusite content from Mn grade;
vi. Goethite content from Fe_i and LOI_ii – residual Fe_ii;
vii. Hematite content from Fe_ii residual grade.

The complete MNC algorithm uses try and error functions to distribute chemical components among minerals in order to honor the final addition of oxides and minerals, which sum should be around 100%.

ESTIMATING DENSITY AND POROSITY OF BIF ROCKS IN GEOLOGICAL MODEL

There are not many published papers about BIF porosity. Ribeiro (2004), studying leaching of itabirites and hematite samples, calculated porosity varying from 30% to 40% for friable ores from Pico Mine (QF). Lima et al. (2013) measured the porosity of friable hematite ore, canga and jaspelites samples from Carajás Province, using helium gas porosimeter. Ramanaidou (2009) published a paper where he calculated indirectly, by normative technique, the porosity of friable hematite and itabirite ore and compact itabirite samples from Capanema Mine (QF).

MNC method combined with screening tests allows obtain indirectly the dry density, porosity, and saturation for all geological block models. The conversion of chemical grades into minerals proportions (M_m) yields the calculation of the absolute density (ρ_0) of BIF which formulae is given by the relation among the sum of mass divided by the sum of volume of each mineral (equation 4)

$$\rho_0 = \frac{\sum_{i=1}^{n} M_{mi}}{\sum_{i=1}^{n} \frac{M_{mi}}{\rho_{mi}}} \qquad (4)$$

Where i is the index of mineral that varies from 1 to n, M_{mi} is the mass proportion of each mineral, and ρ_{mi} represents its specific gravity. The relation between M_{mi} and ρ_{mi} is the volume of each mineral. The total porosity of sample (ϕ) can be estimated indirectly by the relation among dry density (ρ_d) and absolute density (ρ_0) according to equation 5.

$$\Phi = 1 - \frac{\rho_d}{\rho_0} \qquad (5)$$

The saturation of porous (ϕ_{sat}) is given by the proportion of free water inside the rock matrix in percentage of total porous (equation 6).

$$\Phi_{sat} = \frac{(\rho_n - \rho_d)}{\Phi} \qquad (6)$$

In the equation above ρ_n is natural density in wet basis. These parameters are estimated separately for each grain size partition of the rock, coarse and fine, obtained from the screening tests (G_c and G_f). The final density and porosity for the total rock is an average of these two grain size partitions. All BIF derived rocks of the geological model have density and porosity estimated by MNC method, except fresh carbonate and amphibolite BIF. Two case studies from QF were chosen to demonstrate this methodology.

CASE STUDY A

The deposit A is a typical BIF from the western portion of QF. The database is composed of 126 samples, being 47 of friable hematite and itabirite ores, 66 of compact hematite and itabirite ores and 13 of compact cangas (superficial iron rich duricrust).

The results in terms of mineralogical composition (MNC), dry, natural and absolute densities, moisture, and porosity are shown in the tables 1 and 2.

Hematite predominates in most of the rocks, mainly in hematite types, whereas the presence of quartz increases in itabirites types and the occurrence of goethite, kaolinite and gibbsite are more common in cangas and contaminated rock types. In terms of porosity and saturation, the friable material has high porosity and moisture. The average porosity by lithology, varies from 38% to 42%, for friable materials, and from 7% to 10.2%, for compact materials (except canga).

Table 1 - Average in percentage of MNC from samples of friable itabirite (IF), rich friable itabirite (IFR), friable hematite (HF), contaminated friable hematite (HMC), compact itabirite (ICS), compact hematite (HC), contaminated compact hematite (HCC), mixed itabirite (IMS) and canga (CG) from deposit A – QF.

Nb.Sam	*Rock*	*Kao*	*Gib*	*Qtz*	*Goe*	*Hem*	*Other*	*Sum*
9	IF	1.34	0.00	32.04	6.91	59.45	0.140	99.9
10	IFR	2.66	0.00	14.75	4.95	77.68	0.166	100.2
18	HF	0.95	0.38	0.96	3.72	93.82	0.175	100.0
10	HMC	1.29	3.25	0.45	11.25	83.41	0.289	99.9
13	ICS	0.38	0.00	47.05	1.82	50.16	0.176	99.6
18	HC	0.42	0.03	0.05	4.23	94.92	0.255	99.9
16	HCC	0.44	0.00	0.22	3.17	96.07	0.170	100.1
19	IMS	0.43	0.00	45.23	1.01	53.39	0.165	100.2
13	CG	1.61	4.21	0.00	49.40	44.21	0.436	99.9

Table 2 - Average of natural density, moisture, dry density, absolute density and porosity in percentage from samples of friable itabirite (IF), rich friable itabirite (IFR), friable hematite (HF), contaminated friable hematite (HMC), compact itabirite (ICS), compact hematite (HC), contaminated compact hematite HCC), mixed itabirite (IMS) and canga (CG) from deposit A – QF.

Nb Sam	*Rock*	$\rho_n\,(g/cm^3)$	*u(%)*	$\rho_d\,(g/cm^3)$	$\rho_0\,(g/cm^3)$	$\varphi(\%)$
9	IF	2.47	2.92	2.40	3.92	38.8%
10	IFR	2.71	2.22	2.65	4.45	40.5%
18	HF	3.21	2.75	3.12	5.09	38.8%
10	HMC	2.97	6.13	2.79	4.81	41.9%
13	ICS	3.31	0.01	3.28	3.58	8.5%
18	HC	4.86	0.01	4.81	5.18	7.0%
16	HCC	4.70	0.01	4.65	5.18	10.2%
19	IMS	3.32	0.01	3.29	3.65	10.0%
13	CG	2.85	0.06	2.68	4.45	39.8%

Figure 4 presents the scatter diagrams correlating absolute density versus dry density. The correlation coefficient among the densities is high for compact materials (0.99) due to their low porosity and intermediate for friable materials (0.84) because its porosity is higher than compact materials and presents high variability. The two linear equations obtained by linear regression from scatter diagrams are used as functions to transform chemical and grain size variables, estimated in the block model, in absolute density for two classes of size partition. The coarse fraction above 6 mm or 8mm (Gc) is considered as compact and the dry density are obtained from the equation 6.

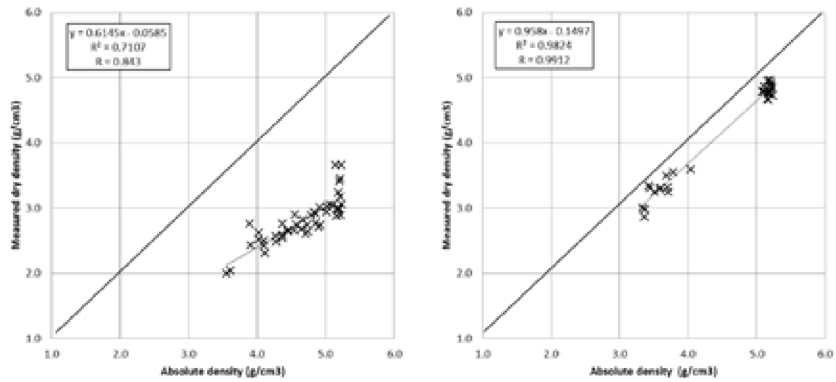

Figure 4 - Scatter diagrams show correlation between absolute density *versus* measured dry density for friable samples, left, and compact samples, right – deposit A.

The fines below 6mm or 8mm (Gf) have their density calculated by the equation 7. The final absolute and dry density of the total block is calculate using the equation 8, combining the results from equations 6 and 7 and the mass proportion Gc and Gf.

$$\rho_{d\_Gc} = \rho_{0\_Gc} \times 0.958 - 0.1497 \quad (6) \qquad \rho_{d\_Gf} = \rho_{0\_Gf} \times 0.6145 - 0.0585 \quad (7)$$

Where ρ_{0_Gc}, ρ_{0_Gf}, ρ_{d_Gc}, and ρ_{d_Gf} are, respectively, absolute and dry densities of coarse (above 6 mm or 8mm) and fine (below 6mm or 8mm) materials.

$$\rho_{d\_tot} = (G_c + G_f)/(\frac{G_c}{\rho_{d\_Gc}} + \frac{G_f}{\rho_{d\_Gf}}) \quad (8)$$

$$\rho_{0\_tot} = (G_c + G_f)/(\frac{G_c}{\rho_{0\_Gc}} + \frac{G_f}{\rho_{0\_Gf}}) \quad (9)$$

Where ρ_{0_tot} and ρ_{d_tot} are, respectively, absolute and dry densities estimated for each block or each sample calculated indirectly from its chemical composition and its grain size partition present on sample database or on block estimation. The block or sample porosity is estimated using these densities and by applying the equation 5.

The table 3 shows the declustered means of these variables calculated for all drilling composites samples database (10 m). The natural density was estimated considering the mean moisture and saturation from channel samples and reverse circulation drilling samples. These samples are more realistic and representative in terms of moisture than the moisture of previous density database. These values represent good estimation for mean of these variables by lithology for all deposit A.

Table 3 - Estimated natural density, moisture, dry density, absolute density, porosity and saturation in percentage from samples of friable itabirite (IF), rich friable itabirite (IFR), friable hematite (HF), compact itabirite (ICS), compact hematite (HC), canga(CG), aluminous itabirite (IAL) and manganesiferous itabirite (IMN) – drilling and channel composite samples database from deposit A – QF.

Samp	*Rock*	ρ_{d_tot} *(g/cm³)*	ρ_{0_tot} *(g/cm³)*	ϕ*(%)*	*u(%)*	ρ_{n_tot} *(g/cm³)*	ϕ_{sat}*(%)*
312	CG	2.6	4.4	40%	12%	3.0	92%
330	HC	3.8	4.9	22%	3%	3.9	48%
1470	HF	3.1	4.8	35%	8%	3.4	76%
92	IAL	2.6	4.2	37%	12%	3.0	97%
418	ICS	2.7	3.7	25%	2%	2.8	25%
1550	IF	2.3	3.7	36%	7%	2.5	47%
564	IFR	2.8	4.4	36%	8%	3.1	71%
112	IMN	2.5	3.9	35%	11%	2.9	86%

The estimation of natural density for the block model of BIF rocks of deposit A is shown on figure 4. Figure 5 presents the original and the new mass estimation based on natural density.

In order to validate the new model, a mass reconciliation of the last four years of production was made.

The majority of the production for this mine in this period, ore or waste, was composed by BIF rocks. The total production of ore (feed of treatment plant) and waste material (sent to dump pile) was considered in the mass reconciliation process. Production data for this period was compared with the estimated exploited data from block models, considering the original and the new natural density parameters. Table 4 shows the results of this reconciliation.

Using the new parameter of MNC estimated natural density the difference between production data and block model, for the considered period, is reduced in absolute value, from -13.4% to -6.5% when compared to traditional natural density values.

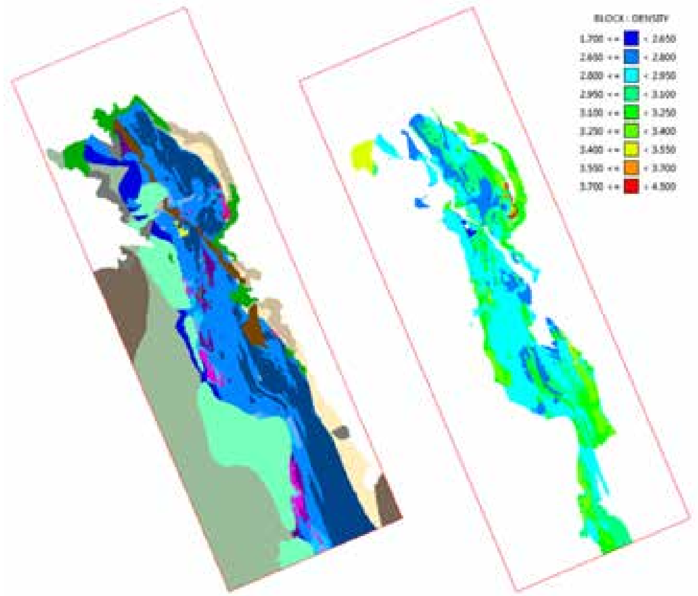

Figure 5 - Block Model of deposit A showing the lithologies (left) and BIF rocks with estimated natural density (right).

Table 4 - Reconciliation among production data (Prod) and estimated mass considering the original natural density (ρ_n) and the CNM estimated natural density (ρ_{n_tot})– deposit A: period of production 2011-2014.

Year	ρ_{n_tot} *(Mt)*	ρ_n *(Mt)*	*Prod(Mt)*
2011	9.7	8.7	10.2
2012	10.9	10.0	11.6
2013	14.0	12.3	16.6
2014	17.5	17.3	17.3
2011-2014	52.1	48.2	55.7
Dif ([mod-prod]/prod) %	-6.5%	-13.4%	0.0%

CASE STUDY B

The deposit B is located in the northern east region of QF, which has higher metamorphic grade than the western portion. Due to high metamorphic grade, the predominant iron mineral is compact hematite with low porosity. Goethite and martite are secondary iron minerals. The database is composed of 884 samples, being 221 of friable hematite and itabirite ores and 663 of compact hematite and itabirite ores (tables 5 and 6).
The scatter diagrams correlating absolute density *versus* dry density for deposit B evidenced high correlation coefficient for compact materials (0.98) and intermediate correlation coefficient for friable materials (0.78). Same methodology applied on deposit A was considered in the calculation of density and porosity parameters for deposit B. Table 7 shows the declustered means of the resulting variables calculated for all 15 m composite samples from drilling database (cell declustering size of 200m x 200m x 15m). Different from deposit A, the natural density was obtained considering fixed 50% of water saturation in calculated porous.

Table 5 Average in percentage of MNC from samples of friable itabirite (IF), manganesiferous hematite(HMN), friable hematite (HF), friable ferruginous quartzite (QTf), compact itabirite (IC), compact hematite (HC), manganesiferous itabirite(IMN), semi-compact itabirite (IS) and compact ferruginous quartizite (QTc) from deposit B – QF.

Nb.Sam	*Rock*	*Kao*	*Qtz*	*Goe*	*Hem*	*Mag*
7	HMN	2.9	0.7	9.8	82.3	1.3
12	HF	2.4	2.1	5.4	88.9	0.5
175	IF	1.1	34.4	2.5	61.3	0.6
6	IMN	5.4	37.8	13.3	41	0.7
21	QTf	7.2	75.0	5.2	11.7	0.6
240	HC	0.8	1.1	1.5	95.7	0.6
411	IC	0.6	43.2	1.5	53.7	0.6
5	IS	0.5	37	1.6	59.7	0.6
7	QTc	3.6	87.2	3.4	5.7	0.4

Table 6 Average of natural density, moisture, dry density, absolute density and porosity in percentage from samples of friable itabirite (IF), manganesiferous hematite(HMN), friable hematite (HF), friable ferruginous quartzite (QTf), compact itabirite (IC), compact hematite (HC), manganesiferous itabirite(IMN), semi compact itabirite (IS) and compact ferruginous quartizite (QTc) from deposit B – QF.

Nb Sam	*Rock*	$\rho_n\,(g/cm^3)$	$u(\%)$	$\rho_d\,(g/cm^3)$	$\rho_0\,(g/cm^3)$	$\phi(\%)$
7	HMN	3.39	7.07	3.15	4.94	36%
12	HF	3.47	6.5	3.25	4.97	35%
175	IF	2.83	2.68	2.75	3.88	29%
6	IMN	2.71	8.68	2.48	3.64	32%
21	QTf	2.17	4.11	2.08	2.89	28%
240	HC	4.84		4.83	5.13	6%
411	IC	3.45		3.45	3.67	6%
5	IS	3.45		3.44	3.86	10%
7	QTc	2.51		2.51	2.77	10%

Considering the total amount of 46109 of samples database and the approximate cost of US$ 50 to evaluate density for each sample, it was incorporated US$ 2.3M as indirect information of density and porosity in the deposit B database. Figure 6 shows the spatial distribution of these samples. Another scenario of cost reduction may be considered: geophysical logging of 113400 meters of BIF drilled samples with approximate cost of US$ 10 per meter. In this case it would save US$ 1.13M.

Table 7 - Estimated natural density, moisture, dry density, absolute density, and porosity in percentage from samples of compact hematite (HC), friable hematite (HF), manganesiferous hematite(HMN), friable manganesiferous hematite(HMNF), pulverulent hematite (HP), compact itabirite (IC), canga (CG), semi-compact itabirite (IS), manganesiferous itabirite(IMN), friable itabirite (IF) and quartizite (QT) from drilling and channel composite samples database from deposit B – QF.

Samp	*Rock*	ρ_{d_tot} (g/cm3)	$\rho 0\_tot$ (g/cm3)	$\phi(\%)$	$u(\%)$	ρ_{n_tot} (g/cm3)
4037	HC	4.55	5.07	10%	1.10%	4.6
6791	HF	3.73	5.03	26%	3.40%	3.86
1458	HMN	3.72	5.01	26%	3.40%	3.85
293	HMNF	3.67	5.03	27%	3.60%	3.8
1547	HP	3.42	5.04	32%	4.50%	3.58
2726	IC	3.48	3.76	7%	1.10%	3.52
718	CG	3.22	4.64	31%	4.50%	3.38
5674	IS	3.13	3.84	18%	2.90%	3.23
1121	IMN	2.97	3.95	25%	4.10%	3.1
19644	IF	2.85	3.95	28%	4.70%	2.99
210	QT	2.37	3.1	24%	4.80%	2.48

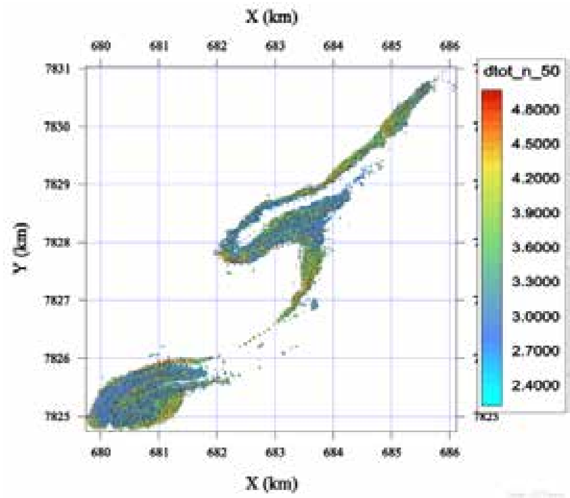

Figure 6 - Spatial distribution of drilling and channel samples with natural density represented by colour scale – deposit B.

The MNC methodology allowed the estimation of the porosity and density for the entire model (Figure 7). The information about local porosity is a very important tool for hydrologic studies and helps the location of wells to pump water in order to lower the underground water level in the mine.

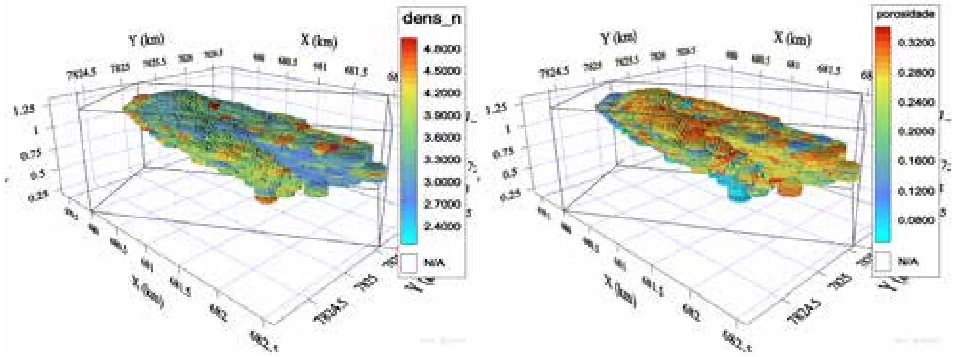

Figure 7 - Block model estimation of natural density (left) and porosity (right) – southwest portion of deposit B.

The mass reconciliation of deposit B for the same period of ore production (2011 to 2014) improved with the use of the new natural density compared to the traditional natural density. The error decreases from -6.5% (traditional natural density) to -2.8% (new MNC natural density) in comparison to actual ore production data.

CONCLUSIONS

The new methodology to estimate indirectly the natural and dry density, combining MNC with screening tests of BIF samples or blocks, improved the factors of mass reconciliation of two important models of iron mines located in QF. In both mines the errors between actual production for the period 2011 to 2014 and estimated mass from the block model were reduced in comparison to the traditional method of natural density evaluation.

In addition to density, this method allows the estimation of porosity and mineral proportions which are important tools to define better ore treatment routes and hydrological studies. Results of the mineral and porosity estimation for deposits A and B reveal their difference in terms of ore geneses: deposit B, with high metamorphic grade, shows, in general, higher density and lower porosity than deposit A. Goethite proportion, associated to weathering, is higher in deposit A than deposit B.

Other appealing aspects about this method are: reestablishment of coherence among estimating of density and porosity *versus* chemical and grain size partition; low cost method allowing estimating of density in all database including destroyed samples; incorporating some millions of dollars as indirect density information; probable increasing of mineral reserves mass for both deposits A and B between 7% and 3.5%.

Important revenue could be reached for the company if the MNC application spreads as a standard evaluation procedure for other iron deposits.

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INFLUENCE OF PULSATION FREQUENCY IN IRON OXIDE FINE PARTICLES JIGGING

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INFLUENCE OF PULSATION FREQUENCY IN IRON OXIDE FINE PARTICLES JIGGING

ABSTRACT

The importance of pigments for the civilization is obvious and well documented. Although these materials have been discovered many years ago, research continues nowadays. Industries often require new shades, colours and more homogeneous and stable pigments. The selection of mineral pigments is of major importance to acquire high quality, colour, purity and mostly free of chemical contaminants, such as chemicals from froth flotation process. The region of Catalão, Brazil, has several minerals species in the site, including apatite, barite, magnetite, monazite, niobium, titanite and vermiculite. Nowadays phosphate, niobium and barite are economically exploited. For the production of these minerals, the magnetite is removed through magnetic separation and sent to a tailings dam. The aim of this study is to evaluate the possibility of producing iron oxide to be used as pigments as well as to evaluate the pulsation frequency influence in the jigging process. A Denver jig, in lab scale was used. Test were carried out using six different particle sizes and four pulsation frequencies, keeping the water flow rate fixed at 20 litres per minute. The results indicate that iron oxide production for pigments is viable from phosphate rock tailings; since it was possible to produce magnetite concentrated with grades over 90% and magnetite recovery around 40%.

KEYWORDS

Iron oxide, Jigging, Phosphate

INTRODUCTION

According to Casqueira and Santos (2008), humankind has used colours for more than 20 thousand years. The first dye to be known to humankind was soot, around 3,000 BC. Pigments are insoluble dyes (small corpuscles) in the medium in which they are dispersed. In the case of soluble dyes, the solutions penetrate the material to dye (especially textile), lending it not only colouring but also reacting with this material. The selection of raw materials is of major importance to acquire high quality mineral pigments, colour luminescence and purity, that is, free of chemical contaminants. The usual particle size for pigment industry is around 50 µm. However, for Barros et al. (2005) pigments can be produce by grinding with particle size up to 100 µm. Mineral pigments are gaining commercial acceptance because of the growing ecological awareness, in order to reduce environmental impacts caused by production methods generally associated with synthetic ones. The most common mineral pigments according to Casqueira and Santos (2008) are iron oxide, manganite, chromite, quartz, feldspar, monazite, zircon, titania and micas (such as muscovite and biotite).

Magnetite (Fe_3O_4) is a magnetic oxide of iron formed by natural ions Fe^{+2} and Fe^{+3}. It is the most common magnetic mineral and is present in small amounts in almost all crustal rocks. Most of the iron ore production in Brazil, by magnetite or hematite processing, is done by froth flotation or magnetic separation. Costa (1982) state that froth flotation inhibit the use of iron oxide for purposes other than steel production due to the amount of chemicals, particularly surfactants, involved in this process. According to market, the metal load (sinter, pellet or granular) for steel production should contain approximately 65% of iron in its composition regardless of the mineral (hematite, magnetite, goethite, etc.). For pigments production, it is needed to obtain magnetite with high purity (above 95%), free from chemical contaminants and with particle size smaller than 100 µm.

Kelly and Spottiswood (1982) points that jig separation is one of the oldest methods of gravity concentration. According to the same authors, even today, jigs are widespread in coal preparation because of its high separation precision, cost-effectiveness and high throughput rate. A jig is a gravity separator that uses pulsation (repeated expansion and contraction of a vertical bed of particles) of water. The result of this movement is the bed stratification, where the materials density splits in ascending order from the top to the bottom (Tsunekawa et al., 2005). Jigs uses has been documented since De Re Metallica (1950) and there are several works describing the use of jigs in fields such as mineral processing and even recycling.

Mukherje and Mishra (2006) used a Denver jig to study the effect of three important process parameters on particle segregation during fluidization stage of the jig cycle (maximum water velocity, feed characteristic, such as size ratio, volume fraction and particle size, and frequency of pulsation). The jig was fed with iron ore (10.0 mm size fractions) from Joda's iron ore mineral processing plant of Tata Steel Limited. Tsunekawa et al. (2005) applied a TACUB jig, from which the Batac jig originated, to separate waste plastics used in copy machine. Grades of 99.8% PS, 99.3% ABS, and 98.6% PET were recovered as the products in the top, middle and bottom layers, respectively. According to the authors, this jig is an excellent separator for particle mixtures from scrapped copy machines because adjusting and controlling water pulsation condition is easy. In another study with the same jig the separation for crushed plastic particles was tested (Hori et al., 2009). Small plastic particles (two types of burn-resistant PE) and PVC with size of 0.5-3 mm and with specific gravities of the two PEs and PVC were about 1.1, 1.3 and 1.4, respectively, were used. The jig separation experiments were carried out under various water pulsations, at which the amplitude, frequency, and pattern of pulsation were varied. High-grade PE and PVC products over 99.8% were recovered under pulsations of small frequency and amplitude.

According to Mukherje and Mishra (2006) in jigging, amplitude and frequency of pulsation, and feed characteristics are the most important process parameters. Although is well accepted that the movement of water through the jig bed is the key to better stratification of particles (Mukherjee et al., 2005). Effects of these parameters on particle segregation during jigging are studied and explained through experimental as well as numerical means by drawing parallel to liquid/solid fluidization process. This is permissible because jigging could also be viewed as a repetitive process of fluidization and defluidization (Mukherje & Mishra, 2006).

Jigs simulation had been done for many authors in order to better understanding its operation. Mukherjee et al. (2005) used an instrumented U-tube jig in a systematic analysis of the movement of water in it, producing a conventional pressurized pulsion and gravity induced suction cycle. A strong correlation between amplitude and frequency of the water waveform was found, which lead to an increase in the operating range of the frequency of pulsation. Models based on Newtonian mechanics was used to study the stratification behaviour of particles in batch U-tube jigs. Mishra and Mehrotra (1998) treated the motion of solid particles using the discrete element method (DEM). Kellerwessel (1998) used radioisotope tracers to verify that the particles in a jig bed not only follow the vertical motion, but also perform a circulatory motion. Evidence of circulatory motion of particles in jigs has been found by following the trajectories of particles using the positron emission particle tracking technique (Clarke et al., 1997). The fluid motion as well as the coupled particle motion in response to the applied pulsation of water has been numerically studied by several means. Mishra and Mehrotra (1997) used DEM to track the position of particles and hence determine the degree of stratification.

The region of Catalão, Brazil, has occurrence of several different mineral species including apatite, barite, magnetite, monazite, niobium, titanite and vermiculite. Currently Anglo American Phosphates produces apatite and barite both through froth flotation. The company tailings production is around 2 Mt per year with magnetite content around 35%. The magnetite co-production could bring two major gains. Firstly, the generation of material, that can be used for pigments production, and secondly the reduction of the amount of material sent to the tailings dam, avoiding the building of new dams and reducing environmental impacts. Gravity concentration methods may be an option to redeem the tailings of phosphate rock for the production of high-grade magnetite. The jig was chosen because of its cost-effectiveness.

METHODOLOGY

Magnetite samples were donated by the Anglo American Phosphates, collected after the low field magnetic separation, a stage previously the phosphate rock froth flotation, and composed by the magnetic concentrate. Quartz sand was used as gang mineral and blended to the magnetite. The choice of quartz sand for the tests was due to its morphology, the chemical composition and density. The quartz sand has a density closes to the apatite and other silicon based minerals present in Anglo American's ore.

The experimental procedure was divided into three stages. Initially magnetite samples were subjected to particle size analysis using laser particle analyser Mastersizer from Malvern, chemical analysis using an X-ray fluorescence spectrometer and analysed using a SEM Jeol JSM 6610 and the punctual chemical composition was determined using an energy dispersive X-ray detector (EDS) Thermo scientific NSS Spectral Imaging. Then magnetite and quartz sand samples at six different granulometry (-500+300, -300+212, -212+150, -150+106, -106+75, -75+53 µm) were blended to obtain samples of 400 g (200 g of each mineral). Finally, the jigging tests were carried out in a laboratory jig Denver model SJ-1015 from Engendrar. This jig has a fixed screen, and a diaphragm provides the water pulsation. A fixed time of two minutes jigging was adopted, being thirty seconds for the jig feeding and 1.5 minutes from the jig process (without any solid feeding). A bed formed by 1.5 kg of steel balls with 8 mm diameter was used. The water flow was fixed at 20 litres per minute and the pulsation frequencies were 102, 188, 274, 359 RPM. Both products (light and heavy) were collected, filtered, dried in an oven, weighted and then subjected to magnetic separation using a rare earth magnet, as proposed by Tomaz et al (2015). All tests were performed in triplicate, totalling 72 tests.

RESULTS AND DISCUSSION

Figure 1 shows the magnetite's particle size analysis. It is possible to see that around 43% of magnetite is smaller than 100 µm and approximately 15% is smaller than 50 µm, therefore with size adequate for pigments industry.

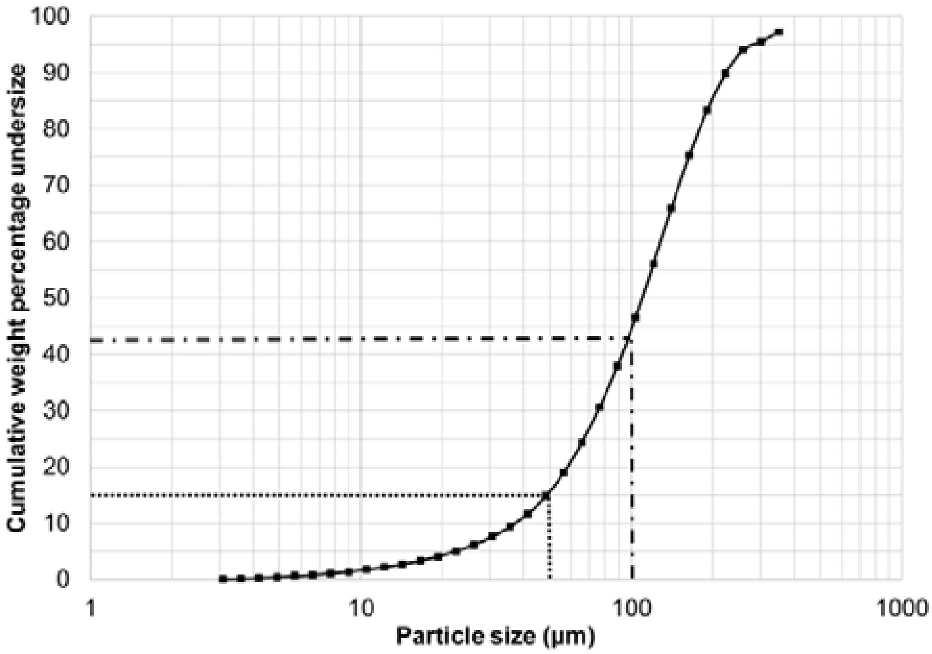


Figure 1 – Magnetite particle size analysis using a laser particle analyser Mastersizer from Malvern

Table 1 shows the magnetite samples chemical analysis per granulometry. Samples showed a high content of iron oxide, around 78% for the coarse size (-500+300 µm) and higher than 84% for the other sizes. By contrast, small amounts of silica, titanium, magnesium and phosphate (less than 10%) and traces of other elements were observed. The P_2O_5 indicated that this sample cannot be used in siderurgy to produce pig iron since phosphorus is a contaminant and its content must be lower than 0.05% (around 50 times less than values observed in the samples).

Table 1 – Results of chemical analysis for magnetite samples separated by particle size trough X-ray fluorescence

Particle size (µm)	Fe_2O_3	TiO_2	CaO	SiO_2	P_2O_5	MgO	MnO	Cr_2O_3	Nb_2O_5	ZrO_2	ZnO	SrO	Others
-500+300	78.943	5.480	3.512	2.691	2.643	1.986	0.601	0.118	0.195	0.132	0.114	0.090	3.495
-300+212	84.036	5.334	2.621	2.419	2.212	1.913	0.737	0.230	0.217	0.131	0.077	0.070	0.003
-212+150	85.312	5.114	2.460	1.706	2.072	1.798	0.720	0.204	0.168	0.116	0.078	0.069	0.183
-150+106	85.367	5.443	2.180	1.315	1.900	2.019	0.765	0.480	0.187	0.135	0.096	0.066	0.047
-106+75	82.186	6.035	2.803	2.776	2.176	2.261	0.829	0.312	0.243	0.170	0.051	0.096	0.062
-75+53	85.624	5.119	2.272	1.596	1.841	1.762	0.598	0.427	0.267	0.126	0.072	0.084	0.212

Table 2 shows the sand quartz samples chemical analysis per granulometry. Samples showed a high content of silica, higher than 90% for all particle sizes. By contrast, small amounts of iron oxide (less or equal to 3%) and traces of other elements were observed.

Table 2 – Results of chemical analysis for quartz sand samples separated by particle size trough X-ray fluorescence

Particle size (μm)	Fe_2O_3	TiO_2	CaO	SiO_2	P_2O_5	MgO	MnO	Cr_2O_3	Nb_2O_5	ZrO_2	ZnO	SrO	Others
-500+300	1.810	0.244	0.257	94.370	0.196	0.000	0.000	0.163	0.000	0.000	2.485	0.100	0.375
-300+212	2.148	0.283	0.075	93.851	0.095	0.199	0.000	0.143	0.000	0.000	3.147	0.000	0.059
-212+150	2.246	1.088	0.310	93.345	0.203	0.000	0.000	0.000	0.027	0.000	2.338	0.000	0.443
-150+106	1.652	0.849	0.000	93.934	0.000	0.000	0.000	0.000	0.000	0.000	2.861	0.160	0.544
-106+75	2.511	1.331	0.417	90.360	0.321	0.233	0.123	0.149	0.088	0.028	3.843	0.000	0.596
-75+53	3.026	1.987	0.229	90.230	0.113	0.237	0.000	0.180	0.294	0.000	3.614	0.000	0.090

Figure 2 shows the SEM results for samples of magnetite and quartz sands. It is possible to notice that the morphology of the particles (both magnetite and quartz sand) are quite similar, even in different sizes.

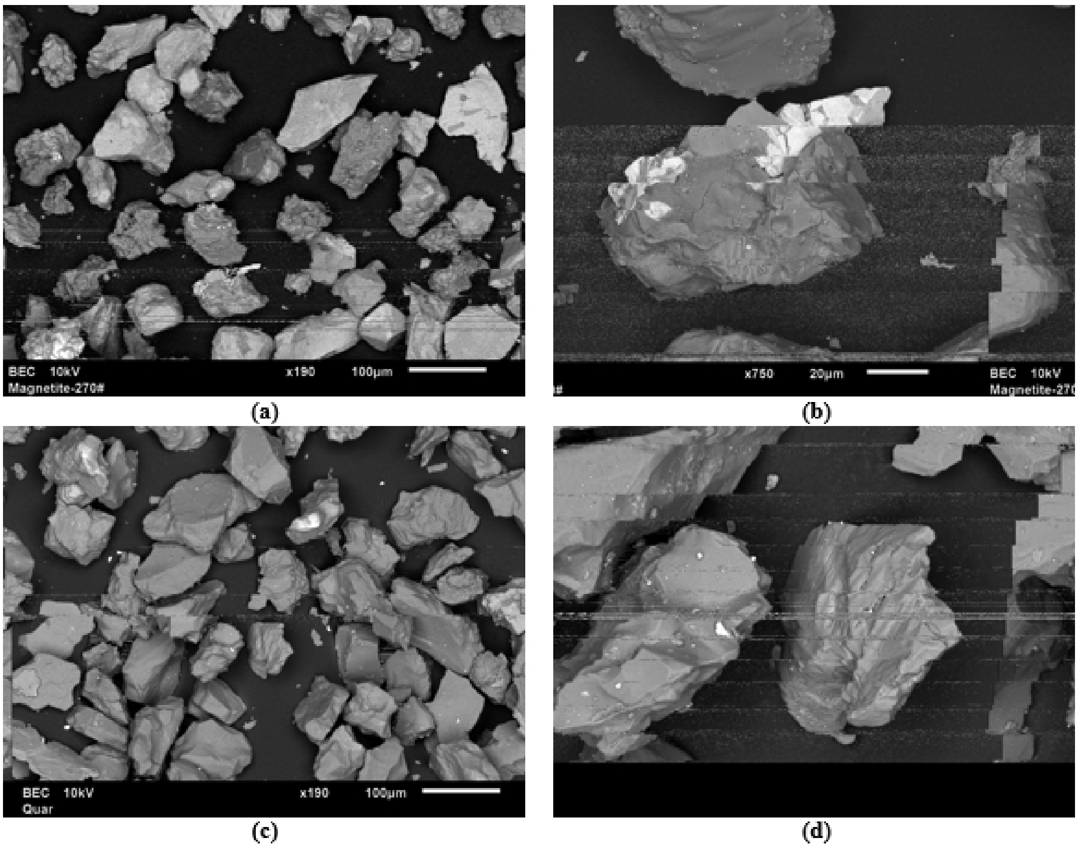
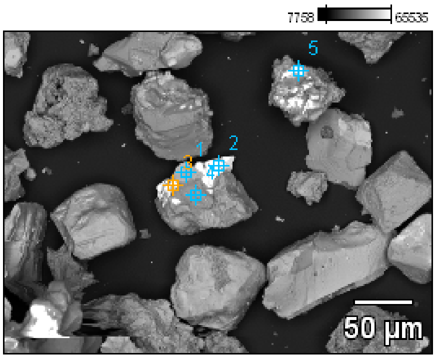
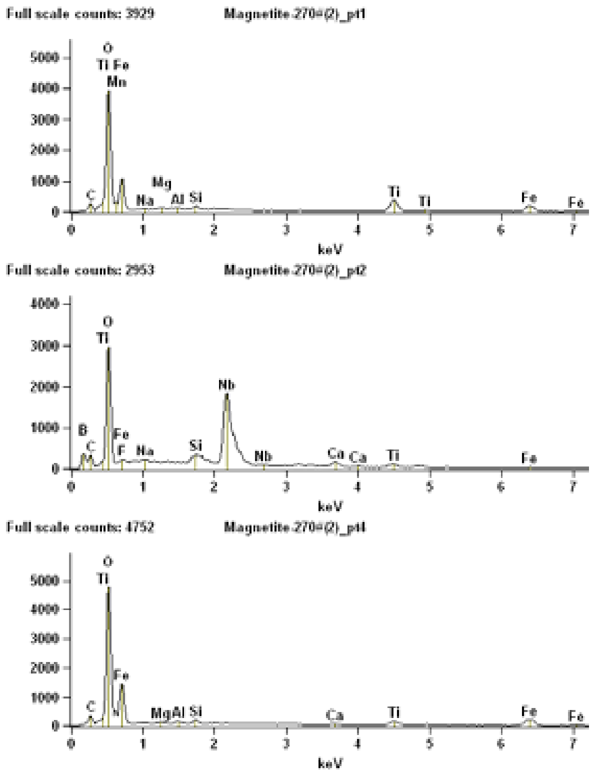


Figure 2 – SEM results for (a) and (b) magnetite sample at -75+53 μm and (c) and (d) quartz sand sample at -106+75 μm

Figures 3 shows the results for EDS analyses in three different points for magnetite samples -75+53 μm and figure 4 for quartz sand -106+75 μm. In both cases, the punctual chemical analyses provided by the EDS were similar to the results obtained through X-ray fluorescence. In figure 3 it is possible to see that niobium bearer minerals are associated my magnetite even in small particle sizes (see point 2 in figure 3a). The liberation degree of these minerals is normally lower at this particle size and further comminution is required before the niobium's froth flotation stages. The average particle size used in this froth flotation is between -53+37 μm. Similar results regarding liberation degree were obtained for quartz sand (see point 5 in figure 4a). The other minerals present in the quartz sand were attached to the quartz particles and not liberated. Since mineral particles were attached in both samples (magnetite and quartz sands) the density of both mineral are expected to be different of pure minerals, but close to it.

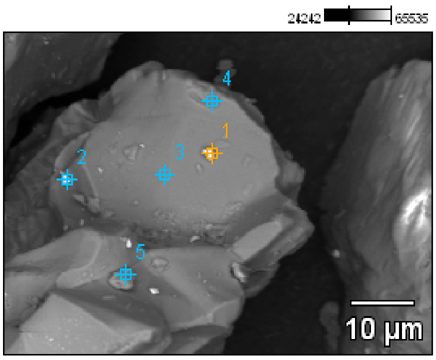


Sample: Magnetite -75+53 µm
Accelerating Voltage: 10.0 kV
Magnification: 330

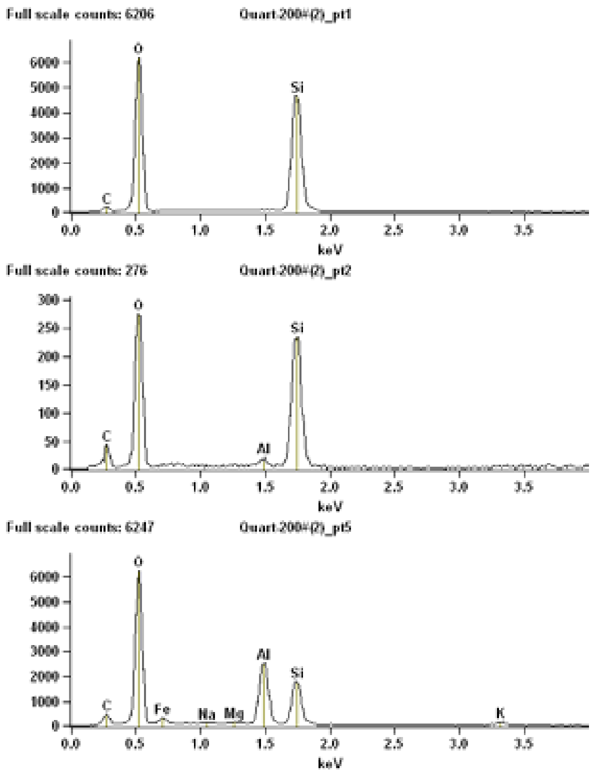


(a) **(b)**

Figure 3 – EDS results for magnetite sample -75+53 µm. (a) SEM image and (b) EDS results for three different points



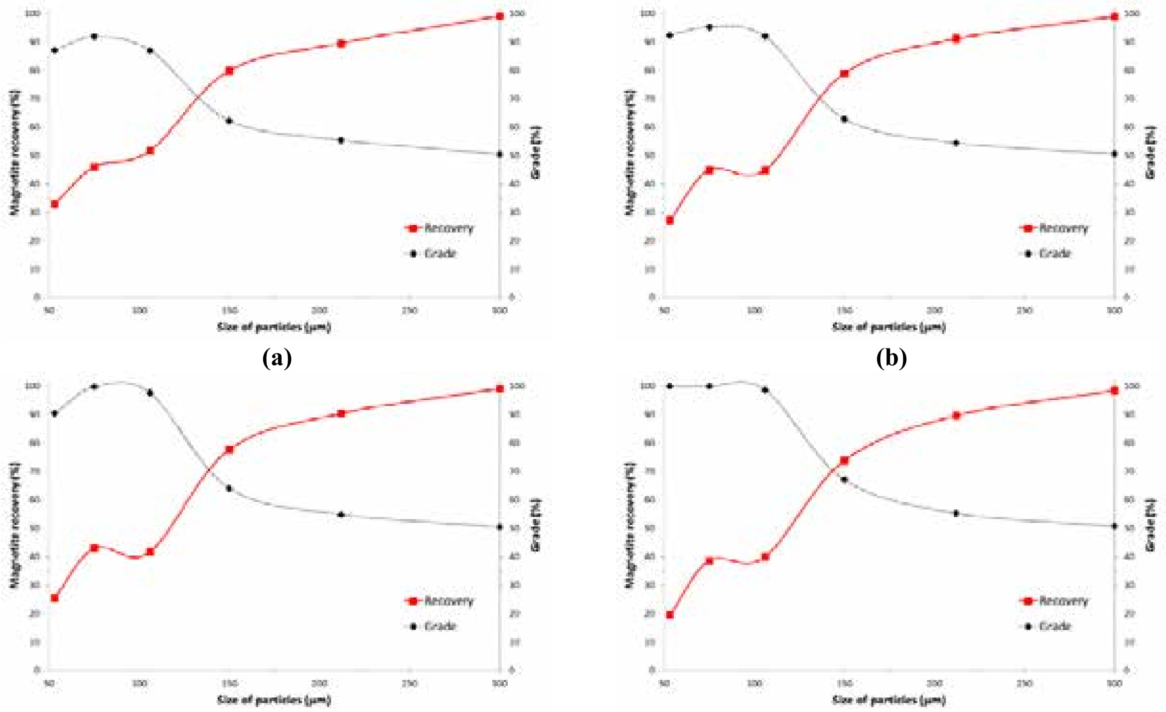
Sample: Quartz sand -106+75 µm
Accelerating Voltage: 10.0 kV
Magnification: 1800



(a) **(b)**

Figure 4 – EDS results for quartz sand sample -106+75 µm. (a) SEM image and (b) EDS results for three different points

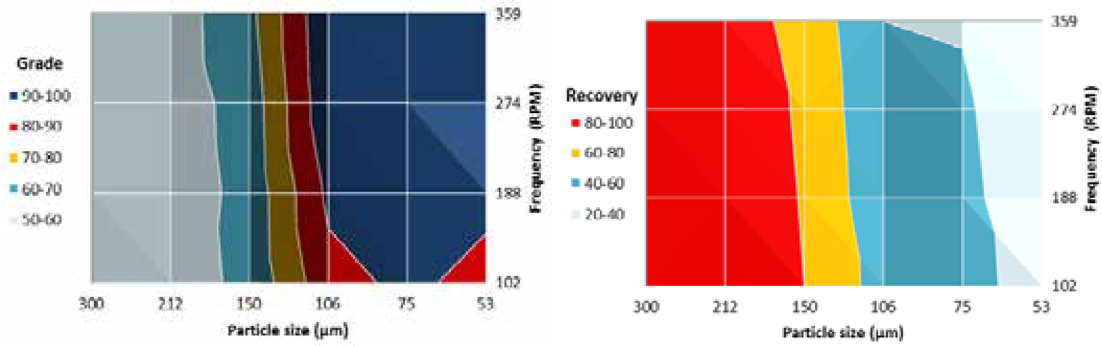
Figure 5 shows the average magnetite recovery and grade in sunk due according the particle size for each tested pulsation frequency. The black dotted lines represent magnetite grade and the red solid lines represent magnetite recovery. The error bar has been suppressed to avoid information overload in the figures. For all tested frequencies, the best values for grades occur in the finer fractions. On the other hand, the best results for magnetite recovery happened for coarser particles (above 150 µm). It can be seen that for pulsation frequencies of 274 and 359 RPM and particle size under 100 µm the magnetite recoveries was around 40% and grades above 95%.



(a) (b) (c) (d)

Figure 5 – Magnetite recovery results and tests grades for each frequency (a) 102 (b) 188 (c) 274 and (d) 359 RPM

Figures 6a shows grades and 6b recoveries obtained versus particle size and jig pulsation frequency, where the warmer colours represent higher levels and cooler colours represent lower levels. It is noted that particle size has more influence on jigging, since at any pulsation frequency was possible to obtain concentrates above 90% magnetite content in the particle sizes under 106 µm.



(a) (b)

Figure 6 – Jigging results in relation to the particle size and pulsation frequency for magnetite (a) grade and (b) recovery

CONCLUSIONS

The test results has shown that there is a magnetite enrichment and a decrease of quartz sand content in the sunk material after the jigging. Thus, it follows that the jigging procedure was effective in separating these two minerals. The samples showed a clear trend of purification for the finer fractions, independent of the adopted pulsation frequency, which means that particle size is a more important factor than the pulsation frequency for this type of jig.

For the production of pigments, it is necessary that the particle size is at most 100 µm, with a purity of 90%. For these parameters, the magnetite recovery was close to 40%. From the granulometric analysis, it is possible to notice that 45% of the material thrown nowadays in the tailings dam is below 100 µm, and thus, if a jig was introduced at the end of the process to produce magnetite, Anglo American Phosphate could produce approximately 360,000 tons per year of magnetite under pigments specification. It is also possible to insert a milling process before the jig, making the tailings recovery even greater.

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INNOVATION AS A COMPARATIVE ADVANTAGE IN THE BRAZILIAN MINING SECTOR

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INNOVATION AS A COMPARATIVE ADVANTAGE IN THE BRAZILIAN MINING SECTOR

ABSTRACT

Countries that are leaders in mineral production or in technological development have reached high levels of competitiveness by adopting robust policies that promote sectorial growth. The challenges that Brazil faces to become more competitive have to do with the need to attract investments for production of mineral goods and the addition of value to raw minerals, which will strengthen the links along the value chains of mineral-based goods, particularly in those sectors with greater comparative advantage. Mining is an industry with a mature technological base, which has evolved at a pace near that of the manufacturing industries. Thus, mineral products in general have had little room for variations in their specifications, especially given the fact that they are commodities. Currently, however, the transformation of mineral-based goods requires inputs with increasingly rigorous specifications in order to meet a variety of new uses and applications. Technological innovation in companies from the mining sector, when not focused on meeting the demand from the processing industry, must aim, as a competitive edge, to increase productivity and reduce production costs, seeking innovation mainly in processes, thus making an important contribution to the equipment supply sector. The increase in process-based productivity, especially in the production of commodities, has evolved in terms of equipment capacity. This improvement has been due to the incorporation of technologies of information and communication as well as innovations in engineering and materials. In order to increase productivity, it is essential to promote innovation, either through expanding access to public and private investments in research and development or through the attraction of innovative companies that would allow for technology transfers in goods and services or establish research centers in the country, or even that support the adoption of innovative business models, allowing Brazil to become a big global player in the mining industry.

KEYWORDS

Innovation, Mining technologies, Competitiveness

INTRODUCTION

The mineral industry has very distinctive characteristics that dictate the standards of performance, innovation and competitiveness of the sector. Firstly, its resources are nonrenewable, which means that, at some point, stocks will be deplete. The geographic distribution of mineral deposits is random and, most often, such deposits are located in hard-to-reach places, far away from consumption centers, resulting in an uneven distribution of deposits relative to regional needs**.** In addition, each mining field has unique features in terms of the quality of its ore (degree of concentration) and the quantity of the mineral targeted for extraction, which generates great economic uncertainty in the early stages of exploration.

It is important to emphasize the high geological and economic risks associated with mining due to the high elasticity of commodity prices and the variability in the degree of concentration of each mineral deposit, as well as the time investment needed to bring projects to maturity. All of these factors need to be considered in the context of high competitive transnational industry. Another important factor to be highlighted concerns the increase in social mobility, especially in developing countries, which leads to ever growing demand for mineral commodities. This contributes significantly to an increase in the intensity of use of mineral-based products. It is estimated that the volume of metals consumed in the last 40 years exceeds the total quantity used since the dawn of Civilization**.**

Given the typical industrialization process of developing countries, future demand tends to increase at a higher rate than the population growth. Thus, it is expected that the per capita consumption in developing countries will grow on par with the increase in their living standards. Furthermore, the drop of in situ levels of minerals extracted in deposits that are more superficial intensifies the need for underground development mines and mineral processing in order to meet the growing demand for mineral goods.

In this scenario, the world will increasingly depend on more expensive ore sources as well as on metal reprocessing. It is important to keep in mind that metal replacement strategy is mere palliative, and only detracts from the inevitable course of shortage. As the stocks of nonrenewable natural resources decrease, the cost of production of each additional unit grows and this loss in efficiency puts mining in conflict with society, as the latter becomes increasingly aware of issues pertaining to environmental conservation and water resource protection.

It is important to note that, when technically well developed, mining is one of the industrial activities that least affect the environment. It also allows for the exploration of new frontiers, thus creating the foundations for geographical and economic development, providing the basis for industrialization by inducing the need for the processing of raw minerals into higher value-added products. Mining provides raw materials for various production chains and promotes the internalization of economic activity, generating demand for the development of infrastructure and services, which in turn lead to the creation of jobs in regions historically plagued by inequality.

In the 1990s, the replacement of mineral *commodities* with new materials as a result of changes in international technological standards, as well as the increased recycling of mineral-based raw materials, have contributed to the decrease in demand. During this period, the reordering of the international division of labor led to a relocation of investments in mining operations to developing countries as these offered more lenient legal environments and the promise of higher profits obtainable from projects.

The first decade of the second millennium changed significantly this scenario that has being considered a boom time for mining. The end of the period of economic stagnation and gradual resumption of growth of nations intensified the demand for minerals with respect to the supply, raising the price of major commodities and the profitability of the industry. As a result, pension funds in developed countries began to add mineral commodities to their investment portfolios. The growth and accelerated industrialization of China, at the time, was the main driver of this boom..

THE BRAZILIAN MINING INDUSTRY

Many sectors in the Brazilian economy have high relevance in the global production chain. It is important to mention that the country's mineral and metallurgical industries are key suppliers for the machine/equipment, transport and energy industries, among others. In Brazil, mining is the main source of fundraising and constitutes an important regional development vector.

In this context, the Brazilian mining sector registered, in the first half of 2015, exports of US$20 billion and imports of US$ 12.5 billion. The balance of trade surplus generated US$ 7.5 billion in this period. In 2014, exports reached US$ 47.4 billion and imports US$ 12.32 billion. The participation of exports from the mineral sector in the total of Brazilian exports amounted to 21% (iron ore exports alone accounted for 11.5% of the total Brazilian exports).

Therefore, mining carries relevant weight in industrial production and exports in Brazil - with iron ore being its main export item with 11.5% of the total Brazilian exports. The supply structure is highly concentrated in a small group of large companies - mainly in the iron ore industry - although it has several other kinds of different sizes and levels of technological development. An important destination of Brazil's mineral production is the international market, which absorbs a significant part of mineral goods currently produced in the country, with iron ore being the main focus of production.

THE MINING SUPPLY CHAIN

Global mining is a capital-intensive economic activity, and its technology-intensive processes focus mainly on increasing production efficiency and on cutting costs. The stiff competition that takes place at a global scale - virtually all across the mining industry - has prompted initiatives which led to technological advances. The sector has invested in the acquisition of increasingly sophisticated machines to raise scale and, thus, productivity.

The technologically disruptive advances of the mining sector, in general, are the result of the efforts of its suppliers to promote research and development, especially in the segment of mining machines and equipment, which requires a lot of capital and significant engineering capability. Thus, mining companies often can benefit from the results of investments made by its suppliers in the development of bigger and better machines and equipment. (BARTOS, 2007 and FAROOKI, 2012), *apud* BERTASO & CUNHA, 2013.

In this scenario, some companies have made investments towards the development of integrated mines, using automated equipment. Automation reduces the need for labor, raises the safety of operations by lowering the risk of accidents by workers employed in the mines, allowing for significant gains in productivity - which includes a reduction in the number of maintenance stops.

It is important to highlight that some Brazilian mining companies have been adopting strategies towards automating their operations in spite of the fact that current conditions do not necessitate such technological intensity, given that labor shortage is still not a pressing reality in Brazil and that labor costs do not constitute a burden to local mining companies, as in the case of some Australian mines, for instance (BERTASO & CUNHA, 2013).

In addition, the development of higher technology in mines increases the competitiveness of mining companies. On the other hand, the investment necessary for the near-term modernization and automation of the mineral sector should aim to increase efficiency; however, gains in scale will depend on the international economic scenario.

Nonetheless, as the world's economy still suffers from the effects of the global crisis, high capital costs need to be offset by gains in productivity through increased labor specialization so as to meet the technical requirements of new equipment . Perhaps some of the more complex information systems available may not be attractive enough considering low commodity prices, which currently have returned to historic levels..

TECHNOLOGY TRENDS AND INNOVATION

Technological trends within the mineral sector reveal a growing use of information technology and automation in the mines and ore-processing units. This is especially true in the segment that produces machines and equipment for the development of mining technology, with major advances in Brazil and in the world at large.

Constant improvements are made to portable equipment that facilitates content analysis and mineral exploration, as, for instance, the detection of metals, minerals and contaminants in the field, with instant mapping by means of GPS, and GIS technologies. The results are generated immediately, helping to determine the next action to be performed throughout the mining process, such as exploration, classification/QA process or environmental sustainability.

There is also some sophisticated equipment for probing, as well as computer programs used for modelling geological deposits. Another major breakthrough is the use of dynamic positioning of drone sensors to view targets of the research areas.

Local and international mining activities have developed new technological solutions in production processes, such as continuous mining, very large trucks, and the use of robotics in the processing phases. Recently, many companies have used simulators on worker-training operations. Depending on the production process of the mineral good, some innovative techniques can be observed in mining operations that carry out mineral dissolution by drilling horizontal holes, or that manage to extract iron ore by using high-power water jet systems, among other technologies.

Underground mining, for example, requires higher technological intensity in several operations, such as the use of drilling, disassembly, loading and transportation equipment adapted to the confined conditions of the galleries. Some highlights of such operations are robotic drills and pipes with high-strength alloys, which allow for greater depth with controlled risks.

In underground mining, there is a need for the adoption of ceiling support systems, lighting systems, and mine ventilation systems, which are factors that demand high Capex and Opex. The technological intensification of such operations is made more necessary due to concerns over environmental impacts as well as the safety of the mines. For instance, deep underground operationscan lead to underground water contamination. In sum, environmental issues and job security - typical mining industry issues - are amplified in underground mines, and in andin a scenario of scarcity of mineral goods, a movement towards growing technological intensity and sea-bed mineral exploration will become predominant in the foreseeable future.

In mineral processing, pressurized flotation cells have led to improvements in tank performance, leading to energy efficiency gains with volume increases and cylinder shape changes.

As for mineral-based final products, innovation efforts will focus more on development of new materials, particularly alloys, as well as on customization processes. This is particularly true for technical specifications for higher resistance to abrasion, corrosion, thermal or electrical conductivity. In these cases, some mining companies are advancing one or two steps in the value chain to tap into a market of higher value-added products. These new, lighter, and more efficient materials (steel and other alloys/compounds), as well as 2D-Graphene-based materials launch a new era of foward-looking innovations. Also currently used are digital platforms for manufacturing customized materials at smaller scales, as well as increased specialization associated to equipment prototyping, which includes 3D printing for customized production.

Additionally, Brazilian companies' advanced technology also have a strong presence and lead the industry in such areas as automation systems, information technology and equipment for environmental control. Some highlights of technologies currently in use are as follows: intensive use of information technologies and automation in mining and processing operations; increasing concern with the minimization of environmental impacts and risks related to the safety of mines; big data services of high technological content for the development and maintenance of geological data, operation models, logistics and market. In order to adapt traditional equipment to a Big Data environment, the system should receive and transmit efficiently information, allowing for greater operational control of mines and processing plants. Also noteworthy are digital platforms for operational simulations and mine development, autonomous transport systems that do not require mining operators as they run with smart sensors, and the control of production parameters. In terms of the reduction of risk and of environmental impacts, new technologies have sought to minimize the ROM, the tailings disposal and water use.

The aspect that stand out as trends in the development of mining technology production is the increasing sophistication of equipment by the suppliers. This often requires a close interaction between equipment producers and mining companies. Many manufacturers have formed strategic alliances with mining companies for product development in order to reduce risks. This migth have significant effects on the productivity of its mining activities in the near future.

The technological development of mining sites has involved the direct participation of foreign producers and engineering services, in many cases by specialized foreign consulting that helps to execute and improve extraction projects.

Countries such as Canada, Australia, South Africa and Chile have some of the largest concentration of mines, with a network of support institutions, such as universities, research institutions, and underground mining equipment suppliers – all of that are benchmarks to Brazil's supply chain.

Due to the small number of underground operations in Brazil, which leads to lack of scale in the provision of services, mining companies end up having to implement mechanical and electrical maintenance activities themselves. These operations sometimes are equal in importance to production itself. Furthermore, as the support equipment for modern mining has to be imported, at high costs, logistical hurdles and customs clearing issues lead companies to stock up on spare parts or to replicate them in their factories (BERTASO & CUNHA , 2013).

The low interaction between local research centers and businesses is characterized as a structural phenomenon for several Brazilian industry segments.. Regarding engineering development and manufacture of mining machinery and equipment, the rapid industry growth is leading major changes not only in the mining activity level, but also in the supply.

The evolution of equipment and extraction techniques has enabled productivity gains along the different stages of the production process, leading to cost reduction and increased efficiency. Improvements in competitiveness requires monitoring the technological features related not only to extraction and processing operations, but also transport, storage, distribution and product application. Constantly updated technological standards must be applied to equipment, techniques and processes, and the concept of quality management and productivity must be adopted, aiming for the continuous optimization of functions pertaining to deadlines, costs and quality control throughout the production chain.

Brazil's supply market for capital goods is quite expressive, and the quality of its technical support services is comparable to that observed in worldwide mining. The structure of this equipment market is mature and many of the world largest suppliers of mining equipment are established in Brazil, some with a large manufacturing sites.

The presence of the largest global suppliers of mining equipment in Brazil ensures that the technological quality of products offered is compatible with international standards. The competitive advantage is, therefore, often based on additional services that a company can offer. This conceptual approach has also led equipment vendors to provide engineers to all major Brazilian mining areas in order to carry out maintenance and training activities.

The growing emphasis on additional services extends to multinational companies operating in Brazil. Companies installed here offer their customers the same level of service and training offered in Europe. This is a key element of competitiveness for equipment suppliers aiming to be successful in the Brazilian mining industry (BERTASO & CUNHA, 2013).

As the Brazilian mining market continues to expand into strategic areas with high mineral potential, reliability and productivity, national manufacturing have been increasing the choice of equipment and service package suppliers operating in Brazil. This trend strengthens Brazilian mining and engineering services, which in recent years have registered strong growth associated with large investments made in the industrial production chain as a whole. In addition to having access in the country to the best international consulting firms in mining engineering, the services market can also count on the expertise of qualified Brazilian firms that are capable to develop advanced technologies in order to overcome the country's mining challenges.

The vast majority of engineers and consultants who have been working in the domestic market is made up of Brazilian professionals that mitigate the challenges posed by the need for highly qualified work, despite the fact that the supply of such professionals tends not to be enough.

An increasing number of mineral engineering Brazilian companies is expanding into the global market with a particular focus on some regions in Africa, Latin America and Europe. This trend is reinforced by the increase in international partnerships or acquisitions of local firms by international engineering and consulting companies.

RESEARCH, DEVELOPMENT AND INNOVATION IN BRAZILIAN MINING

The Brazilian mining market is gaining autonomy and can become less dependent on the research, development and innovation held abroad to develop its industry.

Brazil's mining services sector, despite its high level of technological development, still has extraordinary potential for growth vis a vis the challenges facing the country's mining industry. The Brazilian mining industry should structure itself to compete in the international market with products of higher added value.

The engagement of the federal and state level government with local industry executives is a paramount to ensure an optimal balance among demand, local development and the foreign suppliers share of industrial goods in Brazil. The market seems strong and large enough to accommodate the demand and ambition of all business players.

Nevertheless, global private investment in research and development by the non-ferrous mining and metallurgy companies has declined over the years. Technological-intensity-oriented spending in R,D&I has declined in recent decades: from 0.7-0.8%, in 1980-1990, to 0.2-0.5% in the 2000s. This movement is explained by an intense process of acquisitions and mergers. In addition, there is a cost-sharing trend of R,D&I through research consortia, in pre-competitive projects, with the participation of Governments and institutions of science and technology. Although commonly practiced in Australia and Canada, they are still y rare in Brazil.

There have been no significant changes in terms of the regulatory framework for mining activities at the international level that would provide incentives for investments in R, D & I. In the context of the approval of a new Brazilian Mining Code, it is important to know that the new Law will contain a provision pertaining to the allocation of a portion of the net operational revenue from mining enterprises to fund R&D projects, as well as environmental programs related to the respective mining activities, which shall include the participation of the enterprises themselves. Such legal mechanism would induce the development of competitive technologies, thus raising contracted enterprises to better positions in global value chain.

In addition, as a public policy, it is important that the mining industry's helps to build programs to be supported by the Government, which does not necessarily involve the creation of new instruments or credit lines. This support should be geared towards the development and pioneering production of machinery, equipment, software and systems for mining.

Brazil just have launched a program that helps mining companies or their suppliers to gain access to available funds, leveraging their business through the offering of reimbursable funds or support lines with non-reimbursable funds. In order to improve the competitiveness of the Brazilian mining sector, this new financial support tool is able to funding the development of technologies and products related to Strategic Minerals, including rare earths, among others. This should also target products and technologies with high mineral trade deficit. Another criterion could involve the development, improvement and the scaling-up of mining technologies.

Due to increasing environmental requirements placed on mining activities, special attention would need to be given to technology and processes aimed at reducing and mitigating the risk of environmental impacts. The decentralization of environmental resources and the application of R, D & I is another fundamental aspect, since the mineral activities are numerous and are scattered throughout the national territory. That would include expenditures for the mitigation of environmental impacts directly over affected areas and communities.

Finally, the continuous application of funds in research and recovery of impacted areas simultaneously to the cycle of productive activity, and not only at its end, when economic and financial capacity is in decline, as well as the generation of new technology-based businesses and environmental support, partnership and even ownership of mining companies, are urgent challenges to be overcome.

CONCLUSION

In order for innovation to effectively become a competitive advantage for the Brazilian mining sector, it is necessary to strengthen its technological and mineral engineering structures. To this end, the country should intensify the actions of technological development and innovation along the whole supply chain of the mining industry, as well as promote the addition of value to mineral commodities in the country. Another key aspect for raising competitiveness is the diversification of economic activities away from the mineral endowment of different micro-regions, where intense mining activity take place. Therefore, it is necessary to promote the convergence of actions that stimulate competitiveness, so as to increase global value chain integration, thus positioning Brazil as a major player in the mining supply chain serving domestic and foreign markets. To conclude, it is necessary to redefine Brazil’s role, so that it can go from being a mere supplier of raw mineral resources to a supplier of high-technology goods and services for the world mining industry.

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INNOVATIVE SOLUTIONS OF MINING MACHINERY FOR UNDERGROUND APPLICATIONS – COMMERCIALIZATION OF RESEARCH RESULTS

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INNOVATIVE SOLUTIONS OF MINING MACHINERY FOR UNDERGROUND APPLICATIONS – COMMERCIALIZATION OF RESEARCH RESULTS

ABSTRACT

The paper presents results of scientific, research, and technical projects of innovative character realized at the KOMAG Institute of Mining Technology. Some mechanical and mechatronic systems are described in the aspect of safety, reliability, and operational efficiency. Special attention is paid to equipment used for controlling dust and methane hazards as well as to computer systems for control, monitoring, and diagnostics of mining machines. The paper is ended with a description of the KOMAG's role in the European Research Area and with some information concerning an integrated commercialization model of research results.

KEYWORDS

Mining, machinery, innovation, research, safety, commercialization

INTRODUCTION – KOMAG'S SCOPE OF ACTIVITY

KOMAG Institute of Mining Technology plays an important role in improving operational safety and efficiency of mining machines. Relationships among an operator, a machine, and work environment are investigated in research projects of interdisciplinary character. Designs, tests, and industrial implementations of research results are within the scope of its basic activities offered to machinery producers and users. KOMAG is an EU Notified Body within the following directives:

- Directive 2006/42/EC on machinery,
- Directive 94/9/EC concerning equipment and protective systems intended for use in potentially explosive atmospheres (ATEX),
- Directive 2006/95/EC relating to electrical equipment designed for use within certain voltage limits (Low Voltage).

It is a research and development centre of organizational and proprietary structure adapted to the market activity in the European Research Area and of the organizational culture creating a friendly climate for generating new ideas and transforming them into innovative products. Its scope of activity covers mechanical and mechatronic systems, machines and equipment for underground mining and processing of minerals, control, diagnostic and monitoring systems, environmental protection, ergonomics and work safety, model, laboratory and in-situ tests of machines and equipment, technology transfer, assessment and certification of machines as well as training activities aimed at sharing knowledge and dissemination of best practises. A realization of ambitious, interdisciplinary projects, in collaboration with Polish and foreign scientific and industrial partners, is successful due to advanced testing infrastructure and a staff of highly qualified researchers. The European Commission nominated KOMAG to a position of the Centre of Excellence in the domain of mechanical systems which are operator and environment friendly.

INNOVATIVE SOLUTIONS OF MINING MACHINES AND EQUIPMENT DEVELOPED AT KOMAG

Development of KOMAG Dust Control Systems

KOMAG realizes research projects oriented onto ventilation and dust control systems, in particular developing innovative solutions of spraying installations, dust control equipment, vortex ventubes, ventube fans, and auxiliary equipment such as aerating chambers and blowers. In roadway workings, driven with roadheaders, ventilation and dust control installations enable to obtain a proper air composition and a neutralization of methane and dust hazards. In roadway and longwall faces, on the run-of-mine haulage routes and in conveyor discharge areas spraying systems are used. At present UO-type dust collectors are commonly used in Polish coal mines. An example of such equipment is shown in Figure 1. Technical specification of UO series is given in Table 1.

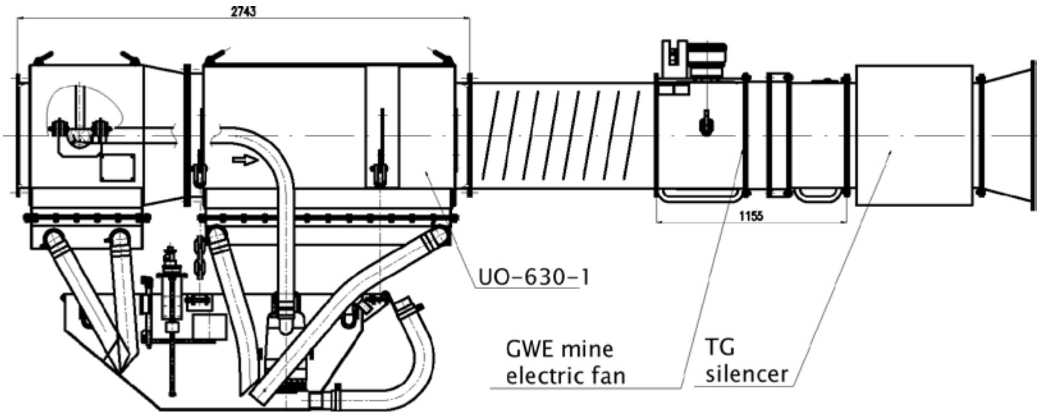

Figure 1 – IO-630 dust collector operating together with mine electric fan of GWE type equipped with silencer (Jedziniak and Prostanski, 2013)

Table 1 – Technical parameters of UO dust collectors (Jedziniak and Prostanski, 2013)

Technical specification	UO-630-1	UO-630-2	UO-1000
Major diameter of spraying chamber and drop collector	630 mm	800 mm	1000 mm
Output	200 ÷ 400 m^3/min	250 ÷ 450 m^3/min	400 ÷ 800 m^3/min
Water consumption	1 ÷ 10 dm^3/min	1 ÷ 10 dm^3/min	1 ÷ 15 dm^3/min
Dust collecting efficiency	98.4 ÷ 99.5 %	98.4 ÷ 99.5 %	97.4 ÷ 99.7 %
Flow resistance at output of 300 m^3/min	about 1500 Pa	about 1420 Pa	about 1500 Pa

New directions of research work are oriented onto increasing efficiency of dust control. In the case of shearers and roadheaders warming-up of cut grooves and tips of cutting bits to high temperatures is, apart from dust generation, a side effect of coal cutting process. Also sparks can be generated then. When methane releases from the solid coal, there is a risk of its ignition and a risk of methane-and-air mixture explosion hazard. An innovative technical solution was developed at KOMAG in collaboration with the "Barbara" Experimental Mine. The stand for testing efficiency of protecting and extinguishing methane ignition was installed in a model of a roadway. The testing roadway was closed at both ends. On one side an inlet window was installed and on the other side an entry gate was made. Inside the roadway a model of coal wall, that precisely reproduced the shape in natural conditions, was built. The roadway was equipped with indispensable electric equipment and measuring instrumentation to record and analyze the test results as regards efficiency of extinguishing burning gas as well as gas ignition prevention. The tests showed that a strict relationship between the parameters of compressed air and water is an important factor enabling to achieve the required efficiency of spraying installations, i.e. water consumption 30-34 l/min, at pressure 0.4 MPa and compressed air consumption 1.3-1.7 m^3/min at pressure 0.4 MPa. Time of extinguishing burning flames in the distance 0.1 m from the face varied from 5 s to 35 s, but when the burning gas was half way of the cutting drum web depth, extinguishing of flames was instantaneous which is illustrated in

Figure 2 and Figure 3. The system was installed in the KSW-460NE shearer produced by the KOPEX Machinery and tested in the Pniowek mine.

Figure 2 – Air-and-water installation on a shearer

Figure 3 – Extinguishing of flames on a shearer

Positive results of longwall shearer tests, confirmed in operational conditions, aroused interest of a manufacturer of roadheaders. A prototype air-and-water spraying system was installed on a R-200 roadheader, which is shown in Figure 4. In-situ tests were carried out in the Murcki mine, giving positive results and confirming the benefits resulting from an application of this innovative air-and-water spraying system.

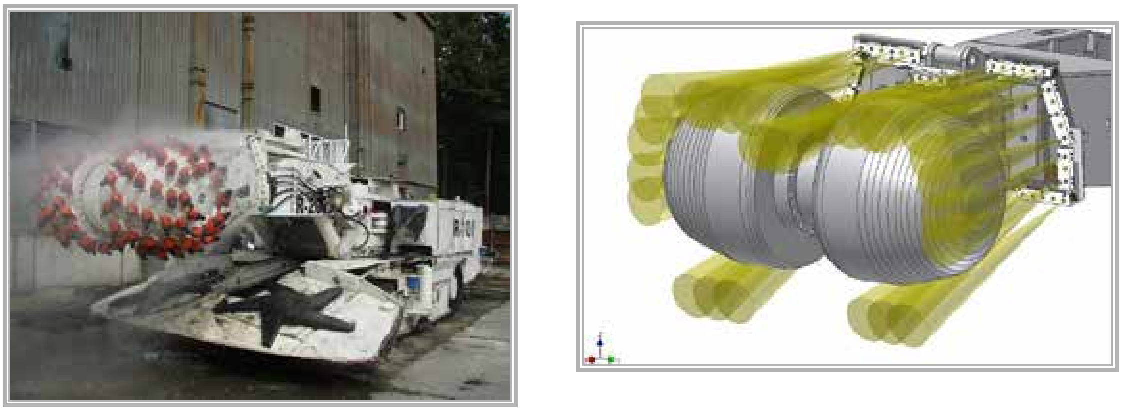

Figure 4 – Air-and-water curtain of roadheader

Advanced Technical Solutions Implemented in the KSW-800 NE Shearer

The KSW-800 NE shearer was a result of a research project co-financed by the National Centre for Research and Development. It was conducted in close collaboration with two companies: KOPEX Machinery and KOPEX Electric Systems. It is designed for bi-directional, mechanical coal cutting and loading in a longwall, stable-less system on an AFC of 850÷1100 mm width with a chainless haulage system of Eicotrack type. It has a self-supporting construction and it is designed for use in highly productive longwall systems, operating in seams which are from 1.4 to 3.5 metres thick. Several innovative design solutions, improving work safety and comfort of operators were implemented in the KSW-800 NE shearer. The machine is equipped with a compact assembly of frequency converters with the energy recuperation and a system of dual data transmission, using Ethernet. It is important to mention that an access to shearer components is facilitated due to an implementation of hydraulic nuts and special protecting inserts. The machine is equipped with a state-of-the-art diagnostic system, an identification system of machine components and of operational conditions. They incorporate the following subsystems:

- vibrodiagnostics used for detecting failures of gears in changeable operational conditions,
- a subsystem for an observation of operational conditions using vision cameras at the face ends,
- shearer diagnostics using thermovision camera,
- Interactive Operational Manual.

The KSW-800 NE shearer was exhibited at the International Fair of Mining, Power Industry and Metallurgy Katowice 2015, what is illustrated in Figure 5.

Figure 5 – KSW-800 NE shearer exhibited at Katowice Fair

Its detailed specification is given in Table 2.

Table 2 – Technical specification of KSW-800 NE shearer

Technical specification	Value
Scope of cutting [m]	1.4 ÷ 3.5
Maximum installed power [kW]	766
- cutting drums	2 x 30
- haulage system	2 x 75
- hydraulic system	2 x 8
Supply voltage [V]	3300
Diameter of cutting drums [mm]	1350 ÷ 1800
Web depth [mm]	800 ÷ 1000
Pull force [kN]	2 x 400
Haulage speed [m/min]	0 ÷ 30
Minimum height [mm]	~ 1000
Weight [t]	~ 46.0

The presented shearer is an ergonomic design solution. It consumes significantly less electric energy than other machines of similar type and size. It guarantees a failure-free operation and safe operational conditions. Its overall dimensions are also advantageous, i.e. the highest power concentration in relation to overall dimensions.

INSTO – System of Interactive Operational Manuals

The sources of knowledge, indispensable for a correct operation and maintenance of mining machines, include: experience, training, and operational manuals. Getting knowledge from experience is time consuming. The efficiency of trainings does not seem satisfactory due to complicated construction of machines. Operational manuals which are most commonly used at present are in a form of printouts, consisting of a text and static drawings. Such operational manuals have several limitations such as:

- limited possibilities of presenting knowledge, in particular as regards maintenance, transport etc.,
- difficult search for information meeting instant needs of a user,
- difficult up-date and broadening of knowledge.

System of Interactive Operational Manuals consists of three applications: INSTO-1, INSTO-2, and INSTO-3. They are state-of-the-art alternatives to traditional operational manuals used in the mining industry. They have the following advantages:

- a presentation of knowledge in a clear and easy to understand manner,
- a presentation of some aspects of knowledge, which can be disseminated at present only practically during trainings as it is not possible to describe them with traditional forms of presenting knowledge using text and static drawings,
- a quick access to knowledge according to current needs,
- an access to knowledge at any time and at any place, independently of the conditions.

The INSTO System offers new possibilities in the scope of operational knowledge dissemination indispensable for companies using mining machines as well as companies rendering maintenance services. The INSTO applications are based on state-of-the-art ICT solutions incorporating computer modelling and VR technologies. The INSTO-1 is used in the client-server mode. The INSTO-2 and INSTO-3 are installed on individual computers. The user's interface is shown in Figure 6.

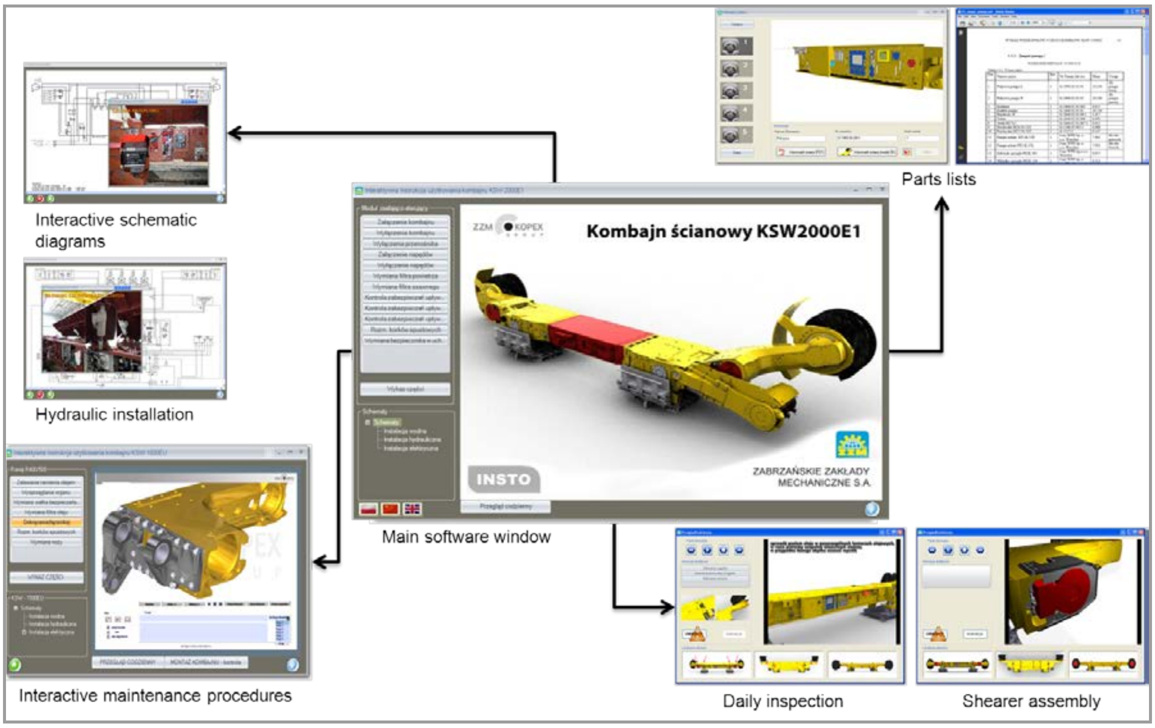

Figure 6 – INSTO Interactive Operational Manuals – user's interface (Michalak et al., 2015)

Equipment platforms enabling to use the INSTO System are shown in Figure 7.

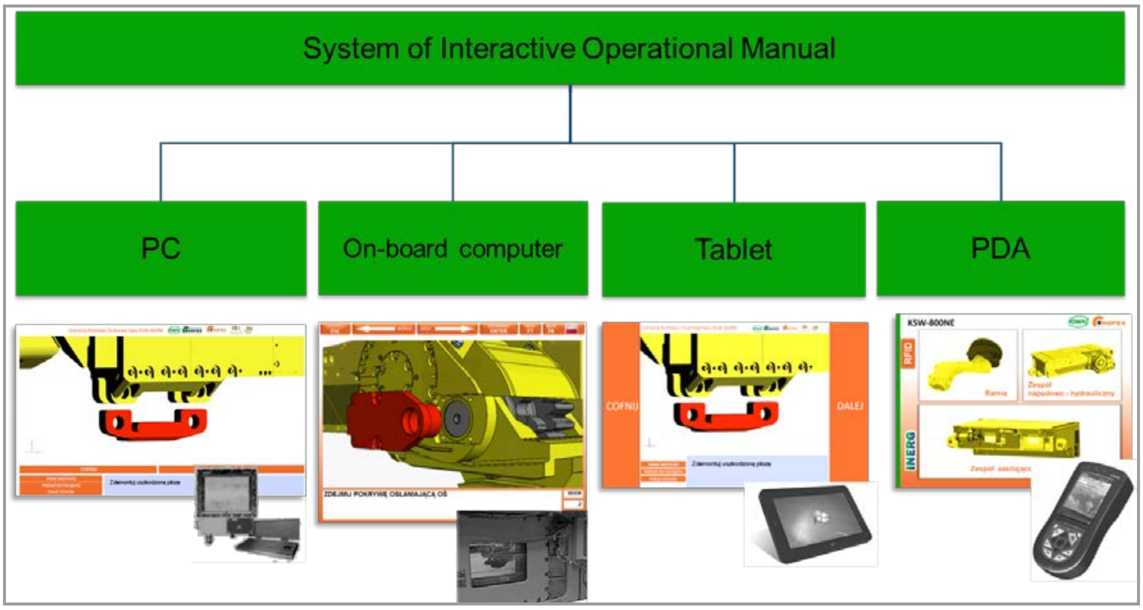

Figure 7 – INSTO Interactive Operational Manuals – equipment platforms (Michalak et al., 2015)

The INSTO System hardware and software requirements are as follows:

- a personal computer equipped with at least Intel Pentium 4, minimum 2GB of RAM memory, graphic card 256 MB RAM, Microsoft Direct X, version 9,
- Windows operational system (XP, Vista, Win7), Acrobat Reader 9 or higher, NET Framework 3.5 or higher.

One of the advantages of the INSTO system is a possibility of starting it directly from a CD. There is no need for its installation on a computer disc. It does not require an Internet connection, which usually causes problems in underground conditions.

iRIS – System of Electronic Evidence of Fixed Assets in Mining Plants

The iRIS Intelligent Rapid Identification System is a complex tool for an electronic fixed assets register. It consists of five platforms used for marking and identification of the following fixed assets:

- PECM – machines, equipment, and components used in underground workings,
- PEUBP – machines and equipment of explosion-proof construction,
- PEST – means of transport,
- PEMP – machines and equipment used on surface,
- PESTB – office equipment.

A structure of the iRIS System is shown in Figure 8. It is of modular type and can be adapted to individual users' needs.

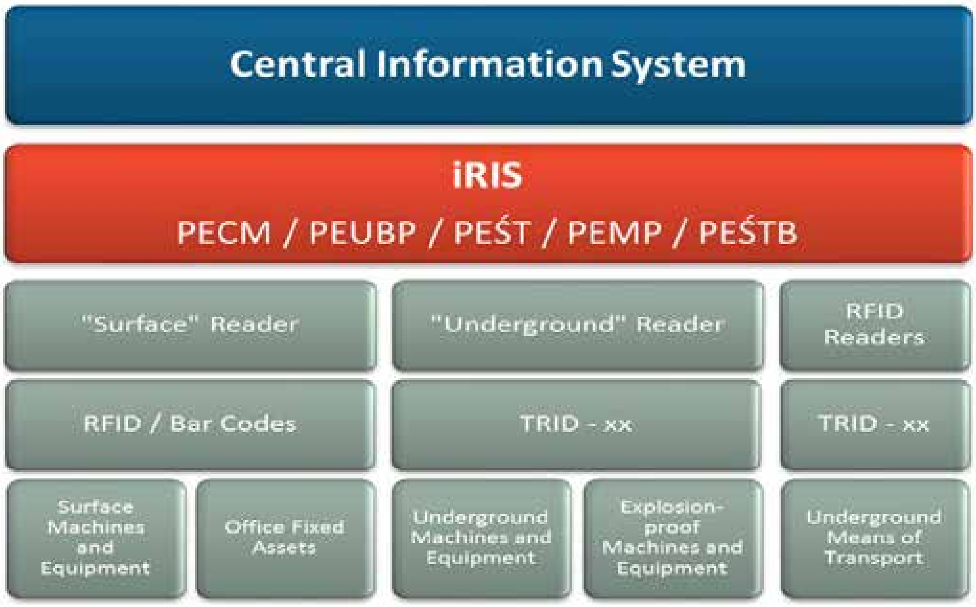

Figure 8 – Structure of iRIS System

iRIS is a reliable tool enabling its users to manage fixed assets in a relatively simple way, facilitating and speeding up stocktaking activities. The hardware and the software are sources of credible information about a technical condition of machinery components. A new model of the iRIS System guarantees its functional flexibility, meeting the legal requirements. Its basic advantages are as follows:

- unmistakable identification of fixed assets both on mine surface as well as in underground workings where an explosion hazard occurs,
- credible information about the previous operation of machines and equipment,
- use of the state-of-the-art computer data bases and analytical tools for storing and processing the data concerning fixed assets and a possibility of obtaining an easy access to them,
- long marking life,
- reliability of the identification code readout in difficult operational conditions.

KOMAG IN THE EUROPEAN RESEARCH AREA

KOMAG started a realization of European international projects in 2002 with the MECHSYS Centre of Excellence project enabling an improvement and strengthening the abilities of the Institute to achieve a full integration with the European Research Area. The project facilitated a participation in international conferences, seminars, and workshops, giving a chance for establishing collaborative links with leading European research centres. One of the first projects was IAMTECH, co-financed by the Research Fund for Coal and Steel. Its basic objective was oriented onto increasing the efficiency of roadway drivage through the application of advanced information, automation, and maintenance technologies. A visualisation of the main roadheader components is illustrated in Figure 9 as one of the examples of the project tasks.

Afterwards KOMAG researchers were invited to one of the biggest projects realized within the Sixth Framework Programme for Research, Technical Development and Presentation. The VIRTHUALIS project consortium consisted of 49 participants from 17 countries. It concerned a use of Virtual Reality and Human Factors Applications for Improving Safety. Its main objective was a control of hazards in production facilities as well as warehouses. A presentation of Augmented Reality Technology is illustrated in Figure 10.

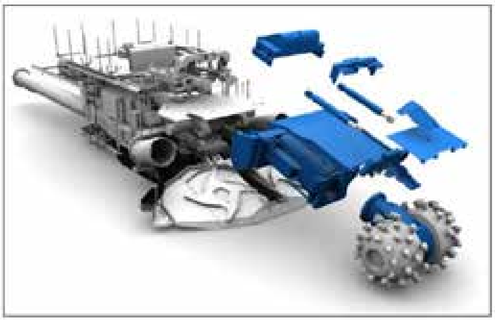

Figure 9 – Visualisation of main roadheader components (Hordyniak, 2015)

Figure 10 – A presentation of Augmented Reality technology at the Technical University of Milan (Hordyniak, 2015)

A strong position of the KOMAG Institute in the mining branch can be seen in different research projects, co-financed by the Research Fund for Coal and Steel (Hordyniak, 2015). It is worth mentioning a few of them:

- NEMAEQ – New mechanization and automation of longwall and roadway drivage equipment,
- ADRIS – Advanced drivage and roadheading intelligent system,
- MINTOS – Improving mining transport reliability,
- EMIMSAR – Enhanced miner – information interaction to improve maintenance and safety with Augmented Reality,
- MINFIREX – Minimising risk and reducing impact of fire and explosion hazards in underground coal mining,
- INREQ – Enhanced effectiveness and safety of rescuers involved in high risk activities by designing innovative rescue equipment systems,
- BEVEXMIN – Bucket wheel excavators operating under difficult mining conditions including unmineable inclusions and geological structures with excessive mining resistance.

Since 2008 KOMAG has been an active member of the European Association for Coal and Lignite EURACOAL, which is the umbrella organisation of the European coal industry. It has 34 members including national coal associations, importers' associations, research institutes, and individual companies. Members come from 20 countries: Belgium, Bosnia-Herzegovina, Bulgaria, the Czech Republic, Finland, France, Germany, Greece, Hungary, Italy, Poland, Romania, Serbia, Slovakia, Slovenia, Spain, Sweden, Turkey, Ukraine, and the United Kingdom.

EURACOAL's mission is to highlight the importance of coal within the EU to security of energy supply, to energy price stability, to economic added value, and to environmental protection. EURACOAL seeks to be an active communicator, with the aim of creating an appropriate framework within which the European coal industry and coal consumers can operate. It is an active leader of many initiatives, presenting the voice of coal in Europe.

COMMERCIALIZATION OF RESEARCH RESULTS

Technology transfer, knowledge-sharing, and commercialization of research results have always belonged to the most important activities of the KOMAG Institute of Mining Technology. Basing on the author's multi-year experience in managing research projects of innovative character, an integrated commercialization model of research and development work results in the domain of mining machines and equipment was elaborated. It should be highlighted that this model presents three stakeholders of the commercialization process, i.e. a research institute, an industrial partner, representing producers of mining machines and equipment, and end-users representing mining plants. The integrated model is illustrated in Figure 11. It shows that a commercialization of research results is a multi-stage, complicated process sensitive to mistakes and very risky. It requires a collaboration of all the three stakeholders starting from the very beginning of the process.

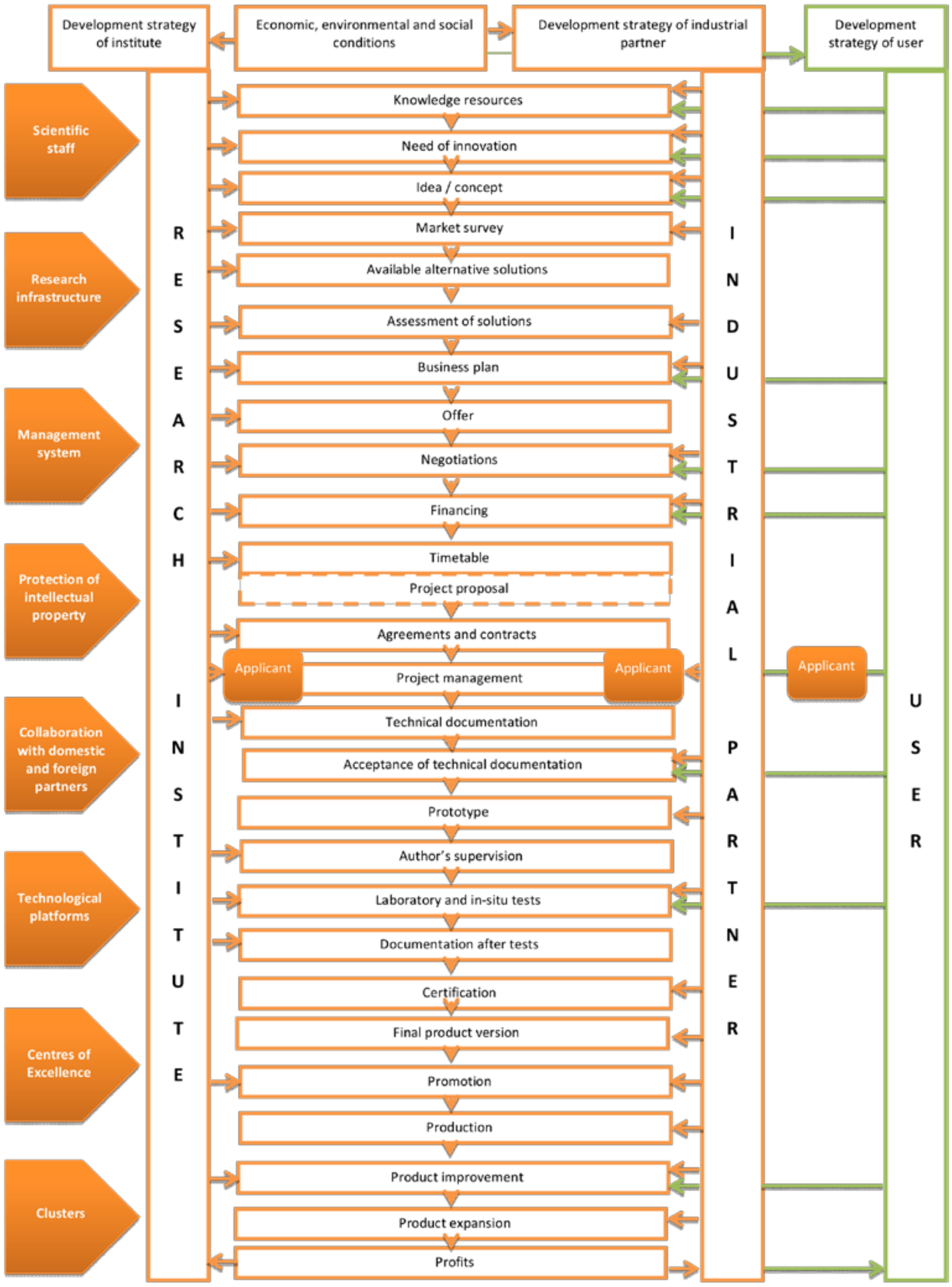

Figure 11 – Integrated commercialization model of research and development work results in the domain of mining machines and equipment (Malec, 2013)

Financing of research projects of innovative character is often a serious barrier which is difficult to overcome. According to the model costs of research projects can be covered by all the three stakeholders who can also apply for state, regional, and local funds intended to be spent on innovations. The integrated model, built on the base of knowledge and experience, resulting from different research and scientific activities presents practical information and can be used as a tool for an efficiency assessment of commercialization processes.

An analysis of the model confirms an impact of economic, environmental and social aspects on a development strategy of a research institute, of an industrial partner, and of a user. It is important to evaluate a market potential of an innovative solution, because a research project of an innovative character should be treated as an investment. A need and demand for an innovation can appear at any of the stakeholders. In the case of mining machines and equipment both researchers and producers as well as users usually participate in tests conducted in laboratory as well as in-situ conditions.

The basic advantage of the integrated model is a possibility of analyzing every single step to check its correctness and of taking immediate measures to rectify the situation if some discrepancies or mistakes are detected. The model is a convenient diagnostic tool enabling a continuous assessment of technology transfer process and of commercialization of research and development work results.

CONCLUSIONS

- KOMAG Institute of Mining Technology plays an important role in the process of increasing safety, reliability and efficiency of mining operations.
- Presented examples of innovative technical solutions of mining machines and equipment, developed at the KOMAG Institute, confirm the correctness of the technology transfer processes enabling a commercialization of research and development projects' results.
- The integrated commercialization model of research and development work results in the domain of mining machines and equipment reflects roles of the three stakeholders of this process, i.e. a research institute, an industrial partner, representing producers of mining machines and equipment, and end-users representing mining plants.
- The integrated model, built on the base of the author's knowledge and experience, resulting from different research and scientific activities, presents practical information and can be used as a tool for an efficiency assessment of commercialization processes.

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INVESTIGATION ON THE PRODUCTIVITY OF A VIBRATORY COLUMN FLOTATION

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INVESTIGATION ON THE PRODUCTIVITY OF A VIBRATORY COLUMN FLOTATION MACHINE

ABSTRACT

The study of the flotation productivity depends both on the collision efficiency of individual gas bubbles and on the size and density of the gas bubbles in the flotation column.

The paper presents the mechanism of air bubbles creation in a vibratory column flotation machine. The use of vibration enables the regulation of the average diameter of gas bubbles depending on the parameters of the vibration process.

The factors which influence the increase in the duration of the bubble in the area of contact with the mineral particles are discussed. Formulas for the parameter effect of vibrations, applicable to the design of flotation machine are derived.

KEYWORDS

Vibratory column flotation machine, Kinetic model of flotation, Collision efficiency, Effect of vibrations

INTRODUCTION

The work efficiency in flotation machines depends on the terms of the dispersion of air. Aerators should provide maximum gas content in optimum average particle size of the bubbles. The requirements for aerators are the following: ensuring appropriate size of the bubbles to provide buoyancy in the flotation complex and minimum macro-circulation of the pulp chamber.

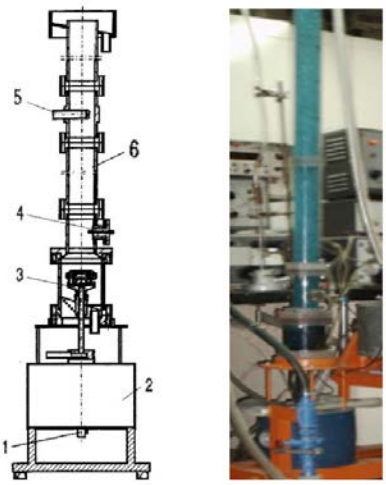

Figure 1 – Principle scheme and photo of laboratory column flotation machine: 1 - sensor; 2 - vibrator; 3 - air disperser; 4 - module for creation of single gas bubbles; 5 - feeding device; 6 – machine chamber

Dedelyanova and Metodiev (2002) presented a laboratory model of vibratory column flotation machine constructed in the Laboratory of “Vibro-acoustic Intensification of the Technological Processes” at the Department of Mineral Technologies of the University of Mining and Geology, “St. Ivan Rilsky”. It is realized on a modular principle, which provides a possibility for determination of the optimum height of the feeding device, infinitely-variable regulation of the pulp level and precise determination of the necessary quantity of extra water for the froth layer irrigation. The vibratory column flotation machine basic elements are: a sensor, a vibrator, an air disperser, a module for creation of single gas bubbles, a feeding device and a machine chamber. The main elements of the vibratory column flotation machine are shown in Figure 1.

In a column apparatus under low turbulence, the probability of attachment and retention of large particles to the bubble is greater as under certain conditions, the use of column flotation machine can be useful for flotation of coarse material (Foot, 1986).

PRODUCTIVITY AS A FUNCTION OF THE EFFECT OF VIBRATION

The productivity of flotation devices is determined by the flotation rate $\frac{d\varepsilon}{dt} = f'(t)$. In the case where the properties of the material in flotation conditions are constants $(\varphi_{att} = const, \quad N = const)$ then equation assumes the shape $\frac{d\varepsilon}{dt} = k(1-\varepsilon)$. According to this equation the flotation velocity is proportional to the mass of flotation material $1-\varepsilon$ and is characterized by a probability k for flotation per one unit time. The solution of equation is $\varepsilon = 1 - e^{-kt}$, which determines the nature of the flotation process reported in time.

To obtain an integrated evaluation of the flotation machine efficiency the concept of collision efficiency $E = \frac{M_0}{M}$ is introduced, where M_0 is the mass of particles that meet with bubbles and M is the mass of all particles which are located at the level of contact at a point in time t. In (Ralston, 1999) the parameter k is called flotation rate constant and is regarded as proportional to the defined collision efficiency E there. In Rubinschayn (1989) the parameter E is called coefficient of grip. The dependence $\frac{\ln(1-\varepsilon_1)}{\ln(1-\varepsilon_2)} = \frac{E_1}{E_2}$ is presented for two flotation processes labeled 1 and 2 for particles of different sizes and ceteris paribus.

According to to Derjaguin and Dukhin (1960) three zones can be defined around the bubble:
- Zone 1 – outermost part – of hydro-static interaction;
- Zone 2 (of attachment) – area of effect of the surface forces;
- Zone 3 – of stability efficiency of the bubble – particle aggregate

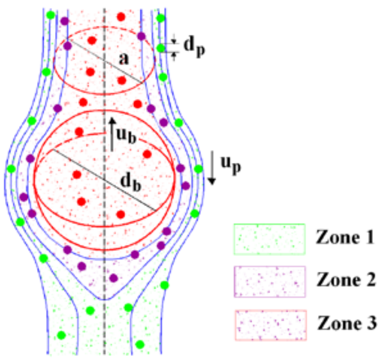

Figure 2 - Scheme of collision of the particles with the bubble

In order to make an analysis of the process of collision we assume that the bubble is spherical and around it we analyse a rotational body with vertical axis passing through the center of the sphere. We denote with d_b the diameter of the bubble, d_p - diameter of the particles and a - diameter of the section of the cylindrical area including zones 1 and 2 prior to the level of collision (Figure 2).

In accordance with the areas of interaction in the aggregate bubble - particle (zone 1 and zone 2) the effectiveness of collision can be represented by $E = E_{il} + E_{vl}$ where E_{il} is efficiency of laminar (gravitation) flow (ideal liquid) and E_{vl} - efficiency of viscous flow (viscous liquid). Taking into account the formulas presented by Rubinschayn (1989) and Bogdanov (1990) with some approximations we can accept the formulas $E_{il} = \frac{G}{1+G}\left(1+3\frac{d_p}{d_b}\right)$ and $E_{vl} = \frac{3}{2}\left(\frac{d_p}{d_b}\right)^2 f$, where $G = \frac{u_p}{u_b}$, $f \in [0.1,1]$.

To analyse the collision process we consider the bubble as a sphere and we consider a rotating body around a vertical axis passing through the center of the sphere. We denote with d_b the aligned bubble diameter, d_p the aligned particle diameter, and a - diameter of the cylindrical areas including zones 1 and 2 prior to the level of collision.

The cross section of the cylindrical region with a diameter a is $S_0 = \pi a^2/4$ and at the level of collision is $S_b = \pi(d_b+d_p)^2/4$. Thus, the effectiveness of conflict is:

$$E = \frac{S_0}{S_b} = \left(\frac{a}{d_b + d_p}\right)^2. \qquad (1)$$

PRODUCTIVITY AS A FUNCTION OF THE SIZE OF GAS BUBBLES

Regardless of the aeration process application, its efficiency is determined by the shared surface at the boundary gas – liquid, for which formation a definite energy has to be used up. Besides, the process application at flotation processing lays down conditions for the obtained bubbles size, which determine their ability to transport a definite number of solid particles to the froth layer.

Experimental determination of the relation between total surface area of contact and the average size of the gas bubbles

Stereology analysis is used to determine the essential parameters of the flotation process . Figure 3 represents a photograph of a part of the column machine camera.

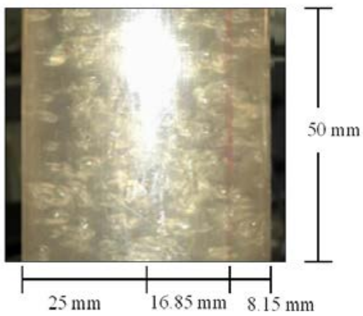

Figure 3 Photograph of a part of the camera in a column machine and a scheme for stereological analysis

A marker with a red line is applied on the photo. The photo has been made at a level *h=110 cm* and at emergence of the bubbles without vibrations. Figure 3 shows also the dimensions necessary for the calculation of the geometric parameters of the cylinder which represents the photographed part of the camera. The marker, presented by a red line, sets the vertical planar section of a cylinder, which has a certain volume - sector of the cylinder. The photographic image helps us to obtain the number of bubbles in the sector and the number of those of them that cross the section marked with the red line.

The section of the cylinder is a rectangle with a height $50\,mm$ and a width that after the corresponding calculation is determined to be $36.94mm$. The surface of the section is $S = 50x36.94 = 1847mm^2 = 18.47cm^2$. The surface pressure of the walls influences some of the bubble during their emergence in the column. This results in a very small number of bubbles in a certain layer of the liquid, circumfluent walls of the column. The volume of the sector separated with the red line is reduced $V_1 = 115.5x50 = 5774mm^3 = 5.774cm^3$. 45 bubbles in the sector limited by the red line are counted in the photographic image in Figure 3. Then $N_V = \dfrac{45}{5.77} = 7.794cm^{-3}$ is the number of bubbles per unit volume. The number of bubbles that are in contact with the section limited by the red line can also be determined from Figure 3. This number is 21. We obtain the number of bubbles per unit area from section - $n_A = \dfrac{21}{18.47} = 1.137cm^{-2}$.

A stereological equation is used for the determination of the average diameter of the bubbles (Saltykov, 1976, Weibel, 1980):

$$d_b = \frac{n_A}{N_V} = 0.146cm = 1.46mm. \tag{2}$$

The total area of contact S_C per unit volume is expressed by a stereological equation

$$S_C = S_0 N_V \tag{3}$$

where $S_0 = \dfrac{\pi d^2}{4}$ is an average value of the section in diameter of the bubble.

The number of bubbles per unit volume N_V is expressed by

$$N_V = \frac{V_V}{<V>} \tag{4}$$

where $<V> = \dfrac{\pi d^3}{6}$ is the average of the bubble and V_V - the amount of gas per unit volume in the area of contact – it depends on the total amount of gas that is fed into the column flotation machine. From equations (3) and (4) for the total area of contact S_C we obtained

$$S_C = S_0 N_V = S_0 \frac{V_V}{<V>} = \frac{\pi d^2}{4} \cdot \frac{V_V}{\dfrac{\pi d^3}{6}} = \frac{3}{2} \cdot \frac{V_V}{d}. \tag{5}$$

From equation (5) can be concluded that the total area of contact S_C depends inversely on the average diameter of the bubble.

A mechanism for creating gas bubbles

The vibratory disperser provides the opportunity to research certain technological parameters: the size change of gas bubbles, the change in the speed of the ascent of gas bubbles and the rate of sedimentation of solids, the impact on the ability for attachment of mineral particles to air bubbles, and the cleaning of the foam layer of carried away rock particles. The module for gas bubbles production provides the opportunity for air micrometric supply through changeable air nozzles with certain initial diameter (Metodiev & Dedelyanova, 2003).

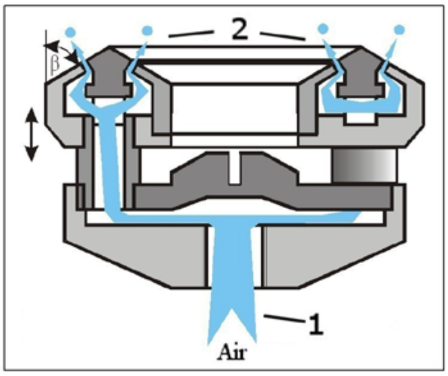

Figure 4 – The vibratory air-dispersing unit

The gas phase dispersion in liquid medium is realized by its transmission through an annular slot in the upper disperser part (Figure 4). The air quantity (Figure 4 – 1) depends on the pressure difference in the supplied air and the water column in the flotation machine.

When the disperser does not vibrate the air is dispersed in big bubbles, which are obtained at random places on the annular slot (Figure 4 – 2) due to its inability to limit bubbles of a certain size because of the slot type. At the disperser vibration, the bubble formation process considerably changes – when the vibrations' frequency and amplitude increase there is a decrease in the obtained bubbles size and an increase in their number. Through experiments and observations it has been ascertained that the bubbles number and respectively – their size depend on the vibrations' velocity $v = Aw\cos wt$.

The growing bubbles from a random annular slot sector are influenced by three forces during the disperser vibration: force of surface tension F_σ, Archimedes force F_A of the liquid removed by the bubble and resistance force F_R caused by the liquid flow circumfluence the bubble and moving on the inclined disperser face. Archimedes force is always pointing up while the resistance force at circumfluence is depending on the phase fluctuations. There is an increase in the Archimedes force with the bubble diameter growth. There is also an increase in the resistance force at circumfluence but it changes in accordance with a sinusoidal law.

In the general case – when the slot plane and the vibrations direction form a certain angle β, the velocity of the bubble circumfluence by the liquid is $v_C = 2\pi A f \cos wt \cdot \sin\beta$. The resistance force at circumfluence is $F_R = \frac{1}{2} c\rho_l \pi^3 d^2 A^2 f^2 \cos^2 wt \cdot \sin^2\beta$.

Control the average diameter of gas bubbles by the parameters of the vibration process

The elementary act of formation of a bubble is accompanied by the action of retaining forces and forces that cause detaching the bubble.

Retaining force is the force of surface tension

$$F_\sigma = \pi\sigma_l\delta\,, \tag{6}$$

where $\delta = 0.6\,mm$ is width of the nozzle of the disperser; $\sigma_l = 7.275\ g \cdot mm^{-1} \cdot s^{-2})$ is surface tension of water.

Forces that detached the bubble:

1. Archimedes force

$$F_A = \frac{\pi d^3}{6} g(\rho_l - \rho_g), \tag{7}$$

where d - diameter of the bubble; $g = 9810\,mm/s^2$ - acceleration of gravity; $\rho_l = 0.001\,g/mm^3$ - water density. The gas density ρ_g is ignored.

2. Resistance force at circumfluence

$$F_R = \frac{1}{2} c \rho_l \pi^3 d^2 A^2 f^2 \cos^2 wt, \tag{8}$$

where $c = 0.09$ - resistance coefficient at circumfluence for sphere (Dedelyanova, 2004); $A[mm]$ - amplitude of the vibrating medium; $f[1/s]$ - frequency of the vibration process; $w = 2\pi f$.

The empirical values of amplitude and frequency are in the ranges $A = 0.5 \div 3.0\,mm$ and $f = 20 \div 50\,Hz$ in the experiments made.

At the moment of detachment the bubble's diameter d is such that the following equation is fulfilled

$$F_\sigma = F_A + F_R \tag{9}$$

In case the disperser operates without vibration effect through (6) and (7) equality (9) takes the form $\sigma_l \delta = \frac{d_0^3}{6} g \rho_l$. We express the diameter of the bubble $d_0^3 = \frac{6\sigma_l \delta}{g\rho_l} = 2.7\,mm$ and we obtain $d_0 = 1.4\,mm$.

The detachment of the bubble in the vibration process is described by (6), (7), (8) and (9) and as a result is obtained

$$\sigma_l \delta = \frac{g\rho_l}{6} d_0^3 + k d_0^2 \cos^2 wt, \tag{10}$$

where $k = \frac{1}{2} c \rho_l \pi^2 A^2 f^2$.

One cycle of vibration at sound frequency is much smaller than the time for the setting and removal of the bubble. The action of the force F_R is considered for a given period of time. Then in equation (10) $\cos^2 wt$ is replaced by the mean value $\frac{2}{\pi}\int_0^{\frac{\pi}{2}} \cos^2 wt dt = \frac{1}{2}$. In this way we get to the diameter of the bubble equation

$$B = C d_0^3 + D d_0^2, \tag{11}$$

where $B = \sigma_l \delta$, $C = \frac{g\rho_l}{6}$ и $D = \frac{1}{2} k$.

We use the average amplitude and frequency $<A>= 1.50$ and $<f>= 32.5$. Then, $B = 4.365$, $C = 1.635$ and $D = 0.527$. Using these values the equation (11) results in $d_0 = 1.28\,mm$. If we take some optimum values for amplitude and frequency $A = 3.00\,mm$ and $f = 50\,Hz$ then we obtain a new value for the parameter $D = 4.996$. The solution of the equation (11) is $d_0 = 0.83\,mm$.

EXPERIMANTAL RESULTS OBTEINED IN A VIBRATORY COLUMN FLOTATION MACHINE

Research has been carried out for the effect of frequency and amplitude of vibrations on the speed of precipitation of mineral grains of different density. It is known that the speed of sedimentation decreases with increasing the intensity of oscillating, however the question for its effect on the diameter and the specific gravity of particles is still open. For that purpose, a series of experiments has been carried out with single mineral grains of a diameter from 0.09 to 0.155 mm and densities of different grains 2.65; 5.1; and 7.6 g/cm^3. The frequency of oscillation changes from 20–50 Hz, and the amplitude from 0.5–3.0 mm. The average results from the experiments are presented in Figure 5.

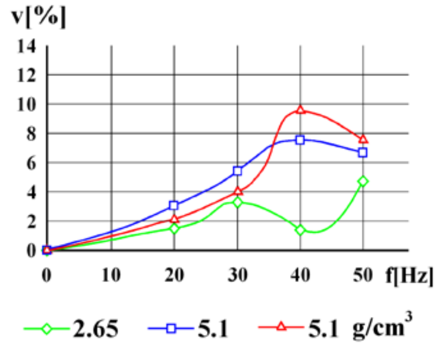

Figure 5 – Reduction of the speed of sedimentation depending on the vibration frequency and density of particles

Laboratory experiments with copper ore (cake taken from a control thickener) have been carried out to determine the influence of vibration frequency and amplitude on the technological results determination. The results of measurements of the amplitude A by height of the water column $h = 150\ cm$ at the various frequencies are presented in Table 1.

Table 1 – Measured amplitudes by height of the water column

Frequency (Hz)	A1 (mm)	A2 (mm)													
	$h(cm)=$	20	30	40	50	60	70	80	90	100	110	120	130	140	150
20	1.0	0.4	0.6	0.5	0.5	0.6	0.6	0.6	0.4	0.5	0.6	0.5	0.5	0.6	0.6
	2.0	0.5	0.5	0.6	0.5	0.7	0.5	0.6	0.5	0.6	0.5	0.5	0.4	0.5	0.5
	3.0	0.5	0.5	0.5	0.5	0.8	0.6	0.7	0.5	0.6	0.5	0.5	0.4	0.6	0.5
25	1.0	0.5	0.5	0.5	0.4	0.6	0.5	0.5	0.5	0.5	0.5	0.6	0.5	0.6	0.5
	2.0	0.6	0.5	0.5	0.5	0.7	0.5	0.5	0.5	0.5	0.5	0.4	0.5	0.7	0.6
	3.0	0.6	0.5	0.5	0.5	0.8	0.7	0.6	0.6	0.5	0.5	0.5	0.6	0.6	0.6
30	1.0	0.4	0.6	0.5	0.4	0.7	0.5	0.5	0.6	0.5	0.5	0.5	0.5	0.5	0.6
	2.0	0.5	0.5	0.6	0.6	0.6	0.6	0.5	0.6	0.6	0.5	0.6	0.6	0.6	0.7
	3.0	0.6	0.6	0.5	0.7	0.7	0.6	0.6	0.6	0.6	0.5	0.4	0.5	0.6	0.8
35	1.0	0.5	0.5	0.6	0.5	0.6	0.6	0.6	0.6	0.5	0.4	0.4	0.5	0.4	0.4
	2.0	0.5	0.5	0.5	0.5	0.7	0.6	0.5	0.6	0.6	0.5	0.6	0.6	0.5	0.6
	3.0	0.4	0.5	0.6	0.5	0.9	0.5	0.6	0.6	0.6	0.6	0.5	0.5	0.6	0.7
40	1.0	0.5	0.5	0.5	0.5	0.7	0.5	0.5	0.4	0.5	0.5	0.4	0.4	0.5	0.4
	2.0	0.6	0.6	0.6	0.6	0.6	0.6	0.5	0.5	0.6	0.4	0.5	0.5	0.6	0.6
	3.0	0.7	0.6	0.6	0.5	0.5	0.5	0.5	0.6	0.5	0.5	0.6	0.4	0.6	0.6
45	1.0	0.6	0.5	0.5	0.4	0.5	0.5	0.6	0.6	0.5	0.4	0.5	0.4	0.6	0.4
	2.0	0.7	0.6	0.6	0.5	0.6	0.6	0.6	0.5	0.6	0.4	0.5	0.4	0.5	0.5
	3.0	1.0	0.7	0.6	0.7	0.5	0.5	0.5	0.5	0.5	0.5	0.4	0.5	0.5	0.6
50	1.0	0.5	0.5	0.5	0.4	0.5	0.5	0.5	0.4	0.4	0.4	0.5	0.4	0.5	0.5
	2.0	0.6	0.6	0.5	0.4	0.6	0.6	0.6	0.5	0.5	0.5	0.5	0.5	0.6	0.5
	3.0	0.9	0.7	0.6	0.5	0.5	0.6	0.6	0.5	0.4	0.5	0.4	0.6	0.5	0.6

Table 1 shows the measured amplitudes in the two cases: A1 – amplitude measured by the sensor on the vibrator; A2 – Amplitude measured by sensors placed on the respective height.

The paper shows only one part of the experiments, based on which they are made appropriate conclusions.

The vibrations frequency and amplitude are determined by the characteristic effect of vibrations Af^2 and their influence on the technological process.

FORMULAS WITH THE EFFECT OF VIBRATIONS

The formulas reviewed here prove the influence of the effect of vibrations on the flotation process in a vibration column flotation machine (Dedelyanova & Dimitrov, 2013). The effect of vibrations Af^2 on the collision efficiency in the flotation process has been studied.

Changing the size of bubble in emergence

In a state without vibration on a certain level h from the surface of the liquid, the pressure in the bubble is $p_1 = p_0 + \rho_l hg$, where p_0 - atmospheric pressure; p - pressure in the bubble; ρ_l - density of liquid and g - acceleration of gravity. Fluctuating pressure p_2 caused by the harmonious impact on the associated mass of liquid is added to this pressure.

As a result we obtain

$$d_b = d_0 \sqrt[3]{\frac{p_0}{p_1 + qAf^2 \sin \omega t}} \tag{12}$$

where d_b is the diameter of the bubble; d_0 - the diameter of the bubble on the surface.

Determination of the contact angle

According to Bogdanov (1990) forces acting on the bubble and the particle attached to it and reported on the vertical axis are two types: forces acting on the particle and forces caused by the attachment of particle to the bubble. The first type of forces are the gravity of particle $P = -\pi d_p{}^3 \rho_p g / 6$ and the ejection Archimedes force $F_A = \pi d_p{}^3 \rho_l g / 6$. The second type of forces are the Laplace force $F_L = -\pi d_p{}^2 \Delta p / 4$ and the capillary force $F_K = \pi d_p \sigma \sin\theta$. The contact angle θ is expressed in the form

$$\sin\theta = \frac{d_p^2 g(\rho_p - \rho_l)}{6\sigma} + \frac{d_p \Delta p}{4\sigma} + \frac{2\pi^2 d_p^2 \rho_p Af^2 \sin\omega t}{3\sigma}. \tag{13}$$

Changing the form constant of bubble

The ratio of the hydrostatic pressure on the contact side of the particle to the Laplace pressure is defined as a constant of the bubble form close to the particle

$$\beta = \frac{\rho_l g d_b}{\Delta p} = \frac{\rho_l g d_b^2}{4\sigma} = \frac{\rho_l g d_0^2 \left(\dfrac{p_0}{p_0 + 4\pi^2 qAf^2 \sin\omega t} \right)^{\frac{2}{3}}}{4\sigma}. \tag{14}$$

CONCLUSIONS

The productivity of the vibratory column flotation machine depends on both the collision efficiency of gas bubbles with particles of floated material and the total surface area of contact. The latter is directly dependent on the mean diameter of the gas bubbles in the area of contact.

The productivity of the vibratory column flotation machine is influenced mainly by the parameter effect of vibrations Af^2. It determines the:

- changes in the shape and size of the bubbles in emergence;
- contact angle θ and
- form constant of bubble β.

The paper presents a method for determining the basic parameters of the flotation process by setting the amplitude A and frequency of vibration f. Analytical models and corresponding formulas for determining the average size of gas bubbles are presented. Formulas for the determination of other parameters of the flotation process expressed by vibrational impact are also presented.

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JOINING HDPE PIPE – A REVIEW OF TRADITIONAL AND EMERGING TECHNOLOGIES

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JOINING HDPE PIPE - A REVIEW OF TRADITIONAL AND EMERGING TECHNOLOGIES

ABSTRACT

This report will compare the HDPE pipe joining methods commonly used in mining and cover recent advancements in mechanical pipe joining methods. HDPE pipe is beneficial for its flexibility, durability and abrasion resistance in abrasive mining applications. Such services require frequent maintenance such as elbow replacement and pipe rotation. Each joining method—fusion, flanging and mechanical couplings—offers advantages and drawbacks as pertains to system access and ease and speed of maintenance, two of the primary considerations in selecting a joining method for abrasive services. The following report will detail the mechanics, pros and cons of each HDPE pipe-joining method plus introduce the mechanical joining method.

KEYWORDS

Butt Fusion, Electrofusion, Flanged Connections, Fusion Bonded Epoxy (FBE), High-Density Polyethylene (HDPE), Mechanical Couplings, Slurry, Socket Fusion, Standard Dimension Ratio (SDR)

INTRODUCTION

Whether dealing with construction or maintenance, there are any number of factors that require consideration for the installation and repair of piping systems, but when it comes to the specifics of joining high density polyethylene pipe, one must frequently need to resort to either HDPE welding or electrofusion techniques. There are a number of potential issues that can arise with both HDPE welding and electrofusion couplings. Aside from the fact that it’s extremely labor-intensive work, we also need to bear in mind that staff on the ground need adequate training and equipment to ensure they are properly equipped to undertake the task.

Not only that, butt fusion and electrofusion of HDPE pipe takes time, time which costs a substantial amount of capital in the form of wages, plant hire and associated logistics and lost hours for other tasks. This report will look at some of the ways that mechanical pipe joining systems can deal with industry problems stemming from productivity slowdowns caused by traditional HDPE joining techniques, and how you can improve your piping operations and help to save costs in an adverse economic environment.

HDPE

According to data compiled by the market research company Hanover Research, HDPE pipe is the fastest growing segment of pipe in the world today. Improvements in materials science and increased performance standards have made HDPE suitable for a variety of previously limited, or long-term applications, such as water and waste water. This rapidly growing pipe segment is gaining acceptance in new markets, with projected growth of approximately 13% annually between 2013 and 2016. HDPE presents a cost effective solution for a variety of markets including water and wastewater, industrial, mining, oil & gas, and power. Applications include potable water, wastewater, slurry, gas, and oil.

HDPE pipe is flexible, very durable and resistant to breakage due to freezing. The lightweight and longer lengths available lead to cost savings in labor and equipment when installing. It also provides flexibility for going over terrain and requires fewer fittings as compared to other pipe material choices.

Figure 1: HDPE pipe

Because it is smoother than cast iron, steel, ductile iron, or concrete, HDPE has excellent flow characteristics. It is resistant to many chemicals and corrosion. HDPE pipe does not rust or corrode due to internal media or external environments (i.e. being buried underground), and has a long service life.

Limitations of HDPE pipe include a lower-rated working pressure than metallic pipe. It can be easily defaced or scarred if used harshly. HDPE pipe also has a high coefficient of expansion/contraction in temperature swings as compared to metallic pipe and a limited temperature range of -40°F to +140°F / -40°C to +60°C. Depending upon the application, the lack of rigidity can be viewed as either a limitation or benefit.

Table 1 – Benefits and limitations of HDPE pipe

HDPE BENEFITS	HDPE LIMITATIONS
Low Cost Of Pipe Purchase	Significant Expansion/Contraction
UV Resistant	Relatively Low Pressure
Flexible	Inability To Connect To Other Material
No Traditional Corrosion Concerns	Easily Defaced Or Scarred

HDPE Pipe Grading and Markings

The grade of HDPE pipe depends upon the resin used to manufacture the pipe and the hydrostatic design stress (HDS) of the pipe. Pipe made from PE4710 materials is the highest grade of HDPE. This high performance pipe, introduced in Europe in 1988, has become more common in North America over the past decade.

In North America, HDPE is typically PE4710 or PE3408. In Europe, Asia, Africa and Australia, HDPE is typically PE100 or PE80. Central and South America will see a mix of both grades and pipe sizes (IPS and ISO).

HDPE pipe is marked for easy identification. The following will be printed on surface of the pipe continually spaced at intervals not exceeding five feet:

- Name and/or trademark of the pipe manufacturer
- Nominal pipe size
- Dimension ratio
- The letters PE followed by the polyethylene grade in accordance with ASTM
- D1248 followed by the hydrostatic design basis in 160s of psi, e.g., PE3408
- Manufacturing standard reference, e.g., ASTM F714 or D-3035, as required
- A production code from which the date and place of manufacture can be determined

Typically, HDPE is also color coded for intended use either with a continuous color stripe, or as solid color pipe.

- Blue – Water distribution
- Green - Sewer
- Purple - Reclaimed water
- Red- Fire Main
- Yellow – Gas

Figure 2: Example of coded HDPE pipe for potable water

THE CURRENT MARKETPLACE

HDPE pipe can be joined using either fusion or with mechanical fittings. The following are traditional joining methods for HDPE pipe:

- Butt Fusion

- Flanging
- Electrofusion
- Mechanical Couplings

Figure 3: Example of butt fusion

Figure 4: Example of electrofusion

Figure 5: Example of flanged HDPE

Butt fusion makes up approximately 98% of the market for joining HDPE in North America. In Europe, electrofusion is the primary method used to join HDPE; however, butt fusion is beginning to gain traction.

Whether dealing with construction or maintenance, there are any number of factors that require consideration for the installation and repair of piping systems, but when it comes to the specifics of joining high density polyethylene pipe, we frequently need to resort to either HDPE welding or electrofusion techniques.

Butt fusion is an involved process. To begin, one needs to transport the fusing tool to the jobsite and set up the tool and auxiliary equipment. After loading the pipe into the tool, the installers must face, clean, and fuse the pipe. The pipe ends are heated according to the recommended guidelines, then pushed together to create the joint. After the joint cools, the installers pull the pipe into place.

The butt fusion process requires special equipment and tools as well as appropriate training to learn how to operate them. These special tools must be rented or purchased and are a big expenditure. These tools also have large power needs. Power generators are required and add additional cost and safety concerns to a project.

The electrofusion process is even more involved. Some electrofusion manufacturers have specific dimensional requirements for pipe that is to be joined using electrofusion couplers. The process requires the installer to inspect the pipe to ensure the ovality of the pipe does not exceed 2%. If it does, the pipe must be re-rounded. After the pipe meets the dimensional specifications, the installer must scrape the outside of the pipe surface to remove oxidation and other contaminants. The scraping requires a specialized tool and takes additional preparation time. Only after the scraping process is the coupling stabbed on to the pipe.

Fusing pipe requires both heating and cooling times to complete the fusion process. HDPE pipes cannot be joined in inclement weather without some type of tent or protection from the elements. In very cold and windy weather, heating times increase significantly. In very hot weather, the cooling time for the pipe becomes extremely long.

HDPE pipes are also very difficult to fuse in a vertical orientation. Pipes must first be fused, and then installed in place. A very small amount of HDPE pipe is mechanically joined.

Figure 6: Fusing with an umbrella.

MECHANICAL PIPE JOINING

There are a number of potential issues that can arise with both HDPE welding and electrofusion couplings. Aside from the fact that it's extremely labor-intensive work, we also need to bear in mind that staff on the ground need adequate training and equipment to ensure they are properly equipped to undertake the task. Not only that, but poly welding and electrofusion takes time, time which costs a substantial amount of capital in the form of wages, plant hire and associated logistics and lost hours for other tasks.

For over 80 years the method of mechanically joining pipe has been utilized for its simplified design, installation and operation of piping systems. The combination of a fast installation[1], integrity of design and structural reliability; coupled with a significantly decreased disassembly time, allows for the facilitation of rapid pipe rotation reducing downtime, increasing plant productivity and reducing costs.

When comparing mechanical pipe joining to poly welding and electrofusion, the reduced assembly/disassembly time of mechanical pipe joining system is a major advantage. With the elimination of hot works for the installation or expansion of piping systems, workplace safety is also significantly increased; reducing risk and increasing plant productivity.[2]

Figure 7: Mechanical Joining of HDPE Pipe

HDPE welding and electrofusion have predominately been used in the past for their functionality & longevity in the installation of piping systems throughout a myriad of industry applications. As we move through the 21st century, environmental impact within projects and plant operations is of major concern for any organization. With mechanical joining systems the whole process is streamlined to a simple system, requiring no use of specific-use equipment for welding pipework or butt fusion welding machines; ensuring the impact to the environment is significantly reduced.[3]

A Product, Not a Process

As previously stated the standout between HDPE welding and mechanically joining piping systems is the simplicity and the time efficiency associated with mechanical joined piping systems. The removal of the welding process in the installation of mechanical joined piping systems requires no specialized certifications or training; thus the processes are easily taught and transferred.[4]

By utilizing the benefits found in a mechanical pipe joining system you are not improving process, you are ensuring productivity. You are ensuring the means to save huge amounts of time, which means you can tender a more competitive quote for whatever contract you are trying to win.

Mechanically-joined piping systems are designed and engineered to offer a direct replacement to traditional HDPE and electrofusion joining systems. The joint will become the strongest part of the line and pressure ratings are designed to exceed those of the pipe upon which they are installed.

Adverse Environments

Piping work, whether in the installation/construction phase or in maintenance, is rugged and dirty. This means that there are significant difficulties for HDPE welding, including ensuring that the pipe ends and fittings are perfectly clean to ensure there is no failure in achieving a perfect weld. This can be especially difficult in maintenance systems where piping has contained viscous or brackish liquids which are difficult to clean up.

By employing mechanical joining systems, cleaning is not as much of an issue, as the piping system is joined by mechanical pressure rather than a chemical bond. There is no concern with contaminant interfering with the weld, which means another productivity-based time saving which avoids the downtime of having to do rework on failed welds.

Figure 8: Mechanical pipe coupling for HDPE

Key Features

A good mechanical joining system incorporates three simple components: The housing, gaskets, and bolts.

Housing:

The system housing features a durable, ductile iron body for rugged service conditions. Integral to the system is gripping teeth which provide direct connection to the pipe. Dipped enamel finish is a standard on all housings; however specialty coatings are optionally available to protect housings from external corrosion in aggressive environments. The ductile iron body conforms to ASTM A-536.

The housing is designed and shaped to permit the dragging of joined pipe strings, as is a common practice across a variety of installation types.

Gasket:

An elastomer gasket with triple seal design provides a leak free seal for a wide variety of services. EPDM (Ethylene Propylene Diene Monomer) gaskets are commonly specified for potable water[5], is ideal for water services, depending on temperature requirements. Nitrile gaskets are commonly specified for petroleum services or air with oil vapors.[6]

Hardware/Bolting:

Zinc plated carbon steel bolts and nuts or fluoropolymer coatings are commonly supplied as standard with HDPE couplings. Mechanical couplings for HDPE can be supplied in two or four bolt configurations depending on the diameter of the pipe that they will joint. The use of mechanical couplings for joining HDPE pipe speeds installation versus heat fusion butt welding or the multiple bolts required for flanged connections. All bolts have a minimum tensile strength of 110,000 psi| 7584 bar. [7]

CONCLUSION

There are many methods of joining HDPE pipe available in the marketplace such as butt fusion, electrofusion and mechanical joining. Mechanical pipe joining represents one of the most efficient emerging technologies in this field and presents an alternative to fusing while still offering all of the following additional benefits:

- Proven reliability
- Easy and fast installation with two or four bolts/nuts; reduced labor hours
- Minimized rework; allows quick disassembly/reassembly of joints
- Ease of maintenance; only two or four bolts/nuts to access equipment and valves
- Consistent installation

By replacing fusing with this new solution, the customer can reduce costs (there is no need for fusion equipment, specialty tools, or generators), compress the project schedule and minimize safety risks. This technology simplifies vertical installations and is ideal for tight spaces. The design also allows for weather independent installation. The couplings are not affected by the temperature of the pipe and can be installed in hot or cold weather, rain or shine.

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ABSTRACT

Starches are widely used as depressant in froth flotation operations in Brazil due to their efficiency, increasing the selectivity in the inverse flotation of quartz depressing iron ore. Starches market have been growing and improving in recent years, leading to better products attending the requirements of mineral industry. The major source of starch used for iron ore is the cornstarch, which need to be gelatinized with sodium hydroxide (NaOH) prior its use. This stage has a direct impact on industrials costs, once the lowest consumption of NaOH in gelatinization provides better control of the pH in the froth flotation and reduce the amount of electrolytes present in the pulp. In order to evaluate the gelatinization degree of different starches and flour were subjected to the addiction of NaOH and temperature variation experiments. Samples of starch (corn, cassava, HIPIX 100, HIPIX 101 and HIPIX 102 commercialized by Ingredion) and flour (cassava and potato) were tested. The starch samples were characterized through Scanning Electronic Microscopy and the amylose content were determined through spectrometry, swelling and solubility tests. The gelatinization were carried out through titration with NaOH, keeping the solution temperature constant at 40 °C. At the end of the tests, the optimal amount of NaOH consumed to gelatinize the starch or flour from different botanical sources was established and a correlation between the content of amylopectin in the starch and the starch/NaOH ratio needed for its gelatinization.

KEYWORDS

Froth flotation, Gelatinization, Sodium Hydroxide, Starches, Flours

INTRODUCTION

The froth flotation routes of iron can be classified in two groups, direct or inverse flotation. Another classification, regarding the type of collector used divides the process in four groups, cationic or anionic iron oxide flotation, and cationic or anionic quartz froth flotation. In any froth flotation process the hydrophobic particles are carried out from pulp to froth zone by means of true flotation, but undesirable gangue minerals also report to the concentrate through hydraulic entrainment and entrapment rather than true flotation, (Clemmer, 1947).

According to Brandão (1994) now a day the inverse cationic route for iron ore flotation is, by far, the most widely adopted method in Brazil. Amines and diamines are used as collectors and starch (or amylum), after gelatinization with sodium hydroxide, as depressants. Frothers are not used because the collector act as one, (Fillipov, 2014).

Araujo at al. (2005) stated that cornstarch is the default depressant for iron ore since 1978. Modified cornstarches are composed by amylopectin (70-80%) and amylose (20-30%) without impurities such as fibers, mineral matter, oils and proteins normally present in conventional starches. Zein, a class of prolamine protein present in corn, presents a depressant action towards hematite. Despite practical industrial evidence that both types of starch (modified or conventional) yielded similar performance, suppliers of conventional starch claim that proteins content in starch might be harmful to flotation results. Experimental results from microflotation tests in modified Hallimond tube showed that zein has depressant properties for hematite as efficient as amylopectin and conventional cornstarch (Peres & Correa, 1996).

Turrer and Peres (2010) point out that oil (triglycerides) present in conventional starch act as an antifoam agent spoiling the flotation process if its content exceeds 1.8%. The starch's depressant action is due to the coating of a natural low energy hydrophobic mineral surface with a hydrophilic film to prevent the attachment of air bubbles.

Liu at al. (2000) presented that the starch simplified formula is $(C_6H_{10}O_5)_n$, where *n* represents aldohexose units (a monosaccharide). The polymerization index n, and consequently the starch molecular weight, vary in a wide range as well as the ratio between the larger and nonlinear amylopectin macromolecules and the smaller and linear amylose. In the starch polymer, only three hydroxyl groups of the cyclic glucose units are free and may rotate to one side of the molecule ring, making that side more hydrophilic. The opposite side is consequently slightly hydrophobic due to the exposed –CH groups. In fact, in aqueous solutions amylose forms a helix with six glucose monomers per turn. The interior of the helix is hydrophobic, whereas the outer shell is hydrophilic.

Amylose and amylopectin molecules are connected to each other via hydrogen bonds in the starch molecules, forming granules ranging from 3 to 100 µm length that are insoluble in cold water. The amylopectin is a branched molecule consisting of several thousand of cross-linked short amylose chains, with normal chain of glucose units joined by α -1,4-linkages, side chains joined to the main chain by α -1,6-glucosidic bonds and molecular weight varying from 10 to 100 times the amylose weight (Peres & Correa, 1996). Ibrahim and Abdel-Khalek (2000) showed that the side chains could have other side chains (or branches), resulting in a branched polymer molecule.

According to Pinto et al. (1992), amylopectin is also easily dispersed in water and shows a lower tendency to form gel and to retrogradation, which is a reaction that takes place when the amylose and amylopectin chains in gelatinized starch realign themselves as the gelatinized starch cools down. In other hand, amylose is a linear, flexible chained molecule displaying random coil behavior in aqueous alkali solution (Weissenborn, 1996). According to the author, amylose consists of long chains of D-glucose units joined by α-1,4-glucosidic bonds. The average molecular weight of amylose is $0.31x10^6$ and for amylopectin is $1,500x10^6$ for cornstarch, by measured by light scattering.

The amylopectin component of starch takes part in flotation and flocculation, but the amylose are unable to react with any mineral surface. Kar et al. (2013) indicate that starch adsorption onto hematite surface are due to the availability of higher concentrations of hydroxylated sites in the metal.

Peres and Correa (1996) showed the importance of the amylose/amylopectin ratio in starch during hematite depression and pointed out that amylopectin reduces the hematite froth flotation more profoundly than amylose when a primary ether amine is used as a collector. This ratio was measured through potentiometric iodine titrations and spectrophotometric analysis by Pinto et al. (1992) and is different for starch-containing vegetables, or even for distinct varieties of the same vegetable. Nevertheless, better results were achieved by Peres and Correa (1996) when using starch with a 75/25% ratio instead of pure amylopectin.

According to the same authors, all types of large molecular weight non-modified starches must be put into solution in a process known as gelatinization prior to its use as a depressant. The process consists in heating a starch suspension in water above 56 °C weakening the intergranular hydrogen bonds and causing swelling of the granules, loss of birefringence, increase of the clarity and viscosity of the solution.

The gelatinization should not use grains coarser than 1 mm to avoid difficulties in solubilization and can be performed by means of warm water or by addition of NaOH (called alkali gelatinization) at room temperature, with the latter being the method adopted in the mineral industry. However, higher NaOH concentrations induces two changes that have conflict effects on the settling behavior of the hematite. On one hand, it generates acidic groups on the starch, which may promote starch adsorption on the hematite and improve flocculation and settling (referred as bridging). On the other hand, higher concentrations could lead to starch chain breakdown, lowering its flocculation power (Wootton & Ho, 1989). The alkali gelatinization involves a disruption of the starch granule integrity and a loss in its crystallinity. Drying may result in the partial recrystallization, but the original integrity of the starch granule is not regained TANG & LIU, 2012.

According Broome et al. (1951) the effectiveness of the alkali gelatinization is strongly affected by the starch/NaOH ratio used in the gelatinization and the dissolution technique, according Iwasaki and Lai (1965). Both Turrer and Peres (2010) and Leal et al. (1993) agree that the typical ratio is 5/1. Some studies, such as Fillipov et al. (2013) and Iwasaki et al. (1969), suggest that the starch solution was designed to yield a maximum concentration of 0.1% w/w and must be prepared daily to avoid the retrogradation. However, this is not well stablished and Guimarães et al. (2005) disagree using concentrations up to 3%.

Pinto et al. (1992) showed that the temperature needed for gelatinization decreases with increasing of the amylopectin content. Retrogradation occurs spontaneously when starch solutions are stored at low temperatures at neutral pH. Amylose's retrogradation may occur within a period of four to five hours after the gelatinization, while the amylopectin retrogrades only 10% within 100 days.

The primary adsorption mechanisms of starch on hematite were proposed as non-selective hydrogen bonding and electrostatic forces, mainly because of the presence of a large number of hydroxyl groups in starch molecules and on hematite surface. As confirmed by Cooke et al. (1952), starch adsorbs more on hematite surface than on quartz. The adsorption density of starch on quartz surface is approximately 10 times less than that on hematite (Mikhailova, 1972).

Peres and Correa (1996) showed that at pH above eight the starch adsorption is almost nonexistent on non-activated quartz surface, while it is noticeable on hematite surface. The reasons for this selectivity are a better ability of hematite surface to form hydrogen bonds with the depressant and the fact that quartz surface is more negative than hematite, being the macromolecules slightly negative because of OH- adsorption. By contrast, Fillipov et al. (2013) proposed that starch adsorption on hematite increases as the pH value decrease.

Other vegetable species, such as cassava, potato, wheat, rice, arrowroot, can produce starch with potential to be used in flotation. The most attractive among them, considering the cost of production, is cassava, which grows widely in warm weather countries, with no need of fertilizers or soil correction. For Turrer and Peres (2010), the major obstacle to its use is the absence of major suppliers.

The starch fraction content (amylopectin + amylose) extracted from cassava is higher than in corn because proteins and oil contents are lower in cassava, which prevents the risk of froth suppression. According to Leal et al. (1993) cassava starch shows higher viscosities than cornstarch, an indication of higher molecular weight, which can lead to more effective depressant action. Liu et al. (2000) showed that potato flour has been used industrially in Europe, but there are no records of its use in the mining industry so far, mainly due the fact that potato degrades much faster than corn and have a high price.

METHODOLOGY

Samples of starch (corn and cassava) and flour (cassava and potato), donated by Cargill, and cornstarch with commercial name HIPIX 100, HIPIX 101 and HIPIX 102, donated by Ingredion, were used in the tests. The first stage aimed to characterize the starches in order to better understand their gelatinization. To determine the amylose content a method proposed by the American Association of Cereal Chemists (AACC) number 1995 was used, which is a simple colorimetric procedure. Samples of starch were added to a solution of 1 mL of acetic acid at 1 mol.L^{-1} and 2 mL of solution of iodine-potassium iodide, which reacts with starch to form a blue colored complex. This complex is developed due to the imprisonment of iodine inside the chain of amylose. The solution was then stored for 30 minutes in a dark room and then using a spectrophotometer Biospectro, model SP220 read the absorbance at 620 nm. The content of amylose was calculated using the absorbance values read and a calibration curve made with pure amylose (supplied by Sigma-Aldrich) in the range of 0.004-0.024 mg.mL^{-1} (for this curve the obtained fit was r^2=0.998). The results were expressed in mg.mL^{-1} and performed in triplicate.

Starch swelling power and solubility were also measured to calculate the amylose content in the starch samples. When starch molecules are heated in water, the crystalline structure is broken and the water molecules form hydrogen bonds between amylose and amylopectin, exposing its hydroxyl groups. According to Singh et al. (2003), this causes an increase in the granule size (a swelling) and in its solubility. This swelling power and solubility varies according to the starch source, providing evidence of the interaction between the starch chains within the amorphous and crystalline domains. For Denardin and Silva (2009), the extension of these interactions is influenced by the ratio amylose/amylopectin due to molecular features distributed, molecular weight, degree, length of branches. The swelling power was obtained by the relation between the final mass swelled and the starch initial mass. The starch solubility was calculated by the relation between the soluble mass and initial amount of starch (expressed in percentage).

The second stage of methodology aimed to standardizing the ideal starch/NaOH ratio, avoiding the excessive use of NaOH. Starch and flour samples 20 g were placed in a 600 mL beaker on top of a heating plate Ika C MAG HS 4. Then 200 mL of distilled water at 40 °C were added to the sample and the solution was kept under agitation at 1,200 RPM using a mechanical stirrer Fisatom 712 to promote a complete dilution of the starch sample. A pH meter Hanna Instruments HI2221 was installed in order to read the solution pH and temperature. After the complete dilution of the starch sample, the initial pH value was noted and the titration started. An aliquot of 1 mL of 10% NaOH solution was added every 2 minutes to the starch solution. Before a new addition of the NaOH solution, the pH was noted. The process was repeated until the point that the starch solution presented itself viscous and more transparent (also known as the turning point of the gelatinization). Six steps of the process can be seen in figure 1. All titration process was performed in triplicate keeping the temperature constant at 40 °C.

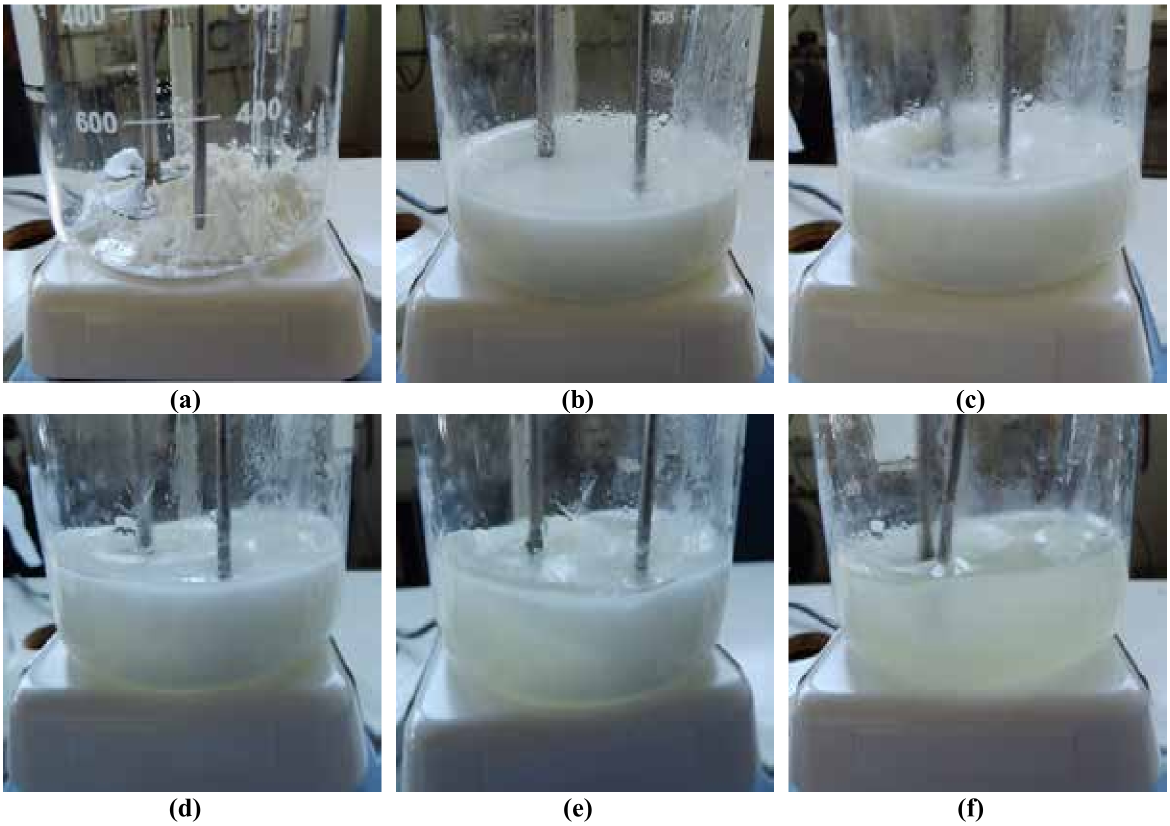

Figure 1 – Six steps in the starches and flours alkaline gelatinization process. (a) starch sample, (b) starch solution after complete solubilization, (c), (d) and (e) steps in the titration process and (f) complete gelatinization of the sample

RESULTS AND DISCUSSION

Figure 2 shows the amylase content measured by the of AACC 1995 method. The amylopectin content was evaluated by exclusion. The values obtained are similar to those available in the literature. The modified cornstarch HIPIX 101, as expected, has approximately 82% of amylopectin, value much more significant than natural cornstarch, which has only 28%. However, other modified cornstarches, such as HIPIX 100 HIPIX 102, showed a lower amylopectin content, which can indicated that these starch come from a different corn specie than HIPIX 101.

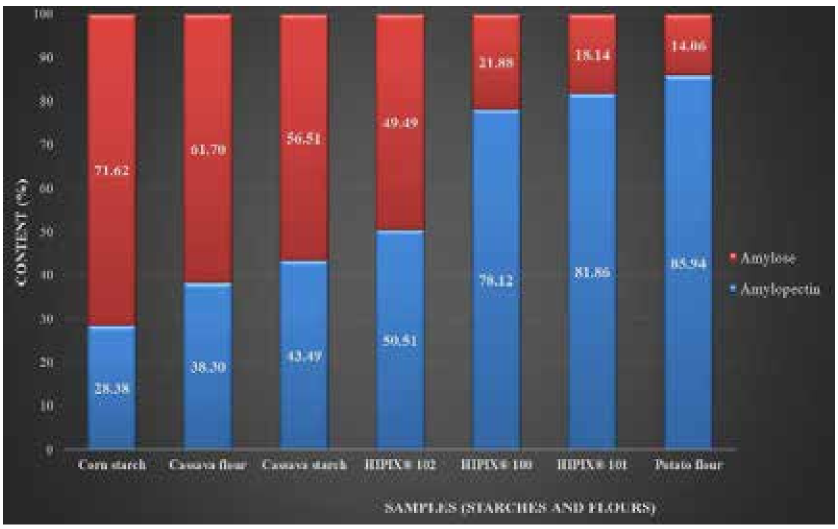

Figure 2 – Amylose and amylopectin contents in starches and flours tested

Figure 3 shows the results of the swelling test for starches and flours tested. It is possible to note that the samples does not follow a specific trend. By contrast, figure 4 shows the contradiction between the real influences of the amylose/amylopectin ratio in the starches and flours solubility. It is possible to notice that cassava flour has the lowest swelling power and has the higher solubility. The potato flour showed similar behavior. However, corn starches (modified or not) showed a different behavior.

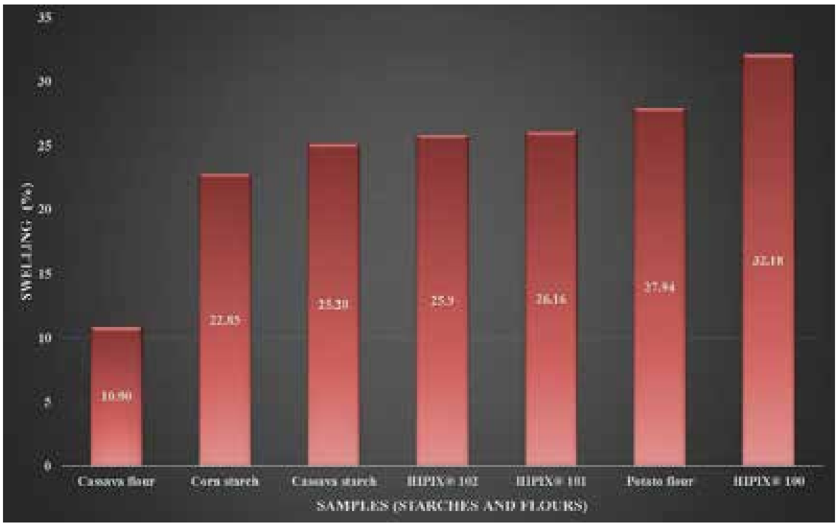

Figure 3 – Swelling test results for starches and flours tested

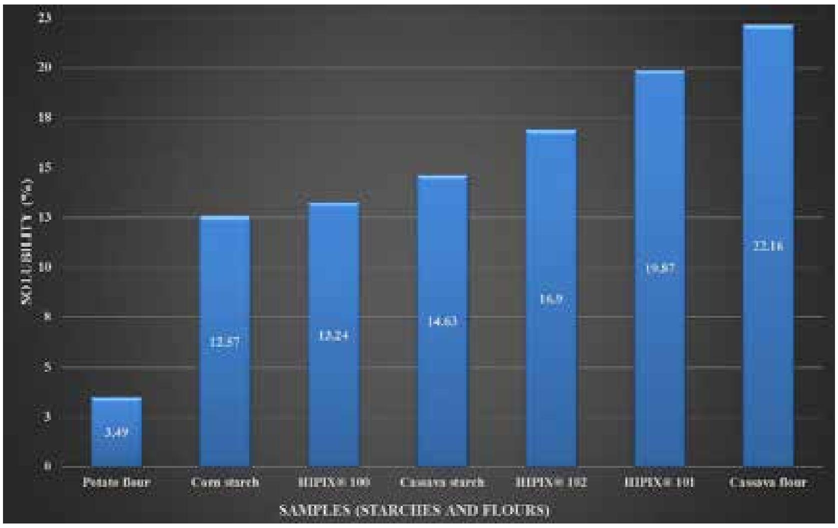

Figure 4 – Solubility test results for starches and flours tested

Pinto et al. (1992) showed that the higher is the percentage of amylopectin the lower is the temperature for the gelatinization. Since the temperature was kept constant at 40 °C, the only variation was the NaOH consumed in the process. The influence of amylopectin ratio in the gelatinization is observed in figure 5, where it is possible to notice that HIPIX 100 achieved the complete gelatinization in the lowest pH (around 11.5) while the others samples did the same at a higher pH (around 12.5).

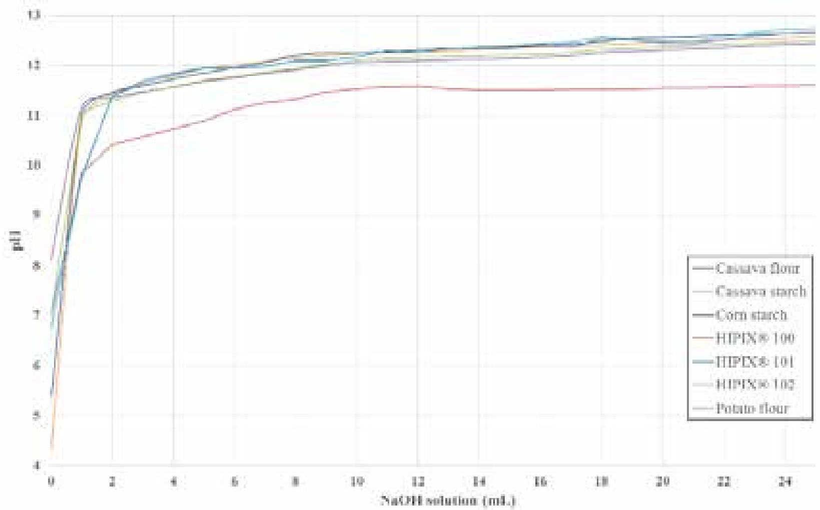

Figure 5 – Variation of the solution pH during the gelatinization tests at 40 °C with the NaOH addition

Table 1 summarizes the found results regarding the turning point (or gelatinization point) for the starches and flours tested. The Amylose / amylopectin ratio measured is presented, as well as the initial and final solution's pH. Cassava flour, which has the higher amylose / amylopectin ratio (1/0.62), consumed the higher volume of NaOH (13.7). By contrast, potato flour has the lower ratio (1/6.11) and consumed the higher volume of NaOH (5.3), as expected. This behavior was not seen with the modified starches (HIPIX 100, 101 and 102).

Table 1 – Gelatinization tests results

Starches and flours	Amylose / amylopectin ratio	Initial pH	Final pH	Average NaOH consumed (mL)
Cassava flour	1/0.62	6.43	12.11	13.7
Cassava starch	1/0.77	4.32	12.28	13.3
Corn starch	1/0.39	4.37	12.36	14.7
HIPIX 102	1/1.02	4.22	12.04	9.3
HIPIX 100	1/3.57	4.11	11.54	10.3
HIPIX 101	1/4.51	5.09	12.12	9.7
Potato flour	1/6.11	6.74	11.96	5.3

The addition of NaOH after the gelatinization of the sample did not increase the solution pH, as would be expected. Instead, a plateau was observed for all samples. Analyze of this result allowed the establishment of the minimum amount of NaOH need to perform the complete gelatinization of the sample as presented in table II. Potato flour showed the higher starch/NaOH ratio (35.7/1). Since this sample has the higher amylopectin content, this result agrees with the expected regarding the temperature need for the gelatinization. The same behavior can be seen for cornstarch, which shows the least starch/NaOH ratio (13.3/1) and amylopectin content.

Table 2 – Optimum starch/NaOH ratio found in gelatinization tests

Potato flour	HIPIX 102	HIPIX 101	HIPIX 100	Cassava flour	Cassava starch	Corn starch
35.7/1	21.4/1	20.7/1	19.3/1	14.6/1	15.0/1	13.3/1

CONCLUSIONS

The titration with NaOH in the complex solution starch/ NaOH presented as an interesting tool for analysis the gelatinization point of starches and flours. The presence of not broken or swollen granules varies with the ratio of starch/NaOH and reaction time. However, the understanding of NaOH consumption in water/starch solutions is critical, once it can have a successful gelatinization with lower amounts of NaOH, as shown in the tests.

The starch acts in the selectivity of the flotation process that is influenced by the state of liberation from the chains of amylose and amylopectin. The found results showed that mineral industry in Brazil are adopting starch/NaOH ratios varying from three to five times higher than the necessary. Another conclusion is that the higher is the percentage of amylopectin the lower is the temperature and the lower starch/NaOH ratio for the gelatinization.

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ONLINE QUALITY CONTROL IN MINING

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ONLINE QUALITY CONTROL IN MINING

ABSTRACT

The recent practices in mining area report to need for mining companies to invest in mineral extraction technologies aimed at improving operating performance, security of work and better use of deposits in its mining and ore processing steps. In the case of iron ore, the high demand in its use as a raw material for the production of consumer goods leads companies to seek to maximize their assets and increase the production volume with the compromise the quality of the final product, according to the market's specifications. This work was developed in the mine Casa de Pedra, located in the city of Congonhas/MG, which belongs to Companhia Siderúrgica Nacional (CSN), being its main relevant parameters analysis aspect of the quality of the ore in determining quality control-related variables, such as size and contents related to iron ore. The objectives of this research were: reduction the variability of the chemical levels in the feed mill and measurement of effectiveness information the template blocks, through comparative analysis of notes lithotypes in field with the global sampled lithotypes (surface map). It was created an online analysis methodology of quality iron ore (chemistry and sizes), directly from mining fronts for the control and meeting customer specifications individually. Thus, one can generate greater predictability and reduction in variability of the ore content information on feeding the concentration plant, thus ensuring better use of the deposit.

KEYWORDS

Quality Control, ROM, Lithological types, Grades, Block Model

INTRODUCTION

The search for competitiveness and competition of an increasingly globalized market and the increasing demands of customers the criteria values such as quality, time, flexibility and price, makes the search for new mining activities of mineral processing important to achieve the strategic objectives of mining companies. It is necessary to develop studies and techniques to enable new mineral reserves from less expressive grades and adapt the mineral beneficiation processes, due to constant risks embedded into mining business, particularly evident from the current crisis. The last two years have highlighted the fall in iron ore prices and consequent reduction of the profit margin of the mining companies.

According to Carvalho (2012), current methods of treatment of iron ores have been through constant adjustments in order to achieve improvements in the performance of merger cases from poorer ores into metal content, with the presence increasingly constant contaminant materials such as silica and alumina. The shortage of iron ore reserves with higher levels, as the reserves of Hematite, also contributes to the search for iron ore exploitation technologies with poorer levels as Itabirite ores and also for more accurate blending.

According to Chanda and Dagdelen (1995), to ensure the uniformity of the supply of the beneficiation process is necessary to mix ores of different grades of various mine mining fronts. A suitable ore blending can broaden the base of the mineral reserve, in that the ore below the cut loom is not considered waste, but can be used if mixed with high-grade material.

Bianchi T. et al. (2009) explore the context of unit operations of open pit mining presents several particularities in relation to desired production and control of quality and costs associated with the availability and utilization of physical resources of mine. All this must be taken into consideration in mining projects, because the entire production chain depends on these operations

According to Bustillo, R.M. et alii (1997) apud Flores e Cabral (2008), one of problems in the mineral sector is the definition of the volume of ore deposit that can be economically viable. The economic reserves evaluation techniques require the construction of a blocks models generated with geological information, normally obtained from surveys in step. Every block of this model are associated information such as content, particle size, density, lithological type, etc. Using this information it is estimated the economic value for each block, which is almost always the benefit or profit expected to get with the extraction and treatment of mineral content present in each block.

The need for better use of the mineral resources, cost control and performance indicators related to the processing plant increased the need to achieve better operating practices of selectivity of mining and improved quality control. The availability of high-precision positioning technology for ore quality control has allowed to increase the selectivity of mining through the use of geological block model for aid to the load equipment operator. The paper consists of creating parameters analysis methodologies relevant to the quality of ore in determining variables related to quality control, as levels and grain size related to iron ore. Existing methodologies currently have heavy reliance on laboratory tests. These analyzes require a certain time to be performed since the sampling, transport to the laboratory and analysis itself. Not always this flow meets the demands of mine planning area because the deposits tend to be increasingly heterogeneous and demands for faster analysis are most needed. The priority targets of the research were to reduce the variability of global chemical levels of the mine block model and samplers of the plant and the control of reconciliation between the volume mined ore and the appointment of the fleet management system. The project was developed in the Casa de Pedra mine, located in the municipality of Congonhas / MG, in partnership with Companhia Siderúrgica Nacional (CSN), which owns mining. Methodologies and processes to be developed have been tested and applied using information obtained directly from the database generated in the mine.

EXPERIMENTAL

The company has a fleet management system that monitors all productive equipment in the mine. Their main function is management of the fleet with the objective to optimize the mining operation and maximize the load and transporting equipment global productivity, thus, reducing operational costs.

The system receive real time information, through wireless network, using high precision GPS and using an embed onboard computer for each equipment.

The telecommunication network by wireless refers to the transmission of data by air passage without the need of cables – whether they are coaxial or optical cables – trough equipment that use radio frequency (communication by radio waves). For the signal distribution in all area for mine, are used mobile repeater antennas that allow the quick distribution of the signal and greater signal coverage mobility in the mine.

High precision GPS system

The high precision GPS (Global Position System) technology has many applications in the mining industry that can help to increase the equipment productivity, the mining selection and reduce the mine operational cost. Further on the possibility of a greater use of the deposit. The mining companies, mainly the open pit mining, offer an ideal environment to use GPS, combine with clean sky with the direct view to the satellites that determine the equipment position.

The precision is determinate by the GPS receptor and can be consider low or high precision. The mean precision of the equipment that have low precision GPS is about 10 meters, while in the equipment that have high precision GPS, the precision reaches less than 10 cm.

Technology refers to products, processes, tools, methods and systems applied or used in the creation of products and services comprise four basic components: equipment; knowledge; Reasons for use; how to use. The last three components can be described as human factors technology.

Many processes or controlled human actions using some form of technology. In this project, the mineral exploration technologies represent autonomous systems that are easily deployed in mining equipment and require operator training or specific skills to use. These systems can demonstrate immediate value added to the investment in terms of change management.

Benefits as an aid in under adverse operations conditions (rain, night time), accurate 3D positioning, 24-hour availability, synchronism in real time and provide services and information to an unlimited number of users, are examples of gains that this technology can bring the mining industry.

Data base

Building a database is the process of storing the data in some way that is controlled by a Database Management System (DBMS). Manipulating a database indicates the use of functions such as query to retrieve specific data, modifying the database to reflect changes in the mini-world (inserts, updates, and removals), and reporting.

• Data: Value of a stored field, the raw material to obtain information;

• Information: Data collected and processed according to the request for consultations and analysis.

Block model

For evaluation criteria of deposit and physical and chemical grades, the block is the smallest unit of selective smaller mining. The choice of block size should be fulfilled according to the type of mined mineral search method used in their estimate dimensions of the equipment used in the mining and height of the mining stands. One of the greatest challenges of geology and mine planning teams is to determine the ore dilution and maximizing the recovery of chemical levels through greater selectivity in mining operations.

ENGINEERING DESIGN

For the initial monitoring of research, an analysis methodology was developed, synchronizing the spatial position of the blocks with the load equipment bucket coordinates issued for high precision.

1. Import Flow of block model files to the dispatch system:

2. Creation of an association of logic between the high precision loadings coordinates with the coordinates of the blocks informed imported model;

3. Process Automation (automatic associations between the coordinates and blocks);

4. The calculation of the grades searches the loading points for each "pass" performed by the load equipment, weighing the grades by mass loading, considering the limits of each wrought / mined block;

5. Creation of a structure (modeling) in the database to receive information from the block model;

Some anomalies related to the blocks that are made by the team of Geology and sent to the short-term mine planning team template files were checked. The main differences were:

• Lack of blocks in mining advances;

• Blocks without grades estimated by granulometry particle size range, only global grades/levels;

• Blocks with overestimated levels/grades;

• Calculation errors when performing the stoichiometric balance.

These differences generate various negotiations to increase the effectiveness and reliability of the block model, such as changing the sampling procedure in ROM stack and product.

Analysis reports have been created in order to accompany such divergences in real time, identifying the grades and granulometry of the whole ore feed the primary crusher and ore stockpiles located in the mine. Figure 1 illustrates the quality monitoring report, which will be updated every 5 minutes:

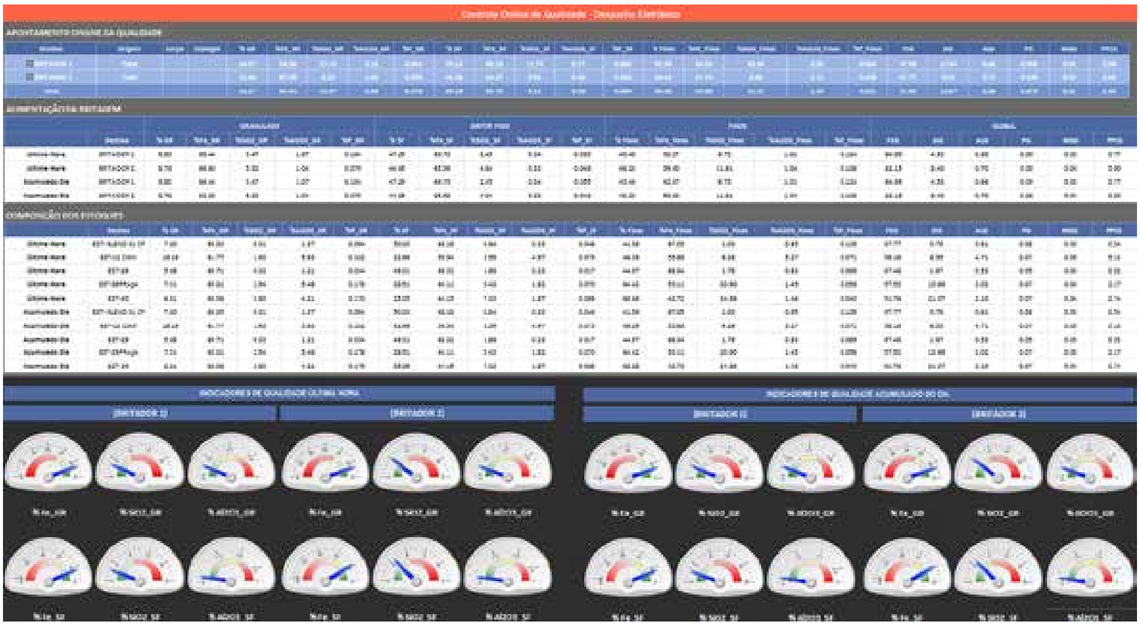

Figure 1 – ROM Online Quality Monitoring

Another created control report was lithotypes comparison indicated by the block model with real assignment in mining excavator through the loading equipment operator together with quality control technicians, as shown in Table 1. From this information is possible to identify the main pointing deviation area and direct the geology team at these locations:

Table 1 - Example of pointing Adherent Report

Pointing Adherent				
Mining areas	**% Adherent Lithotype**	**% Non-Adherent Lithotype**	**% Tonnage unplanned**	**Total Tonnage (t)**
22J_1451	95%	4%	1%	51,089
19I_1451	59%	39%	2%	41,104
12G_1373	20%	46%	34%	38,175
13H_1360	18%	33%	50%	19,104
14G_1464	14%	85%	1%	16,836
60_1113	100%	-	-	2,436
Stocks	-	-	-	44,961
TOTAL	**51.3%**	**34.4%**	**14.3%**	**213,705**

The mining quality information is also compared with the sampling information of laboratory equipment and analysis of the following processes. The monitoring of the ore levels information generated through the sampling carried out each point of the production flow can be seen in Figure 2. The figure shows the simplified flow chart of iron ore processing plant of the Casa de Pedra mine. Featured are the placement of samplers in the processes of the plant, loading and shipment:

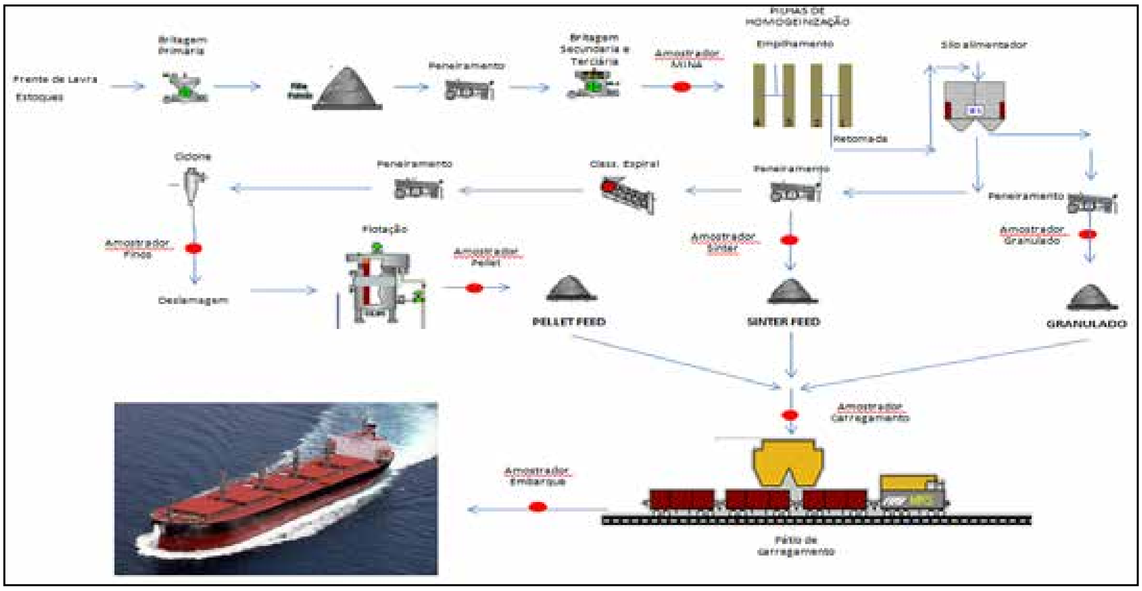

Figure 2 - Simplified flowchart of the plant (Casa de Pedra Mine)

OPERATION RESULTS

The quality control more effective with accurate simulations reduces the variability of the grades of the final products (lump and sinter feed). This enabled a significant reduction in the average grade of the plant feed without leaving the specifications of planned products. It was also possible the use of materials that were previously treated as marginal ore. These improvements have optimized the mine reserves without impacting the final product quality in customer service. It is observed a reduction of 5.41% in the movement of hematite material during 2015. This is equivalent to an approximate reduction in the extraction of 830,000 tons of high-grade materials, ensuring better use of the deposit.

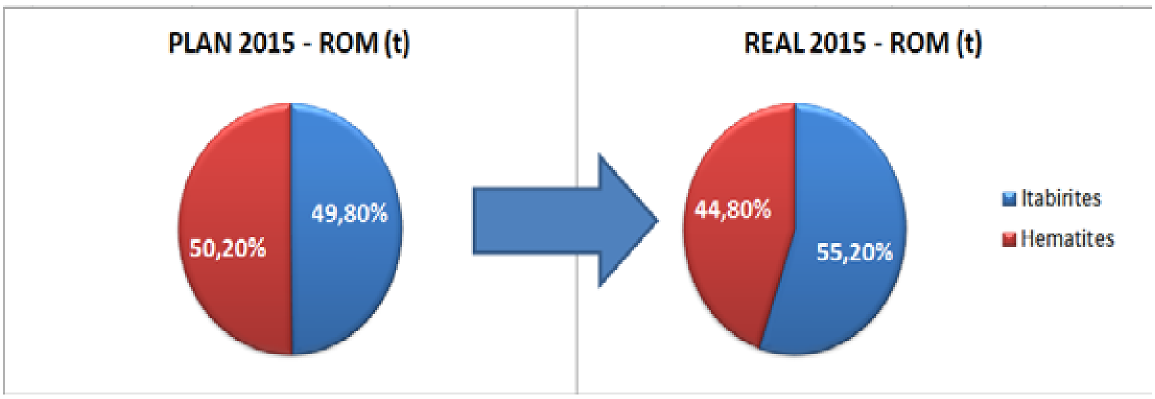

Figure 3 – Comparison of mass movement in the year 2015

The control of the variability of iron grades in the Sinter Feed product in feeding of the mobile plant processing is shown in graphic 1.

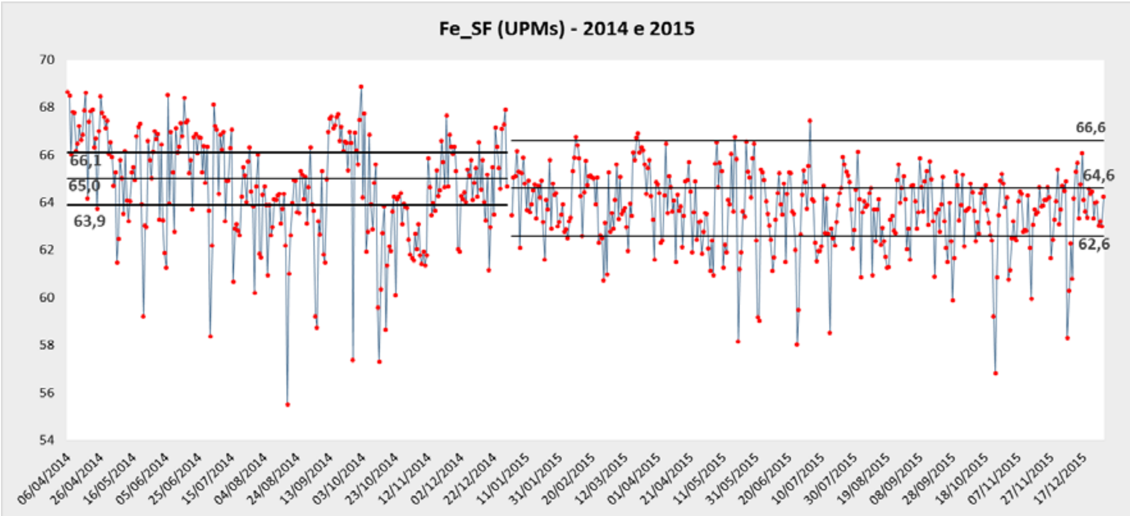

Figure 4 - Variability of Iron Ore grade in the Sinter Feed between the years 2014 e 2015

Table 2 shows an average reduction of 1.03% iron ore grade and reduce the sample standard deviation of 0.68 between the years 2014 and 2015.

Table 2 - Comparison between the iron ore grades with reductions in standard deviation

%Fe_SF UPMs	2014	2015
Mean	64.71	63.68
Std Dev	2.29	1.61
Min	55.5	56.8
Max	68.9	67.43

DISCUSSION

Many companies in the mining sector lose millions of dollars due to the failure to implement appropriate procedures for quality control of its products and the lack of knowledge of reconciliation problems.

Sampling, in mining is an activity of great importance, especially in the short-term planning, where an efficient sampling leads to significant improvements in operating results. This efficiency depends on a reliable reconciliation system, based on the improvement of sampling methods that lead to a proper routine collection and resulting in improved data quality.

CONCLUSION

The objective of this work was the creation of a methodology for analyzing the quality control of extracted ROM in a online manner. Thus it is possible to anticipate decisions in the planning of mining fronts and material changes directed to feeding the primary crusher related blend of ore according to each particle size range analyzed.

It was verified a reduction in the movement of high-grade ore (hematite) of approximately 830,000 tonnes during the year 2015, without impacting the planned specifications physical and chemical of the final product.

It was also observed significant reductions in the variability of chemical concentrations sampled iron, silica and alumina of lump and sinter feed products.

Quality control of ROM stocks was also an improvement generated from this methodology, ensuring reliability in the use thereof.

This tool also enabled greater performance of quality control technicians and dispatch system in shifts. The grades and quality of information they have access, reflect more assertive actions and decisions ROM drive for primary crushing and dry processing plants.

With the use of an automated system of quality control, it was also possible to eliminate the use of manual spreadsheets blending simulation, the real-time control, and with electronically stored data can generate production reports with more reliable data assisting in new decision-making during the shift.

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PROFIT MAXIMIZATION IN THE TRANSITION PROCESS FROM SUR-FACE TO UNDERGROUND MINING METHODS

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PROFIT MAXIMIZATION IN THE TRANSITION PROCESS FROM SURFACE TO UNDERGROUND MINING METHODS

ABSTRACT

The numerous existing mining methods can be classified into two broad classes, surface and underground mining methods. Comparably, open pit mining methods have some advantages compared to underground methods, among which can be mentioned low associated mining operating and development costs, higher ore recovery and production rates, easier grade and production control, safer for workers and the operational flexibility after development/operation starts. Mining by underground methods is generally associated with high operating cost, significant investment in mine development and longer production ramp-up curves. Furthermore, there are deposits that can be mined simultaneously by both methods. Many mines in operation are currently in transition from open pit to underground. The transition to underground mining may be seen as a natural trend in global mining once deposits usually mined by surface methods tends to become unfeasible with the increase on pit depth. This study aims at comparing the existing mine planning routines for the determination of the best transition point between surface and underground mining for a hypothetical copper deposit.

KEYWORDS

Open pit, Underground, Transition, MILP, Strategy, Cut-over

INTRODUCTION

The mining activity consists in exploiting ore reserves from earth’s crust. The mine planning engineers are responsible for the determination of the business scale and the definition of suitable mining methods. The mining methods may be classified into two main classes, the open pit and underground mining methods. Open pit mining (OP) has some advantages over underground (UG): low associated mining operating and development costs, higher ore recovery and production rates, easier grade and production control, safer for workers and the operational flexibility after development/operation starts. Despite some disadvantages associated to underground mining, it still has relevant advantages to the open pit method, such as early access to high grade deeper orebodies, high selectivity, low dilution, lower environmental impact, relatively constant haul cycle times, among others. Bakhtavar et al. (2009) observes that underground mines will often have smaller footprints than equivalent capacity open pit mines.

Currently, some of the largest open pit mines worldwide are reaching their final pit limits. Usually, open pit mining costs steadily increase with vertical advance due to the increasing strip ratio and haul cycle times. Besides, as the open pit operation goes deeper, more geological information is collected from the lower portions of orebody. All these factors may reveal the existence of economic underground ore potential and therefore transition to underground operation may be a convenient possibility. The final pit's depth, or transition depth, must be carefully evaluated in order to provide reliable and optimized results. A trade-off between the overall economic value of the project, ore reserves recovery and assumed geotechnical risks must be assessed in the process of choosing the ideal transition depth. Environmental requirements may also be relevant and contribute to the decision of going underground.

Different approaches can be selected to evaluate the ideal transition depth. Finch and Elkington (2011) explain different approaches and calculate the economic value for the optimized solutions. Some of the earliest studies related in literature about the determination of the optimal solution for an open pit to underground mining transition applied the relationship between mining costs of a tonne of ore mined by underground methods to the waste removal cost necessary to mine the same tonne. In 1982, Nilsson based his approach on the discount effect of the Net Present Value on the cash flow results and later, in 1997, he addressed the discount rate as the most sensitive parameter in the transition process. Since the introduction of the discount rate on the transition optimization study, different algorithms have been proposed. Camus (1992) proposed an algorithm based on optimizing surface mines, taking into account the alternative opportunity cost associated with the underground mining of each block to define the pit limits. Heuristic approaches were proposed by Visser and Ding (2007) and by Bakhtavar and Shahriar (2008). Ben-Awauh, Richter and Elkington (2015) describe a Mixed Integer Linear Programming (MILP) formulation to solve a problem of combining simultaneously open pit and underground methods.

OBJECTIVES

The present study presents different approaches to evaluate the feasibility of combining open pit and underground mining on a hypothetical medium-size copper deposit. Initially, the deposit will be mined by open pit techniques. Transitioning to underground mining is further considered as an alternative as open pit deepens. Simultaneous OP and UG mining is finally considered as well.

The objective of this paper is illustrate through this example the advantages and disadvantages of each approach. This study was carried out using Evaluator, software internally developed by Snowden (Elkington, 2010). This tool incorporates specific algorithms for solving mixed integer linear programming problems. It is well adapted for transition OP/UG studies. Evaluator allows fast configuration of a great number of complex mine scheduling problems, incorporating the operational extraction sequence for each mining method, cut-off grade strategy and economic model, mining and processing capacities, while maximizing the project net present value.

CASE STUDY

The orebody is a medium size copper deposit which dips 70 degrees. Diluted copper grades range from 3% Cu in the upper levels to 6% Cu in the deeper portions following a linear increasing distribution. A vertical section of the orebody geometry and Cu grade distribution are presented in Figure 1. In this exercise, each square in the section was configured with the same tonnage, 500kt. Elevations and horizontal coordinates shown in the vertical axis are not presented at a real scale.

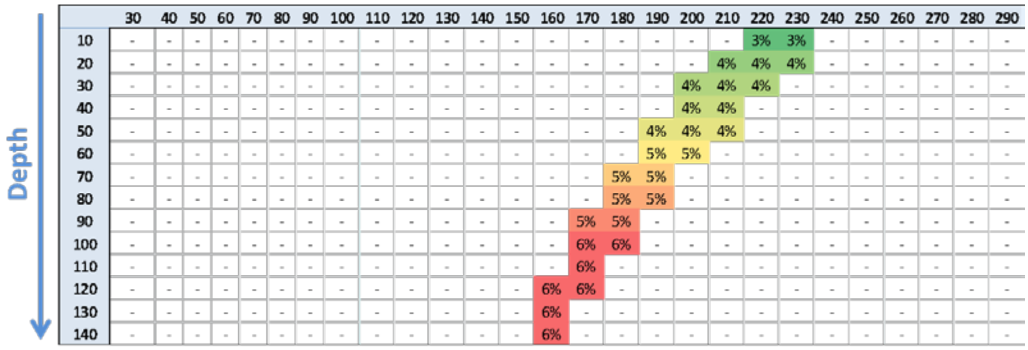

Figure 1 – Schematic vertical section showing the orebody geometry and Cu grade distribution

A financial model was set to estimate the undiscounted value of each block, including commodity price, processing and mining costs (Table 1). An incremental vertical mining cost gradient was included along with development costs.

Table 1 – Financial model

Parameter	Unit	Open pit	Underground
Net commodity price	US$/lb	2.00	2.00
Mining cost	US$/t mined	2.70	32.00
Processing cost	US$/t ROM	35.00	35.00
Fixed costs	US$/t ROM	35.00	35.00

The total reserves are estimated in 14 Mt of ore. No tonnage factor was considered at this stage.

A conventional trucks and shovels system is selected as the open pit mining method. Sub-level stoping techniques are assumed for the underground operation.

The open pit production capacity is estimated in 900 ktpa through the modified Taylor's rule for open pit copper mines (K.R. Long and D.A. Singer, 2001):

$$Production\left(\frac{mt}{day}\right) = 0.0236 * Tonnage^{0.74}$$

The production capacity of the underground mine is estimated in 600 ktpa by a similar formula for underground mining methods (K.R. Long, W.D. Menzie and D.A. Singer, 2000):

$$Production\left(\frac{mt}{day}\right) = 0.0248 * Tonnage^{0.704}$$

As a result, the capacity of the selected underground method indicates that the maximum production rate for underground mining is only 70% of the open pit capacity.

This paper presents three alternative approaches to determine the transition point from open pit to underground.

1. Maximum economic pit analysis

This approach considers two sequential stages: 1) determination of the maximum economic pit; 2) depletion of the remaining reserves using underground mining. This method is based on the fact that operating costs of an open pit operation are generally lower than those associated to underground mining. As a result, mining extraction must commence with an open pit configuration followed by an underground operation.

The first step is the determination of the maximum economic pit beyond which incremental nested pit shells would result in negative marginal net profits if open pit techniques continue being applied. A family of pit shells were generated and further sequenced. No efforts were undertaken to create practical pushbacks given the illustrative nature of this exercise. Nested pit envelopes are shown in Figure 2 (upper left figure). The largest economic pit bottoms out at elevation 100 which results in a discounted present value of US$m 189.3 (upper right figure). Beyond this point, sub-level stoping techniques can be applied to mine the remaining resources. The combination of open pit and underground reserves totalize US$m 206.5 (lower right figure).

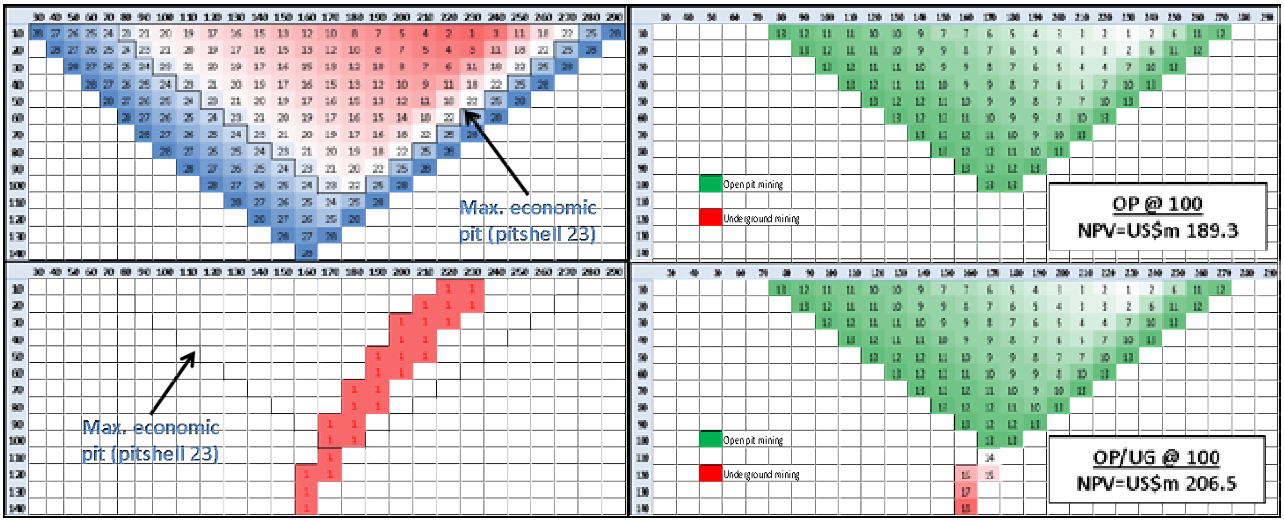

Figure 2 – Nested pit shells against orebody and maximum economic pit analysis

2. Manual heuristic method

This traditional approach considers the creation of a number of transitional scenarios at various elevations. A mine scheduling is completed for each option and NPV calculated. The highest NPV among these scenarios indicates the 'optimal' transition point. This method does not provide necessarily the optimal configuration and it can be regarded as a heuristic procedure.

A number of candidate of transition elevations are chosen, configured and tested. The selected transition elevations are 90, 80, 70 and 60. The mine scheduling results are shown in Figure 3 and the corresponding NPV's are presented in Table 2. NPV is maximised at the 80 elevation with US$m 216.7. The production scheduling of this scenario is shown in Figure 4.

Table 2 – Comparison between the 'maximum economic pit' method and heuristic analysis

Heuristic analysis	NPV (US$m)
OP+UG: Transition at 60 elevation	208.4
OP+UG: Transition at 70 elevation	215.3
OP+UG: Transition at 80 elevation	216.7
OP+UG: Transition at 90 elevation	213.5
Maximum economic pit analysis	**NPV (US$m)**
Maximum OP	189.3
Maximum OP + UG at 100 elevation	206.5

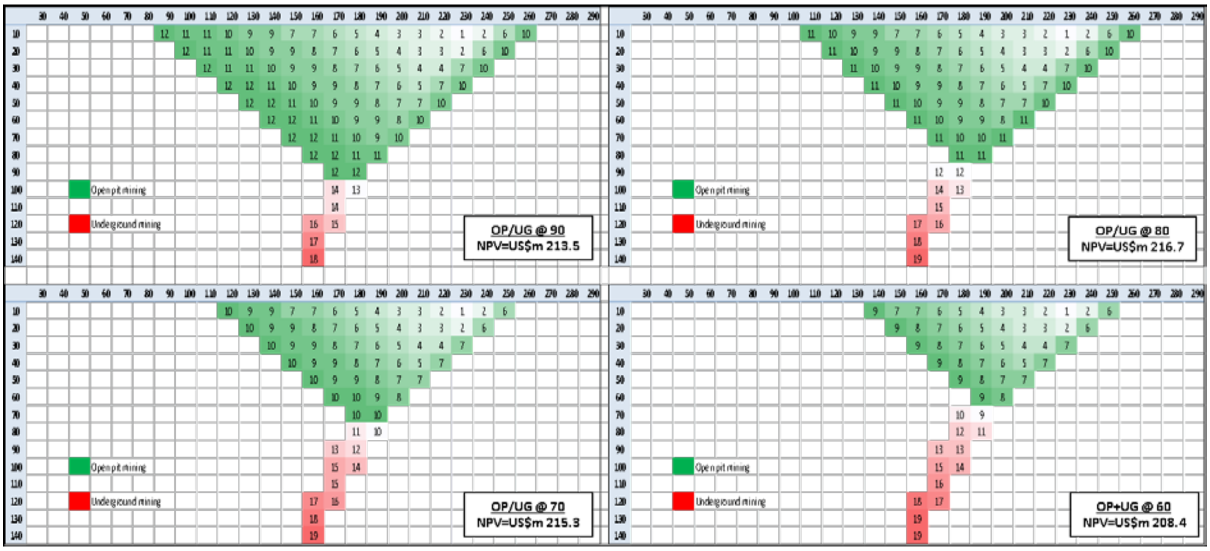

Figure 3 – OP/UG transition scenarios, incluiding the net present value

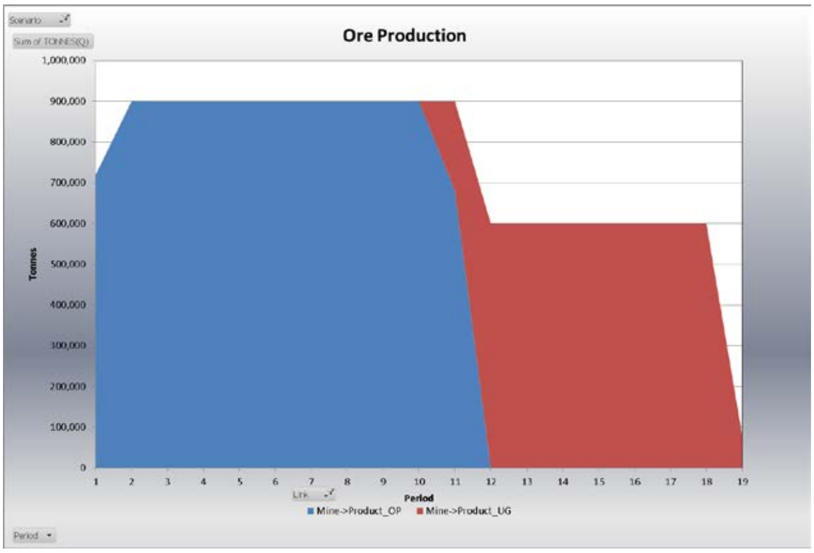

Figure 4 – Ore production schedule for the 'optimal' scenario (80 elevation)

3. <u>Automated MILP concurrent method</u>

In this method, the problem is solved using Mixed Integer Linear Programming techniques. This approach considers the competition of two alternative mining methods, UG and OP for the same mineral reserve (Ben-Awuah, Richter and Elkington, 2015). No predetermined transitional scenarios are configured, leaving the decision to the MILP scheduler. The MILP objective function to be maximized is the net present value of the project. This function is subject to a number of constraints like mining capacity, processing throughput limits, head grade targets and mining precedence.

The MILP approach allows mine planners to assess a great number of automated scheduling scenarios in a short period of running time, including both open pit followed by underground and simultaneous open pit and underground mining. In this illustrative exercise, the problem is modelled as a multiple mine problem in Evaluator, a strategic mine scheduling tool of Snowden Mining Industry Consultants. Two nodes are configured: 1) an open pit mine, containing all economic nested pit shells; 2) an underground mine, containing all economic UG envelopes and development. Material from these nodes is extracted once either through the open pit or through the underground stopes. Figure 5 illustrates a schematic flowsheet of the MILP problem. Links indicate allowed material flows.

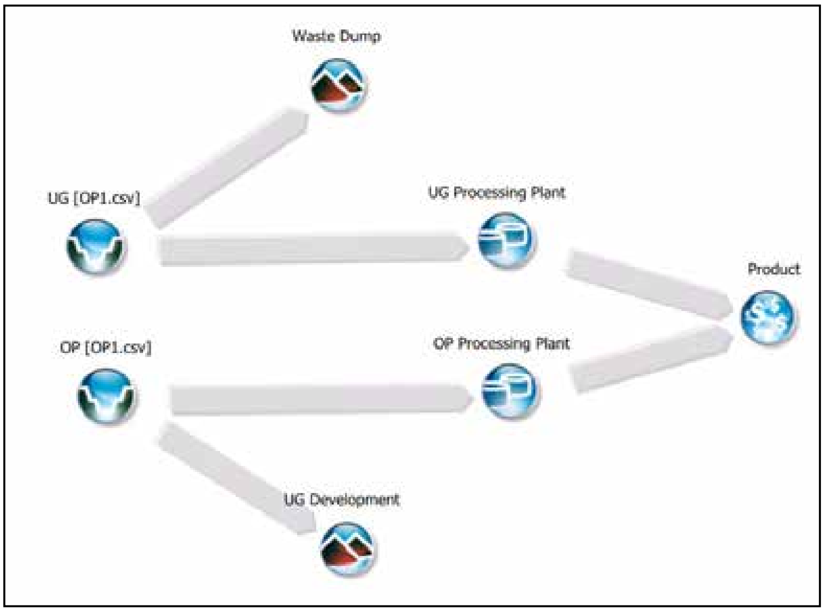

Figure 5 – Schematic flowsheet of origins and destinations in the MILP formulation

In contrast with heuristic methods, where material is initially flagged with a pre-determined origin/destination (open pit or underground), the MILP approach does not input this condition. Diverse mine scheduling scenarios are generated by changing the MILP problem constraints as desired. In this paper, four schedules were configured to illustrate the MILP method:

1) OP only. The capacity of 'UG to UG processing plant' path was set to zero.
2) OP followed by UG. The UG capacity was set to zero in the first stage of mine scheduling and, in a subsequent stage, the OP capacity was limited to zero.
3) Simultaneous OP and top-down UG mining. OP and UG were allowed to concur from the beginning of mine scheduling. UG mining was constrained to a top-down sequence.
4) Simultaneous OP and unconstrained UG mining. As in previous scenario but, not predetermining an operational UG sequence.

Table 3 shows the results for all these scenarios. As expected, results of scenarios 1 and 2 match to their corresponding cases of traditional heuristic methods (Figure 2 and Figure 3). Scenario 4 delivered the best economic solution, US$M 224.7. However, this sequence may not be operational as allows herratic underground mining. Scenario 3 balances economic performance with operational practicality (Figure 6 and 7).

Table 3 – MILP scheduling optimization results

Automated MILP Scenarios	NPV (US$m)
1) OP only	189.3
2) OP followed by UG	216.8
3) Concurrent OP and top-down UG mining	219.1
4) Concurrent OP and unconstrained UG mining	224.7

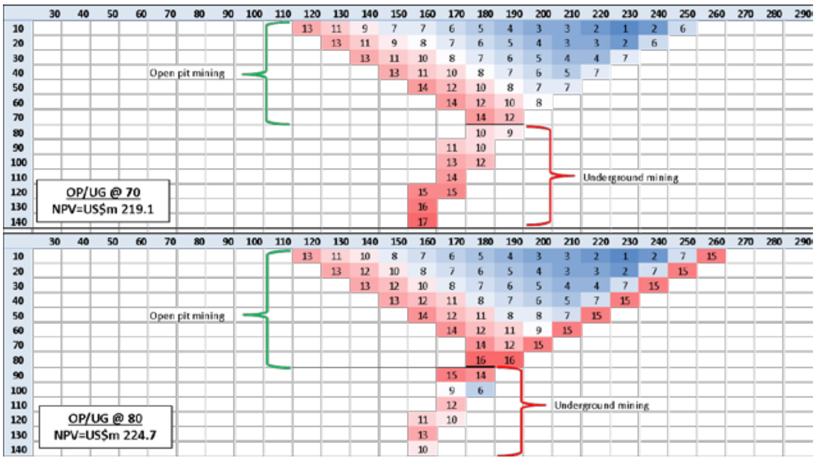

Figure 6 – Mine schedules: Simultanous OP and top-down UG mining (upper figure); Simultanous OP and unconstrained UG mining (lower figure)

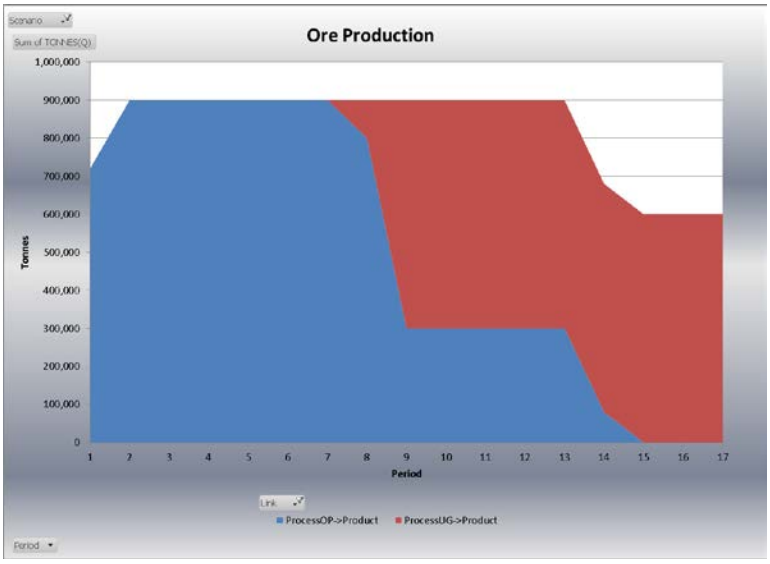

Figure 7 – Ore production schedule for the Concurrent OP and top-down UG scenario (3)

Sensitivity analysis

A sensitivity analysis was completed for those scenarios where OP is followed by UG in order to assess the transition point behaviour when varaying key mine planning parameters, such as commodity price, mining rate and mining costs. The results are presented in Table 4. Optimal transition point is not sensitive to price. However, transition moves up when production rate is reduced. Increasing open pit mining costs translate up the transition height. In contrast, an increment in underground mining costs allow the pit to go deeper.

Table 4 - Sensitivity analysis of NPV (US$m)

Transition Elevation	Base Case	Price		Production Rate		Mining Cost - Open Pit		Mining Cost - Underground	
		10%	-10%	10%	-10%	10%	-10%	10%	-10%
60	208.4	310.2	108.4	220.1	194.7	192.2	224.6	189.8	227.0
70	215.3	318.4	112.2	227.5	206.6	195.3	235.2	200.2	230.3
80	216.7	320.9	112.4	228.9	202.6	193.0	240.4	204.9	228.5
90	213.5	318.6	108.6	225.1	200.8	185.8	241.1	204.7	222.3
100	206.5	312.3	108.4	203.1	194.9	174.9	238.0	200.4	212.5

CONCLUSIONS

In this paper, a comparison of alternative analysis methods of transition OP/UG were developed. Three basic approaches are available to mine planners: 1) maximum pit analysis; 2) heuristic assessment and 3) MILP methods.

Comparatively, the best results in terms of NPV and running time were achieved with the MILP approach. In this case, the reserves are not preliminary associated to any particular mining method, allowing the scheduler to ascertain the optimal transition point. Besides, this method can be used to determine simultaneous open pit/underground mining while maximising Net Present Value. This approach allows for the analysis of many OP/UG complex project configurations, enabling the mine planners to make robust decisions.

Finally, the sensitivity study shows that there actually exists a transition zone that deliver similar economic outcomes rather than an exactly optimum elevation. Hence, it is important to develop a risk assessment which guides decision-makers to select those options that result in reasonable economic perfromance and low risks.

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RECENT ACHIEVEMENTS IN INVESTIGATIONS OF DYNAMICS OF SURFACE MINING HEAVY MACHINES

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RECENT ACHIEVEMENTS IN INVESTIGATIONS OF DYNAMICS OF SURFACE MINING HEAVY MACHINES

ABSTRACT

Load carrying structures of bucket wheel excavators, bucket chain excavators and spreaders looks to remain unchanged for years. However, the design in details is constantly optimized. Due to construction costs and complexity, the machines are expected to operate for decades. Fatigue is the main limiting factor of the structure durability. Alternation of structure stress is caused by fluctuation of operational loads but, in great share, also by vibration of the structure. Due to that fact, the optimization of the load carrying structure is required to fulfill the required operational service time.

The paper presents new achievements in the investigation of load carrying structures dynamics. It covers development of the application of newest research techniques. The three leading topics described in the paper are: experimental and numerical testing; development of the assessment of the structure on the basis of dynamic factor level and identification of the alternating loads derived from the dynamic behavior of the machine's superstructure. The research techniques are supported by proper examples in industrial application. New optimized bucket wheel excavator structure is compared to the older structures. The results indicates significant influence of the presented techniques application to the vibration level of the new machine.

KEYWORDS

Mining machines, dynamics, vibrations, numerical modeling, experimental testing

INTRODUCTION

Heavy machines are main equipment in the surface mining around the world. Increasing demand on energy delivery, keeps strong position of lignite in the world's energy market, despite development of plenty alternative source. With development of new technologies, also excavation of lignite passed through evolution. One of the leading drivers for changes and development are increasing requirement in safety level and environment impact reduction. Dynamics of the machines is directly correlated with noise and vibration. Due to excessive dynamics, degradation of the structure is increased, what can lead to failures, dangerous for the structure and working personnel. The second driver is constant costs optimization. Dynamics and vibration can play significant role also in this field. Some of the vibration derived factors which cause cost increase are listed below:

- structure fatigue cracks
- structure immediate cracks

- increased degradation level of mechanical components
- decreased machine output
- negative influence on the human
- other …

Occurrence of any of first three factors lead to the machine downtime, what in fact generates the cost itself. Of course additional cost are related to the repair, regeneration and cost of the spare parts. High level dynamics can also lead to the increase of the load on working elements or the mechanical components of the structure. As a result, the output must be reduced to decrease operational forces in purpose to avoid overloads and machine emergency stops. Last, but no least is human body influence factor. When operator is exposed to excessive vibration, his performance decreases, so output of the machine can be affected as well. In case of over normative vibration level, the work time must be shortened. In worst case, harmful influence on human body can completely exclude the operation team member (Harris, 2002).

Despite that the dynamics and vibration have strong impact on the machine performance, there are some design aspects which are not fully clarified. One of the main determinant of the structure design, in terms of vibrations, are the *dynamic factors* which are taken under consideration in calculation as a substitution of dynamic loads. However, its definition differs with respect to the standard which is used. Three standards can be distinguished when talking about design of load carrying structures of heavy bulk material handling machines:

- German Institute for Standardization (2015) - DIN 22261-2
- Standards Australia (1995) – AS 4324.1
- International Organization for Standardization (1994) – ISO 5049.1.

The first one is precursor in surface mining machine design standardization and partially bases on TGL 13472 standard (1974). It reached many reprints which the last one was in the last year (2015). The second one is independent standard developed in Australia, however in many aspects it refers to the German one. What is worth noting, Australian Standard, in case of fatigue and dynamics defines *dynamic factors* as the German standard. However, the values of those factors are much higher than the German one. Big disadvantage of both standards is fact that the procedure of experimental determination of those factors is not described. Reader experienced in Digital Signal Processing (DSP) is aware that the signal post processing is crucial in case of evaluation of such parameters. Lack of the clear definition leads to the situation where it might be difficult to compare results if the procedures differ. The last standard (ISO) gives least of all information about dealing with dynamics, while it not define dynamic factors and its level at all.

The paper will present developments in aspects of experimental testing, numerical modeling of dynamics and vibration on heavy mining machinery. There will be presented results of passed and ongoing investigations, which introduce new quality in this field.

EVOLUTION OF INVESTIGATION METHODS

The beginning of the construction of heavy surface mining machinery dates back to the1729, when Wilhelm Petrini design first floating bucket wheel excavator to excavate silt from the bottom of the river. In 1881 Smith's machine, which was the precursor of today's machines, was erected. The machine was drive by steam engine. In the 20' of the XX century, during the strong industrialization process, first machines (Fig. 1) similar to the operating today in open pit mines, were designed (Cohrs & Oberdrevermann, 2000; Durst &Vogt, 1986).

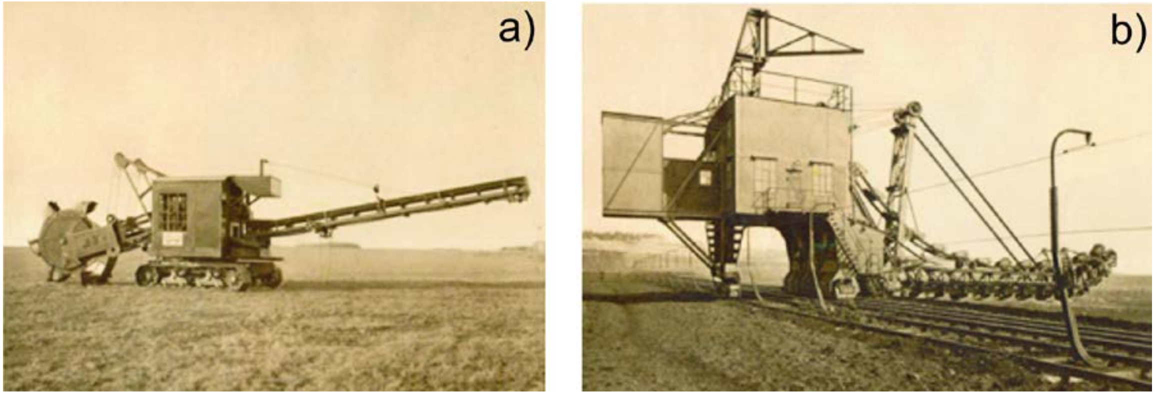

Figure 1 – Bucket wheel excavator (a) and bucket chain excavator (b) from the beginning of XX century (Babiarz & Dudek, 2007)

Simultaneously with industrial development the design/calculation methods of mechanical structures also evolved. Work (Волков & Черкасов,1969) presents model of superstructures of excavators, as the system of rigid elements connected with springs. Calculations were made in planar system, only for vertical direction. In 1990, doctoral dissertation (Petkovic, 1990) was released. It based on similar assumptions but it was developed by the vibration in horizontal plane. However, formulated models were simplified to that structure where counterweight boom was represented i.e. as lumped mass on the cantilever beam. Further develop made by Bosnjak, Petkovic, Zrnic and Petric (2006) introduces mass distribution in all nodes of the truss (Fig. 2). Researchers from Romania (Cioara 2007, 2009; Cristea, 2007) also presents the bucket wheel excavator as a set of rigid elements connected with springs (Fig. 3).

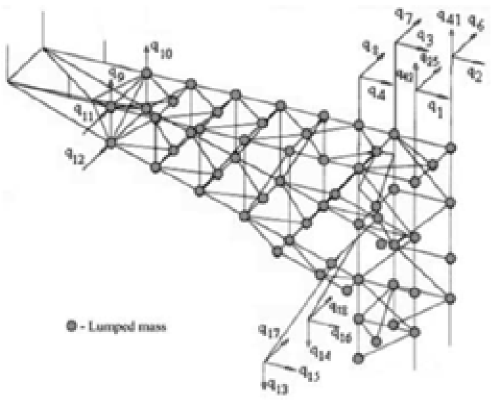

Figure 2 – Counterweight boom discrete model (Bosnjak, Petkovic, Zrnić, Petric, 2006)

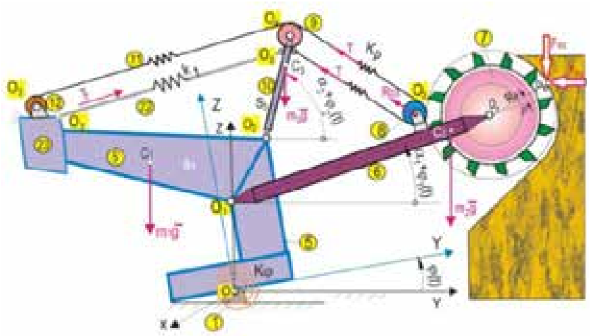

Figure 3 – Simplified dynamic model of bucket wheel excavator (Cioara, Lonei, Ioan, Dimitru, Dragos, 2009)

Modern approach to the heavy machines load carrying structure is based on finite element method (Rusiński, Czmochowski & Smolnicki, 2000). This method enables to create models of high complexity (Fig. 4, 5), which includes all local stiffness or complex boundary conditions i.e.: elastic support on crawlers or alternating support on bucket wheel (Fig. 4, 5).

Figure 4 – Finite element model of the spreader

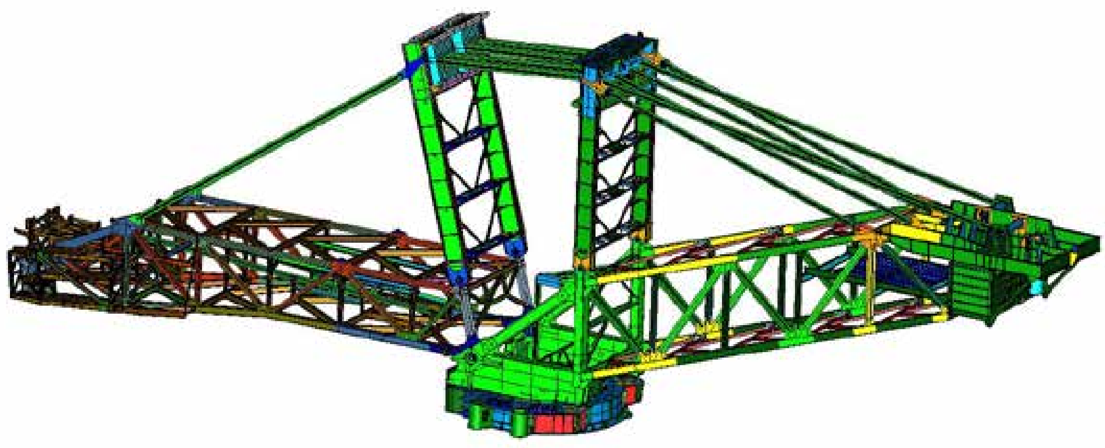

Figure 5 – Finite element model of the bucket wheel excavator

With that approach it is possible to perform complex vibration investigation like modal analysis. Vibration modes shapes identified that way, are accurate and easy for interpretation. However, frequencies of particular modes requires, in many cases, experimental validation and model tuning.

Experimental investigation, in case of machines of that size, can rise some technical problems. The book (Czmochowski, 2008) presents the classic approach to modal analysis with excitation impulse. Its proper realization require special equipment (Fig. 6).

Figure 6 – Mass release with use the special devise (Czmochowski, 2008)

A mass, which is hanged upon the bucket wheel boom, is rapidly released what is close to the impulse described by Dirac's delta. The excitation spectrum is wide, however, in case of the biggest machines, the excitation energy may not be enough. Moreover, delivery of the strict excitation impulse, in plane different than main vertical plane of the machine, is almost impossible. Literature (Gottvald, 2010) describes more spectacular method where the hanged mass is released with use of explosives (Fig. 7).

Figure 7 – Mass release with use of explosives (Gottvald, 2010)

As all can see, the above mentioned methods are difficult in realization and the obtained results, can have some limitation in the response band. Alternative to that method is operational modal analysis which do not require delivery of the external excitation impulse. Application of that method, for investigation of surface mining machinery dynamics, presents Pietrusiak (2013) in his doctoral dissertation. The big advantage of this method is that the investigated object does not have to be taken out from operation for the time of investigation. The excitation impulse is not required, while the method is based on the energy delivered to the system during operation. What is more, identified modal model includes influence of all operational parameters which can change the characteristics (i.e. shift of the frequency).

Coupling of the finite element analysis and the operational modal analysis introduces new standards in numerical-experimental vibration analysis. Numerical simulation and the experimental testing of objects with that complexity is difficult. Solution for that can be standardization of numerical-experimental models preparation. The author's numerical-experimental model development methodic is presented in Figure 8.

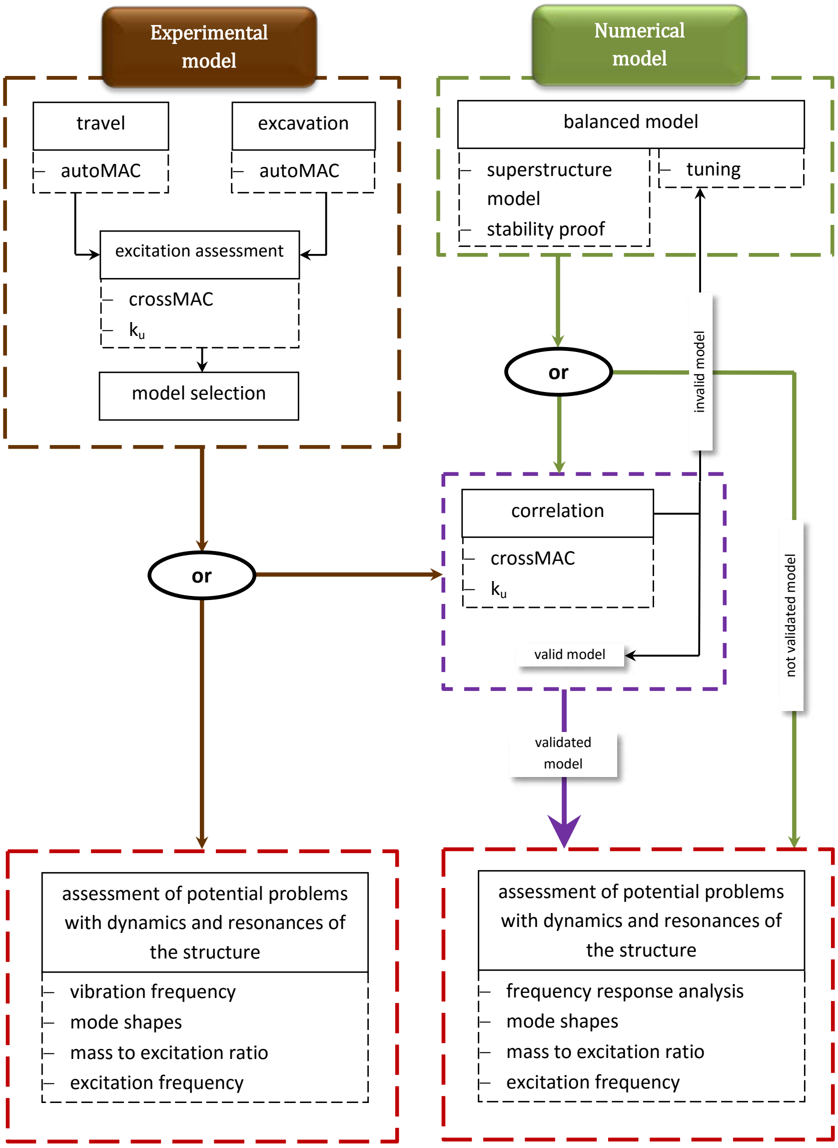

Figure 8 – Numerical-experimental methodic of vibration analysis

Presented approach is a combination of simulation, model tuning, experimental testing and data assessment with known tools like MAC(Modal Assurance Criterion) and developed by authors k_u general correlation factor.

RESEARCH DIRECTIONS AND ACHIEVEMENTS

Dynamic Factors – investigation and applications

Investigation of the Dynamic Factors

It was already mentioned in the introduction that *dynamic factors* are the normative measure for dynamics assessment. Lack of information about procedure of their experimental measure complicates proper investigation process. For example, only the German Standard (2015) presents the diagram with measurement points. However, it is only for three spreaders shown and no general rules for such tests are given. Further steps of signal processing, which influence significantly on the obtained results is not mentioned at all. Authors, in the paper (Rusiński, Moczko & Pietrusiak, 2014) presented comparison of the measured dynamic factors of the bucket wheel excavators type KWK 1200&1500. It is presented in the paper that for the dynamic factor analysis, the energy of vibration (RMS) should be used, different to German Standard (2015) which uses amplitude (pick to pick). That approach was used in several projects of technical condition assessment (Rusiński et al., 2010, 2011, 2013, 2014, 2015a, 2015b). As the vibration energy is significant in those testing, further steps in DSP procedure should be applied in the way to influence vibration energy in minimum. Different from the German Standard (2015), authors use acceleration measurements for determination of *dynamic factors* of BWE, not only spreaders.

Dynamic Factors Interpretation and Application

Assumption in the procedure measurements were described in the subsection above. The most important fact is the RMS application. Several numerical simulations in purpose to assess the fatigue degradation of structures (Rusiński et al., 2010, 2011, 2013, 2014, 2015a, 2015b) proved that the pick-to-pick value results in over range fatigue stress level. While investigated machines still operate, after 30-50 years, this proves that the pick-to-pick value does not represent properly the dynamic factors presented in the standard (German Institute for Standardization, 2015). The RMS value gives more reliable results, and this approach is applied with success also to other kind of mining machines, like for example crushers (Rusiński, Moczko, Pietrusiak & Przybyłek, 2013). In Figure 9 it is clearly visible that numerical prediction of crack location obtained with the use of described procedure matches to the real object cracks.

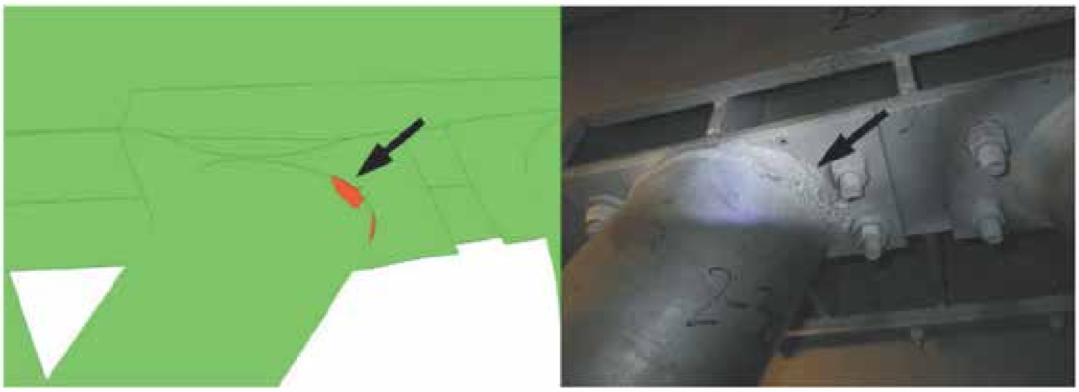

Figure 9 – Identification of the fatigue hot spots location with the procedure based on RMS measurements (Rusiński, Moczko, Pietrusiak & Przybyłek, 2013)

Properly measured dynamic factors are also used to select the mechanical objects which represents the lowest *vibration energy level* – degradation indicator. Moreover, the measured dynamic factors can be compared with the standard one. If the operational value is bigger than normative, the higher one is applied in numerical model. Thanks to that, obtained results are closer to the reality and reliability of proceeded assessment is elevated (Rusiński et al., 2015d).

Structure Numerical Modeling – model creation and validation

Most of the actual operating surface mining machinery in Europe are the objects at least 20, or often even 30 years old (Kasztelewicz, 2012). Design of those structures was conducted with the use of engineering tools adequate to the time of its erection. Development of the CAE tools in last decades, allows its common application in all fields of engineering, including heavy machinery design. Years of development of the numerical/mathematical models of bucket wheel excavators, enable to develop the model which was fully validated in terms of dynamic analysis. Presented in Figure 10, bucket wheel excavator KWK 1500.1 which was put into operation in 2013, is the first numerical model which was fully experimentally validated.

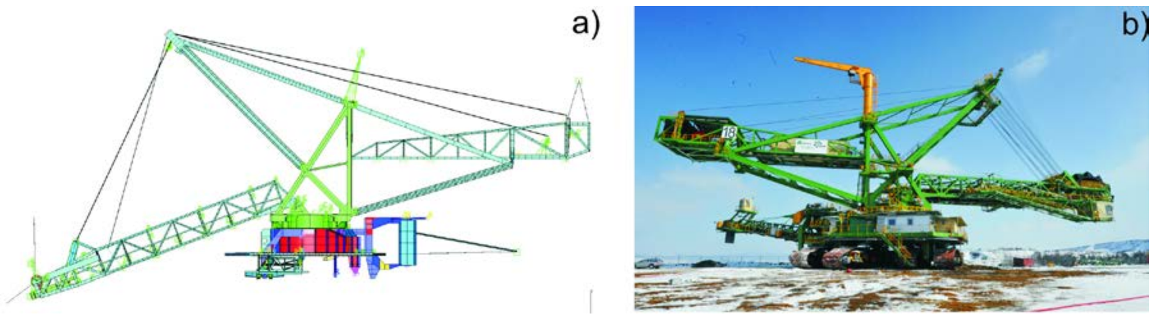

Figure – 10 Bucket wheel excavator KWK 1500.1: a) discrete model, b) real structure

The optimization of the structure was proceeded at the design stage (Rusiński , Kaczyński , Moczko & Pietrusiak, 2012) which resulted in several structure modification as well as change in the support of the counterweight boom, what influence all structure. Experimental testing on the operating object indicated that all the measured *dynamic factors* are within the normative ranges. What is more, in comparison to the similar type of the machines, the new excavator is the only one which characterizes with the *dynamic factors* (D) below the normative levels (Table 1) (Rusiński, Moczko & Pietrusiak, 2014). Result of those investigation are the first one literature documentation of fully validated model of bucket wheel excavator.

Table 1 – Comparison of dynamic factor of KWK type bucket wheel excavators
(Rusiński, Moczko & Pietrusiak, 2014)

machine element	type of D factor	experimental value 1/D			normative value 1/D_N	D/D_N		
		KWK 1200	KWK1500	KWK1500.1		KWK 1200	KWK1500	KWK1500.1
bucket wheel boom	mean D_Q	45	97	93	60	0,75	1,62	1,54
	mean D_V	22	41	20	10	2,20	4,10	2,00
bucket wheel boom jib	mean D_Q	37	45	208	60	0,62	0,75	3,46
	mean D_V	35	53	32	10	3,48	5,30	3,18
counterweight boom	mean D_Q	132	88	168	30	4,40	2,95	5,61
	mean D_V	71	84	30	25	2,84	3,37	1,22

Superstructure Vibration Based Fatigue Evaluation of the Undercarriage Elements

New direction of fatigue research is analysis of the correlation between the heavy machinery undercarriage load and superstructure dynamics. Available literature and standards do not reveal the necessity of undercarriage fatigue calculations of surface mining machines. Only static loads are usually taken into account. However, cases described in literature (Rusiński, 2006; Bosnjak, 2013) proves that fatigue of the undercarriage structures is something common. Confirmation of that fact are results of the recent investigations proceeded in one of the Polish open pit mine. Bucket wheel excavator presented in Figure 11 was equipped with system which allow simultaneous measurements of main vibration of the superstructure (Pietrusiak, 2013) and strain alternation in the undercarriage.

Figure 11 –Superstructure vibration to undercarriage strain relation – testing
(a - accelerometer, t - strain gauge)

Figure 12 presents traces of acceleration of the superstructure (blue trace) and unscaled trace, which represents strain alternations in the undercarriage elements (orange trace).

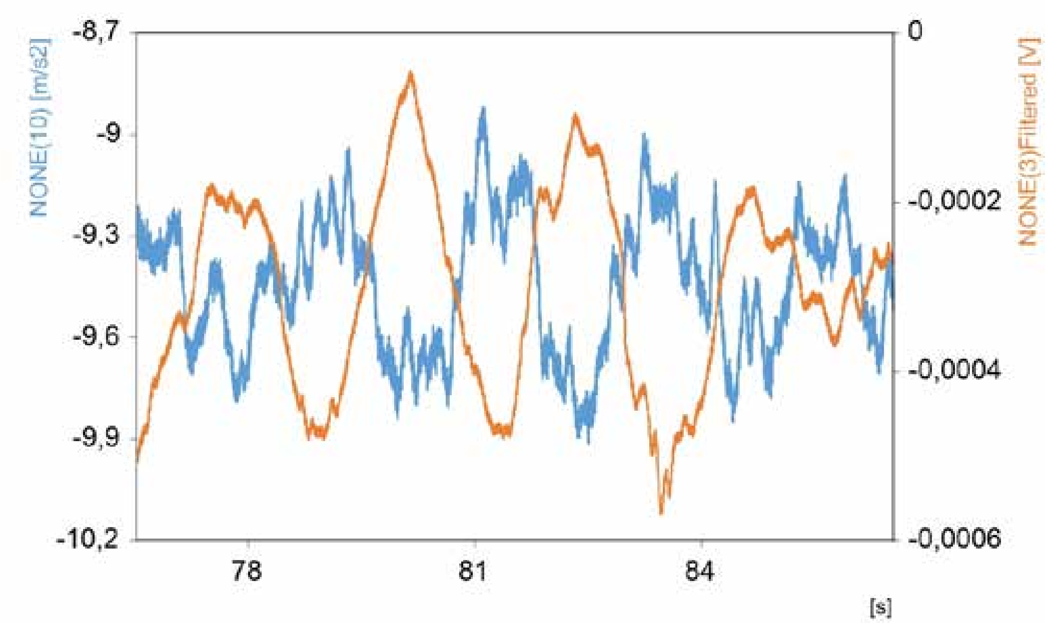

Figure 12– Superstructure vibration to undercarriage strain relation - traces

It is clearly visible that there is phase relation between the main superstructure's vibration component and the changes of undercarriage load. This is preliminary evaluation of the experiment which covered more measurement points on the superstructure and undercarriage as well. Eventually, the presented results confirms presence of the correlation between superstructure vibration and undercarriage load. Identification of this phenomenon is an important step in the undercarriage fatigue prevention.

SUMMARY AND CONCLUSIONS

Methods and examples presented in the paper reveals that even structures developed years ago, are constantly modified and still require development. Common access to specialized numerical and experimental engineering tools, can bring innovative solutions. However, it was also presented that development in design standards is not following the technology development. The lack of dynamics assessment requirements of the bulk handling machines is the good example of such situation.

The future challenges in development of surface machinery dynamics have to face both, maintenance of old, worn-out machines and design of new structures. In case of old machine, the aim is to extend the time of the operation till the end of the ore sources in the mines. The second case is to design safe, reliable and low operational costs machine which will operate in new mine. As it may looks the aims differs much, there is a strong link between them. As the old machines was used as testing field for last decades, the new designs will be raised with the knowledge gained in that time. The future time will be the exam for newest inventions in that field, what shall bring the knowledge for new generations and "close the circle".

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RISK ANALYSIS TO DECIDE THE USE OF A LHD (LOAD HAUL DUMP) EQUIPMENT WITH A REMOTE CONTROL DEVICE INSTALLED ON IT IN OPEN STOPE MINING METHOD

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ABSTRACT

The related work aims to contribute to a decision analysis, for the use of LHD (Load Haul Dump) equipment operated by a remote control device, in an Open Stope Mining Method with steel cables support installed on the stope roof. Currently is made a pre-assessment, based on a drill core analysis (RQD and Q indexes), which defines the stretch to be supported by using cables. At the stage of ore extraction, after each detonated drive section, the mine supervisor evaluates the situation and allows or not the use of the LHD equipment with the remote control device to operate in the drive extraction process within the open Stoping. The proposed method is based on an analysis of technical and economic criteria, defining scores for activities standardization, intending to reduce the dilution index in the ROM (Run Of Mine) (Waste/Ore) and to increase the safety of the LHD operation with the remote control device in an open Stoping mining method extraction process.

KEYWORDS

Stoping, ROM, LHD, dilution.

INTRODUCTION

The increase in dilution leads to higher transportation and processing costs, and generates a higher amount of waste and tailings as well as many consequences to the environment. According to others sublevel caving method studies, the acceptable values to ROM recovery are in general above 75% with 35% in dilution (Kvapil, 1992, p. 1806). On the other hand, the sublevel stoping methods with pillar recovery present typical values economically favorable, with 90% recovery and 20% in dilution (Haycocks & Aelick, 1992).

Previously, in FERBASA Ipueira mines, the exploitation control decision analysis and the LHD use in open stopes were based on the exploitation method (Open Stoping), and inside the programming of the roof steel cabled drifts (CC) above the ore, after each blasted section the mining supervision staff analyzed and authorized the LHD entrance with remote control to begin the ROM extraction work inside the open stope. This decision was made regarding the steel cable system and the open stope roof visualization. The proposed method is based on empirical data analysis, technical and economic criteria to define the risk grade about the LHD use, if it is traditional, when the operator is inside the equipment, or remotely controlled. The presented methodology in this study is being practiced currently with LHD operations at levels and drifts favorable to it.

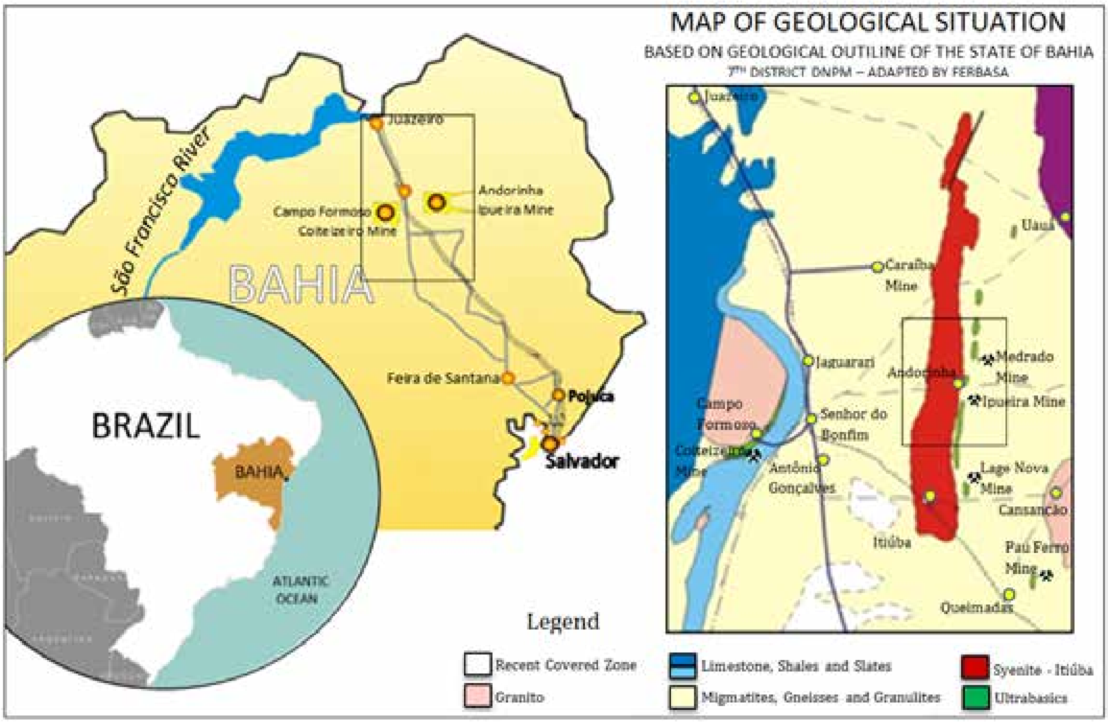
STUDY FIELD ASPECTS

The Ipueira Mine, where this research was conducted, belongs to the FERBASA Group, located at the North-Northeast of Bahia state, Andorinha city – Brazil. By the highway network from Salvador until Ipueira Mine, final destination, totalizes 447 km, on coordinates 39º45’56" West longitude and 10º21’51" South latitude.

The mines inside this group are part of a chromitiferous district of Jacurici Valley with 100 km in extension, various Jacurici Valley maphic-ultramaphic complex rock bodies mineralized in chrome, They are embedded bodies in granolithic and gneissic-migmatitic rocks of the São Francisco Cráton basement, from little to medium extensions, creating the so-called chromitiferous district of Jacurici River Valley (Silva, 1998).

A complex fault system crosses all lithologies, these faults intercut the ore body creating blocks within 2 to 20 meters in width in longitudinal and transversal ways.

Figure 1 – Location of Ipueira Mine and Regional Geological draft.

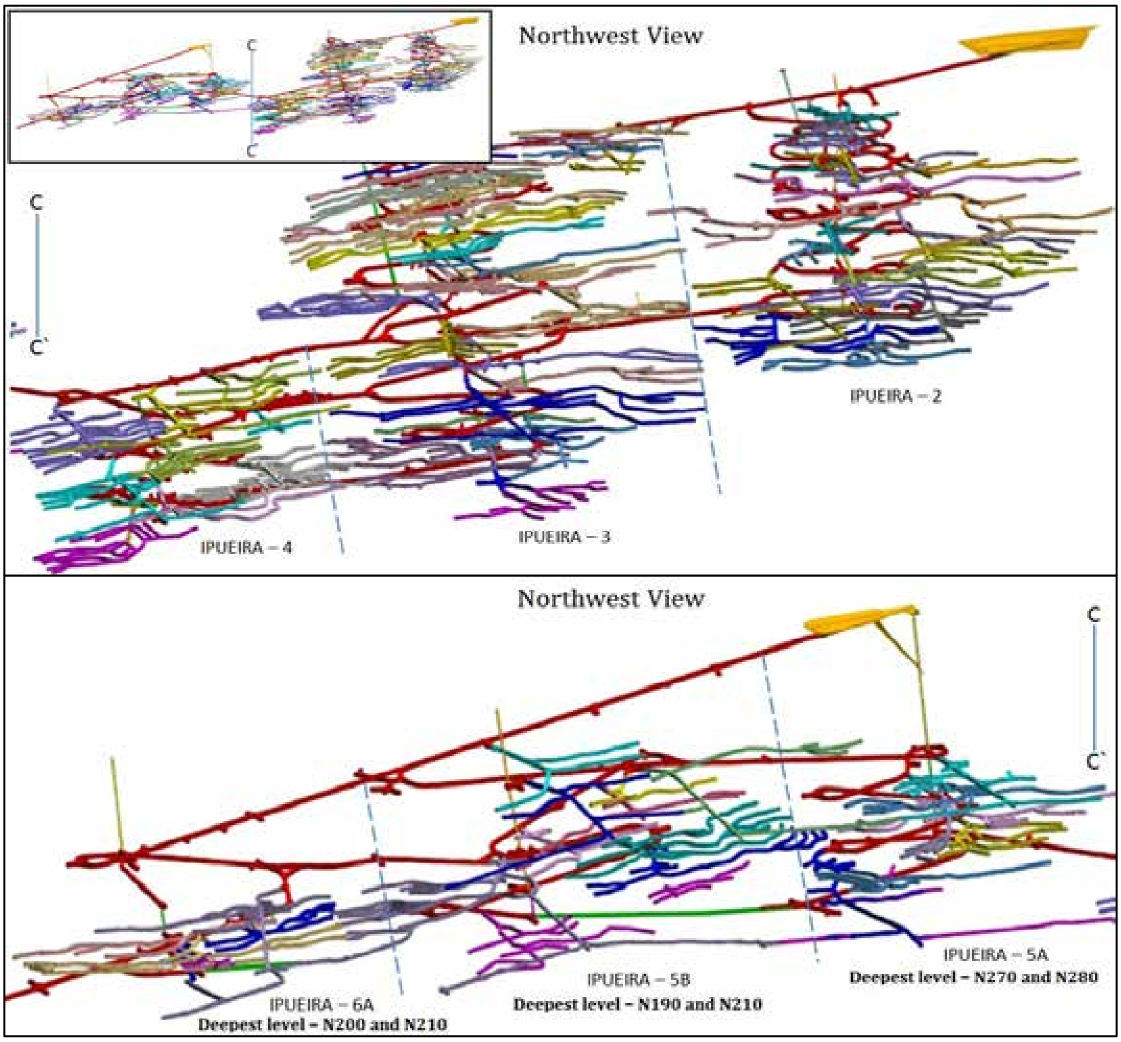


Source: Internal archives modified from the Geology and Planning Division/ FERBASA (2013).

The Ipueira Mine is divided in six operational units: Ipueira mines 3, 4, 5A, 5B, 6A and 6B. The units have in average a longitudinal extension of 500 m North-South way. The mineral asset is chromite, which commercialized products are *lump* 30 to 40% in Cr_2O_3 (fraction higher than 2 ½ inches), the fine concentrated and chromite sand (fine fraction).

This methodology was applied in the case study of levels N190 and N210 of Ipueira mine 5B, levels N200 and N210 of Ipueira mine 6A and currently in Ipueira mine 5A at N270 and N280 levels.

Figure 2 – Exploitation level with controlled extraction.



Source: Internal archives of Geology and Planning Division/ FERBASA (2015).

ORE BODY AND WALL ROCKS CHARACTERISTICS

The chromitite body with an average thickness of 8 m is embedded in serpentinites, which are positioned as described: on the hangwall, serpentinites from of the alteration of olivine-orthopyroxene-spinel cumulates, locally denominated of serpentinite-olivine-orthopyroxene; in footwall, serpentinites orthopyroxene-olivine-spinel cumulates locally denominated of serpentinite-orthopyroxene-olivine.

According to Lima *et all* (2005), the lithologies directly related to Ipueira Mine exploitation operations, present the following basic geomechanical characteristics:

• Serpentinite-olivene-orthopyroxene – result of ultramaphic altered rocks and presents compressive strength between 60 to 100 MPa and RQD between 50% and 80%. In exploitation zones they are positioned on the stopes roofs.

• Chromitite – in units 2, 3 and 4 of Ipueira Mine is presented in a non-altered state, with compressive strength commonly above 100 MPa and RQD between 40 and 70%. It is densely intersected by joints filled with carbonates, which exhibit low cohesion and low shear strength. During the exploitation process they are positioned in the drifts pillars and roofs. Its high density (above 3,5 ton/m³) increases the risk of wedges detachment on the sides and the roof of the production drifts.
• Serpentinite-orthopyroxene-olivine – results of ultramaphic altered rocks and presents compressive strength between 40 to 100 MPa and RQD between 40% and 80%. In the exploitation panels they are positioned in the drifts pillars and on the floor.

CURRENTLY APPLIED EXPLOITATION METHOD

From the end of the 90s, mining started in the ore body layer in horizontal position, verified with the deepening of the ore body, the sublevel caving method started to behave as open stope in retreat method, in which, the openings remain with no link with above levels.

The sublevel caving and open stope in retreat methods with development in the waste and when allowed in ore, in the way practised in Ipueira Mine, have the following characteristics:

- Each production level is accessed by an access ramp with a nominal transversal section of 4,5 by 5,0 m, opened from the access ramp;
- Production drifts, with nominal sections of 4,20 by 4,20 m, are opened in North-South direction, from the access ramps until they reach the level boundaries;
- In open stope in retreat method, two to five drifts are opened, depending on the continuity and the horizontalization degree of the ore body;
- The drifts are supported with shotcrete with steel fiber and steel bolt, CA50 type of ¾" in diameter anchored with water and cement;
- At the end of each production drift free faces are created, by drilling a blind slot on the roof, which are drilled with *in the hole* (ITH) equipment, and after enlarging its holes with explosives (Silva e Moura, 2004);
- After range in holes sections are drilled, to exploitation and to reinforcement holes sections, braided steel cable (seven wires) with nominal diameter of 15,2 mm are anchored in them with a paste of water and cement too;
- The exploitation drilling is made in fan, with 2,2 m burden, and their height varying in general from 10 to 20 m, adding a complement drilling from 5,5 m to 7,0 m to install dilution steel cables.
- The fan side holes of the exploitation sections are projected with a minimum 45 degree angle related to the horizontal axis, thus providing the formation of gutters to direct the ore dismantled ahead of the galleries;
- The steel cable reinforcement holes drilling have sections with 5 or 7 fan holes burdened 2,20 m from each other. They are intermediate sections among the exploitation sections and act as secondary reinforcement during the blasting phase to the safety of the LHD's work;

The drilling is performed with hydraulic drillers mounted in *fandrills Cubex Aquila C, Atlas Simba H 252, Sandvik Solo RTS and Cabolts type*;

- The holes are loaded with assembled emulsion and started with by electrical and or electronic fuses;
- The ore extraction in the piles formed by the blastings is made with LHD's, which their shell capacities varies from 3 to 5 m³, using the currently or the remote control method, depending on the planned programming;

- The ore is transported by the LHD's until enlarged openings built in the access ramp, called PA (Support Point in Portuguese);
- In these support points the piles formed by the LHD's disposal are loaded with loaders on tires (CAT 962 G or Volvo L 120) and put them in trucks Scania P 420 type of 40 t capacity and Scania G380 de 50 t of useful load.

LHD RISK ON EXPLOITATION OPERATION

In a sublevel caving and open stope both in retreat system, which the non-planned dilution according to Henning & Mitri (2008), results in a rupture with subsequent fall and slip of the host rocks, roofs and side into the open stopes. This is a risk situation to the LHD operation, even greater when the LHD must go inside the open stope and recover more material (ROM). The dilution may be minimized installing steel cables to stabilize walls and roofs of the open stopes, and dedicating exclusive time to the level extraction, with a minimum numbers of programmed stops, increasing so the extraction speed, minimizing confinement stress in the rock mass and the risks of great mass blocks as well as little unstable fragments. This dilution is the responsible for great risks at the LHD ore extraction phase.

According the study of (Lima et all, 2008), in a pilot extraction the non-planned dilutions at the level N360 of Ipueira 4, obtained a lower dilution with the new method (3,15%) with steel cables holding the roof and the walls of the open stopes, in comparison with historic data of Ipueira Mine dilution in 2006 (36,25%), with an excellent recovery (97,68%). These data are demonstrated in Table 1 and in Table 2.

Table 1. Comparison between non-planned dilutions in the pilot extraction in Ipueira Mine.

Place	Blasted Ore (t) (1)	Total Blasted (t) (2)	Extracted Ore (t) (3)	Total Extracted (t) (4)	Non-Planned Dilution (%) (5)*
M.Ipueira/2006	424.607	220.547	288.380	687.193	36,24
N.360Ipueira/2007	13.252	2.836	12.945	16.227	3,15
Note: *Calculation using the following equation (5) = 100 x [((1)/(2) – (3)/(4)) / ((1)/(2))]					

Table 2. Recovery in pilot extraction N360 IP4 and in Ipueira Mine in 2006.

Local	Blasted Ore (t) (1)	Extracted Ore (t) (3)	Recovery (%)*
M.Ipueira/2006	424.607	288.380	67,91
N.360Ipueira/2007	13.252	12.945	97,68
Note: *Calculation using the following equation (%)* = 100 x [(3)/(1)]			

Castelo and Alves (2009) showed a comparative dilution calculation with exploitations sections with and without CC cables to quantify the non-planned dilution, and were analyzed exploitation fandrills sections of ROM varying from 7,0 to 12,0 meters in height. In the exploitations sections with CC cables this height was addicted with 5,5 meters more to the containment dilution cabling.

According to Alves *et al.* (2009) based on Lima *et al.* (2008), the ROM relation value and the waste wall rock open stope roof and sides may vary with the rock quality, open stope opening degree, amount of adjacent drifts, presence of open stopes in already exploited levels nearby the one in exploitation stage, among others. Therefore, the extra foreseen ROM from the ore value in the ROM will match to the waste material dilution. The considered reduction on this dilution will imply in a greater safety LHD exploitation operation with the conventional or by remote control method.

Tables T3 and T4 demonstrate the ROM waste reduction data in exploitation stretches with cabled roofs. The table T3 illustrates IP4 and IP5 mines where were done surveys and quantified the exploited sections evaluated, as well as, their foreseen production.

Table 3. Comparative of exploitation sections with and without dilution cables.

Section Status	Mine	Exploitation			Foreseen Exploitation		
		Number of Sections	Number of Holes	Drilled Length (m)	ROM (ton)	Lump (ton)	Relation
Exploitation Sections without	IP4 and IP5A	98,00	924,00	9.392,09	55.928,00	21.616,00	2,59
Exploitation Sections with CC	IP4 and IP5A	98,00	978,00	13.357,40	53.377,00	22.948,00	2,25
Exploitation Sections	IP4 and IP5A	46,00	473,80	7.580,80	25.054,51	10.771,51	2,21

Table 4 illustrates the waste reduction in the ROM in exploited sections with CC cables to temporary contention of ROM waste secondary dilution. The exploited sections (Classified*) with cable (CC) were analyzed with a few criteria such as rock mass quality at the first 20,0 meters from the beginning of exploitation, analysis of present discontinuities among other factors.

Table 4. ROM relation in waste open stope roof dilution.

Section Status	Mine	Foreseen Exploiation			Exploitation	Waste	Relation
		ROM (ton)	Lump (ton)	Relation	ROM Realized (ton)	Open Stope roof	ROM Dilution
Exploitation Sections without	IP4 and IP5A	55.928,00	21.616,00	2,59	102.733,38	46.805,38	1,84
Exploitation Sections with	IP4 and IP5A	53.377,00	22.948,00	2,25	75.254,63	21.877,63	1,41
Exploitation Sections	IP4 and IP5A	25.054,51	10.771,51	2,21	29.564,32	4.509,81	1,18

To increase safety in this kind of operation, some technical criteria were defined for instance: wall rock geomechanical and geotechnical characteristics, mineralized ore body geometry, contention type, exploitation operation time and the potential of the exploitation section in concentrated ore tons in ROM, to decide the LHD type of use (Lima et all, 2010).

ASSESSED ITEMS TO STUDY IMPLEMENTATION

It was observed the following head factors to compose the safety scores, as well the rock mass quality on the roof and on the open stope sides, hydraulic radius versus rock mass type, few steel reinforcement conditions, observation of the state peeling roof, near open stopes proximity, general roof situation regarding to transversal section among others.

OPEN STOPES STABILITY CONDITIONS

Potvin's Empirical Method (1988)

Potvin's empirical method (1988) was originally conceived to its application on sublevel open stope blasting method, in which the ore is blasted using long holes drilled from the drifts located at the sublevels. In this case, the dilution to be controlled results from the waste wall rock knocking down over the ore pile accumulated in the open stopes. The steel cables must be positioned in the waste wall rocks mainly at the cover. To estimate the necessary reinforcement amount, this method is based on the open stopes hydraulic radius and rock mass quality.

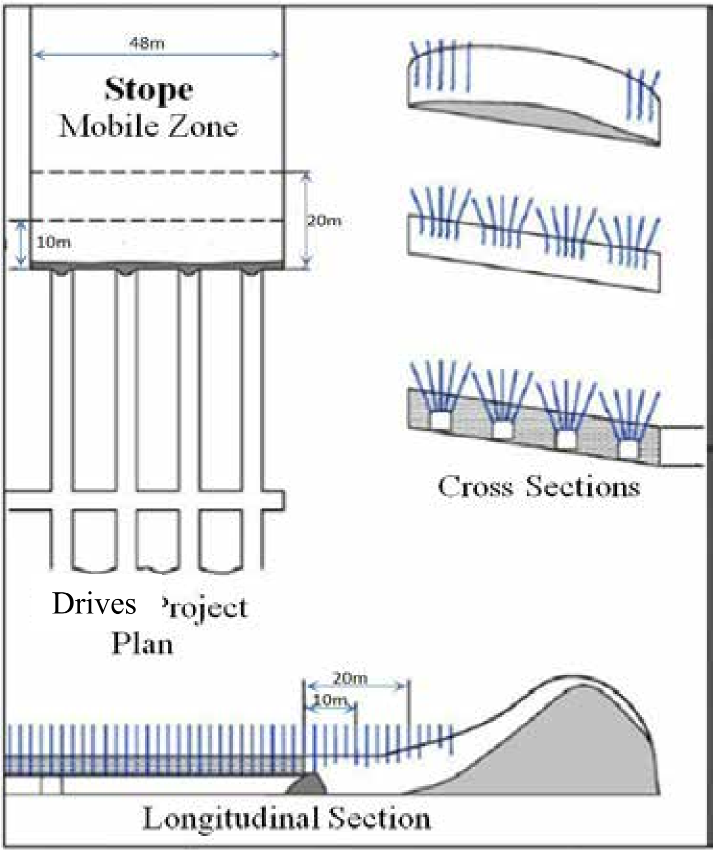
The hydraulic radius (H_r) is defined as the area divided by the open stope face perimeter to be reinforced as a way to quantify the porthole. In Ipueira Mine study, it was adopted the simplified model shown on Figure 1, in which the ore layer is horizontal and continuous, the hydraulic radius of the exploitation panels with 1 to 5 drifts are calculated, zones with 10 and 20 meters in length, and it was obtained the values shown in Table 1.

The maximum number of drifts contributing to form the common open stope was limited to 5 for including the present situations found in Ipueira Mine.

Table 5. Hydraulic radius referred to panels with 1 to 5 drifts.

Nº OF DRIFTS	ZONE LENGTH	ZONE WIDTH	HIDRAULIC RADIUS
1	10	12	2,7
2	10	24	3,5
3	10	36	3,9
4	10	48	4,1
5	10	60	4,3
1	20	12	3,8
2	20	24	5,4
3	20	36	6,4
4	20	48	7,1
5	20	60	7,5

Figure 3 – Estimate simplified model of open stope hydraulic radius.



Source: Lima et all. (2010)

ROCK MASS QUALITY AND STABILITY PARAMETER

The main lithologies existing in exploitation zones of Ipueira Mine, with the respective geomechanical characteristics are the following:

- **Serpentinite**: is the main ore wall rock, it results from ultramaphic rocks alteration and present compressive stress between 60 and 100 MPa and RQD between 50% and 80%. In general they are intercut by various fault systems and intrusion pegmatites. They are present in exploitation drifts of Ipueira Mine and in open stopes roofs

- **Pegmatite**: they are acid rocks in general appears in the form of small shafts, its compressive stress varies too much, depending on the alteration state. Usually they present RQD below 50% and have great propensity to change and generally induce instabilities in the excavations that cross them.

To use the Potvin's empirical method (1988), the rock mass quality is evaluated by the Barton's Q geomechanical classification system (2002), adopting a unitary value to the J_w and *SRF* parameters. The index defined so is called modified Q index, calculated by (Hoek *et al.*, 1997):

$$Q' = \left(\frac{RQD}{J_n}\right)\left(\frac{J_r}{J_a}\right)$$

Table 6 – Geomechanical classification parameters of open stopes roofs rock mass.

PARAMETER	MINIMUM	AVERAGE	MAXIMUM
RQD	50%	65%	80%
J_n	6	4,5	4
J_r	1	1,5	2
J_a	4	3	2,5
Q´	2,5	7,2	25

STABILITY PARAMETER

In Potvin's empirical method (1988), besides the rock mass quality, it must be considered the three below factors. The factors A, B and C are combined with the Q' rock mass parameter to obtain the stability parameter N'.

Therefore, N' = Q'.A.B.C and Q' = (RQD/J_n) x (J_r/J_a), where: A – tension concentration in open stope wall; B- critical orientation of the dominant joint system; C- open stope wall inclination relative to the horizontal.

Factor *A* – stress concentration in open stope wall. Uniaxial compressive strength, σc = 80 Mpa; Tensions to 300 m, σv = σh = σ*z* = 0,027x300 = 8,1 Mpa; stress concentration in open stope wall σ1 = 2σv = 16,2 Mpa; Relation σc/σ1 = 80/16,2 = 4,9. Factor A = 0,4.

Factor *B* - critical orientation of the dominant joint system. Much variability in open stopes roofs directions and dips discontinuities which is adopted an average value of factor *B*. Factor B = 0,6.

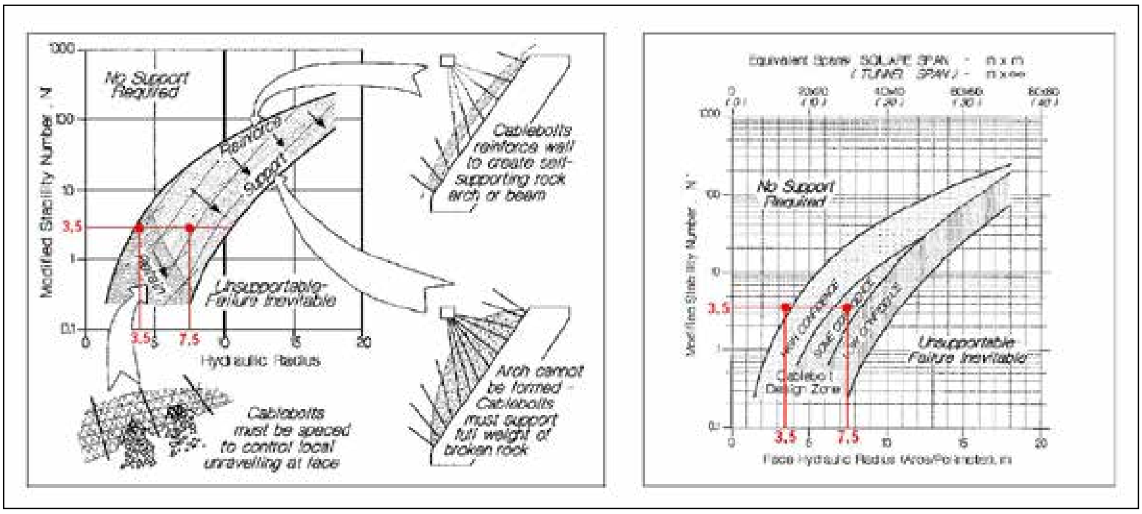
Factor *C* - open stope wall inclination relative to the horizontal. It is considered a sub-horizontal roof with a minimum value. Factor C = 2,0.

The combination among hydraulic radii with the N' parameter results in a point in the stability graph, indicating 3 situations: Self -supporting ; Stable with cables ; and collapse or caving . Applying the equation (2) the average value to *Q'* (Table 2), and the values calculated to A, B and C parameters, the stability parameter is obtained:

$$N' = 7{,}2x0{,}4x0{,}6x2 \cong 3{,}5$$

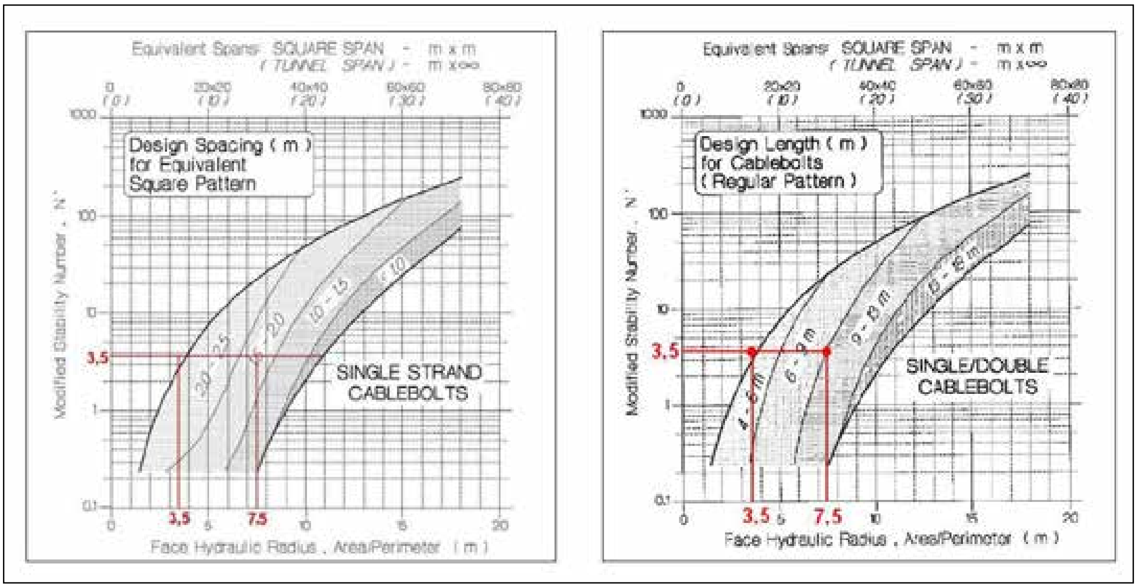
First, in figure 2 it is possible to verify that the studied cases refers to open stopes which may be stabilized with cables with medium to high reliability, to hydraulic radii equal to 3,5 to 7,5 m, respectively. It is also verified that hydraulic radii inferior to 3,5 m need no support. According to the diagram in figure 2, may be observed that the cable performance in this analysis in particular is by open stope wall support or by block retaining, to hydraulic radii equal to 7,5 or 3,5 m, respectively.

Figure 4 – Cable actuation system and strengthening the reliability zones.



Source: Johnson et al. (2000)

Figure 5 - Cable System performance and reinforced reliability areas.



Source: Johnson et al. (2000).

According to Figure 3, the recommended mesh spacing cables in meters is between 2,5 m x 2,5 m and 1,8 m x 1,8 m, to a single cable per hole case adopted in Ipueira Mine, corresponding to hydraulic radii of 3,5 and 7,7 m, respectively. Concluding the empirical dimensioning in Figure 8, it is verified that cable lengths recommended is either 4 or 9 meters.

SCORES TO DECISION MAKING

Four steps were adopted to LHD risks evaluation during the remote control operation inside Ipueira Mine open stope – FERBASA, according Lima's and Alves' (2010) technical and economical viability study. These steps to decision making went through Calculation of Technical and Operational Risk Scores, Production Incentive Score, Total Score and Recommended Action.

I - CALCULATION OF TECHNICAL AND OPERATIONAL RISK SCORES

To make these calculations ten relevant factors among the most relevant observed by the FERBASA technical team attributed for Lima e Alves (2010).

1- OPEN STOPE ROOF ROCK MASS CLASSIFICATION AND CHARACTERIZATION

Drill core geotechnical characterization and classification analysis through geological probing sections (Scale 1:500) to calculate RQD index, RMR system and/or Q system. Such classification is made by drill core holes on the surface or in underground research drifts, rather, the best representation to the chosen lithological horizon. These holes are sampled and essayed in stretches in general from 7,0 m to 10 m above the ore body, due to ore body irregularity in contact with the wall rock.

The Table 2 presents open stopes roof rock mass geomechanical classification of Ipueira Mine 5A, level N210-5A.

Table 7 – Geomechanical classification parameters from Ipueira Mine 5A, Level N210-6A.

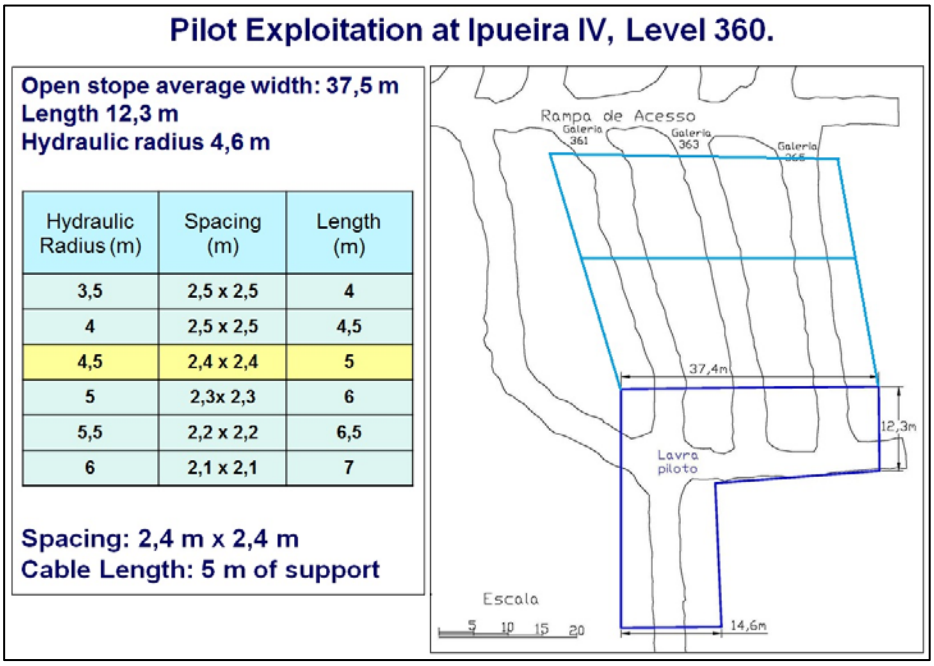
Classification and Characterization	Serpentinite
RMR	56,05
RQD (%)	52,6
Compressive strength (Mpa)	112,5
Average recovery (%)	93,02
Average spacing (m)	0,14
Discontinuities index	17,36
Described samples total lengths (m)	730,15

Source: Geomechanical Sector/FERBASA (2014).

2. HYDRAULIC RADIUS VERSUS ROCK MASS TYPE

The N' parameter combined with the hydraulic radius results in a point of stability graph indicating three situations: self-supporting, Stable with cables and Collapse or Caving.

Figure 6 – Hydraulic radius analysis to LHD exploitation limit.



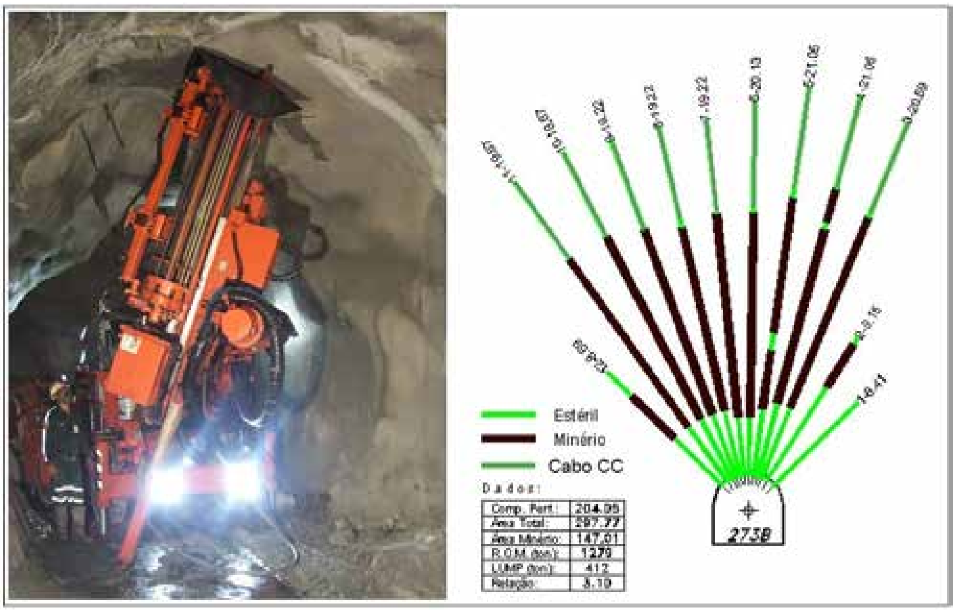
Hydraulic Radius (m)	Spacing (m)	Length (m)
3,5	2,5 x 2,5	4
4	2,5 x 2,5	4,5
4,5	2,4 x 2,4	5
5	2,3x 2,3	6
5,5	2,2 x 2,2	6,5
6	2,1 x 2,1	7

Source: Lima et all. (2010).

3- CABLE REINFORCEMENT CONDITIONS

Basically the reinforcement dimensioning with cables is based on the average spacing between cables on roof drill mark and cables lengths. In practice the spacing and the length can not be made obeying strictly the empirical recommendations, because of operational economical reasons. The relation between installed length and recommended one results in an index used to measure the roof detachment cabling control.

Figure 7 – CC cable drilling system at the same exploitation fandrill.



The operational characteristics of cabling are rigs with seven wires of 5 mm diameter each, rig nominal diameter of 15,2 mm and nominal breaking load of 22 t. In each hole with 51 mm in diameter, where is installed a unique rig and anchored with a cement paste fill with a water/cement factor of 0,35 (water weight / cement weight).

4. OBSERVATION OF THE STATE SPALLING ROOF

The roof stability condition referred to detachment can be visually made by experienced personnel or by monitoring equipments such as the CMS (*Cavity Monitoring System*) or other one similar. This observation shall be done with scheduled stops to event analysis, usually before and after the operations. These intervals shall be established in function of technical information and local circumstances, sometimes intervals of a few hours or days. If occurs detachments which offer risk, it is necessary to standby for stabilization and only liberate a LHD after 24 hours before entrance authorization inside the open stope and limiting its reach. Technical criteria are being studied to enhance this analysis.

5. PROXIMITY OF NEAR OPEN STOPE AND ITS DEEPTH

This parameter considers this effect in risk evaluation. It is known that near open stopes previously exploited cause stress increasing and consequently damage in near rock masses.

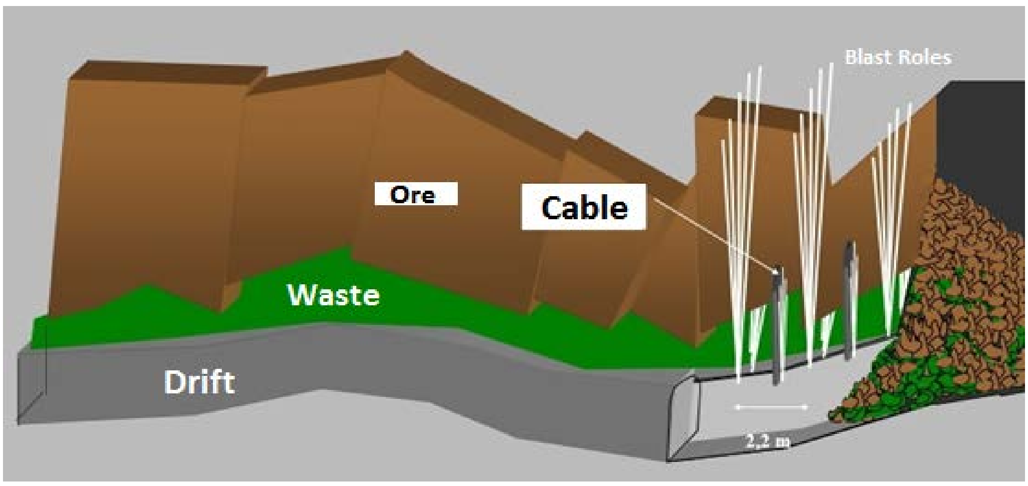
6. OPEN STOPE DEPTH

It is known that the average depth is more favorable to stability, because the horizontal stress helps to maintain the contacts among rock blocks. And low depths cause detachment fall and great depths cause compression rupture.

7- AVERAGE HEIGHT FROM THE LAST 4 FANDRILLS INCLUDING THE CURRENT ONE

The ore and the wall rock characteristic is very irregular, high open stopes result in cabling quality loss and hole deviations. High roofs increase porthole area (hydraulic radius) and side rocks slide and detachment possibilities. Thus, it is necessary to measure an average height of the last four fandrills which matches to LHD work distance (8 to 10 m).

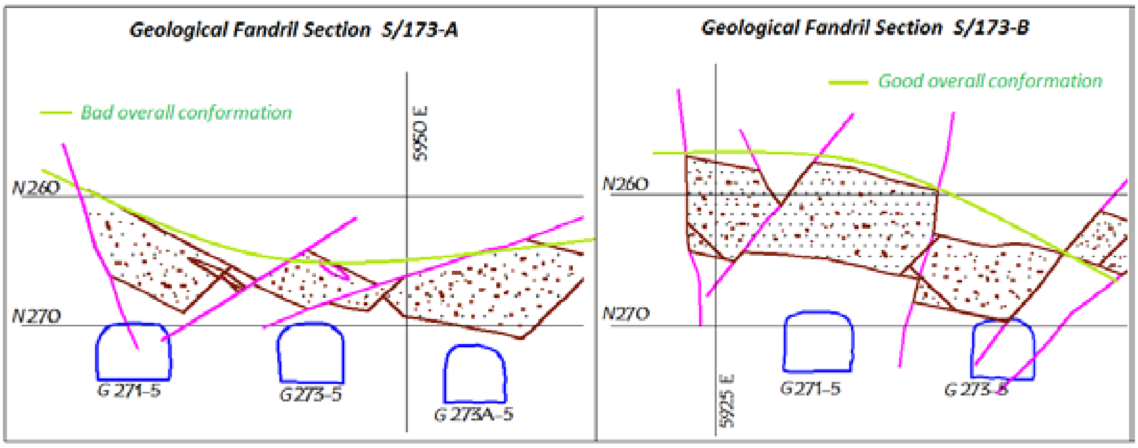
Figure 8 – View in longitudinal section of exploitation open stope.



8- OVERALL ROOF CONFORMATION ON THE CROSS SECTION BENDING

In a set of drifts exists the possibility of generate roofs with bended roof conformation, when the central fandrills reach quotas higher than the lateral fandrills. This condition is favorable to central drifts stability and deserves to be measured. On the opposite case occurs when the stability of central drifts present lower quotes in relation to the lateral fandrills creating an unfavorable condition to the open stope roof stability.

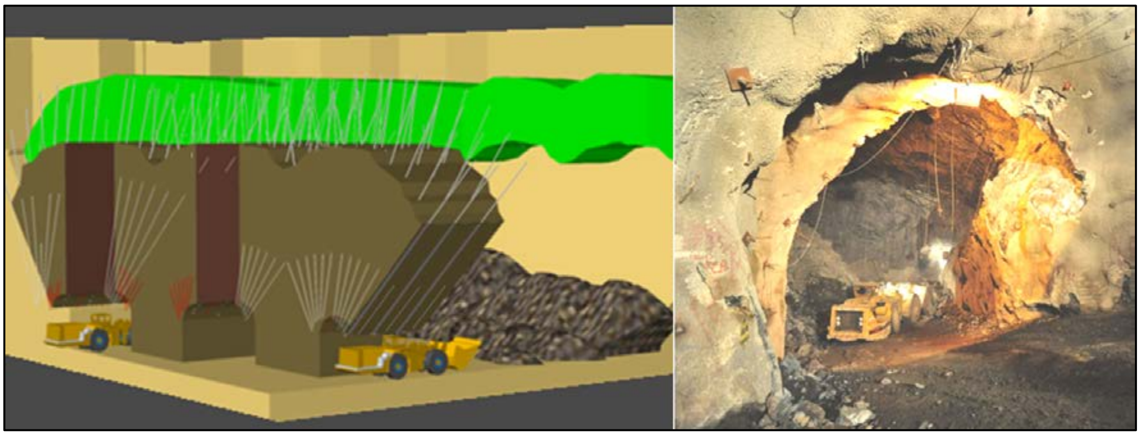
Figure 9 – Open Stope roof camber degree defined by fandrill research section.



9- EXPLOITATION SPEED

When the exploitation is stopped for many consecutive days the drawback zone approximate itself to the production position face. In mining resumption is created a convex zone on the roof unfavorable to stability until the roof planarity be restored. The exploitation time involving the last four blastings (8 meters) is the parameter considered to estimate this effect.

Figure 10 – Open Stope exploitation by LHD with remote control.



10- TIME AFTER CURRENT BLASTING

Soon after the blasting of a fandrill, a very big amount of blocks start to detach on the roof, after occurs a frequency reduction about this detachment "*quiet period*". In the next phase, fissures caused by tensions and vibrations propagate and lead to key blocks provoking additional detachments and formation of new ruptures

Gradually, the roof stability conditions deteriorate as time goes by. It is possible after great rupture volumes the stability be restored with the spontaneous formation of domes. In inferior quality rock masses the ore exploitation should be quick, using a monitoring system such as the CMS it will be possible to precisely estimate the "*quiet period*" to different rock masses classes.

II - DETERMINATION OF INCENTIVE SCORE

This analysis considers the ROM lump ore potential contained in the exploitation fandrill section to determine if the LHD may enter the open stope. If the fandrill section possess a low *Lump* potential it is not worth the risk of use the LHD to exploit this material, because the exposure risk is the same to a high potential fandrill section.

III - DETERMINATION OF TOTAL SCORE

Two scores are analyzed, the operational and technical risk and the incentive score. These two are added and fit to a total score. So a score may have an acceptable punctuation and the other a negative one, which facilitates the decision making to the use or not of the LHD inside the open stope.

IV - RECOMMENDED ACTION

It is the action which determines the guidance of LHD operational use with remote control to the open stope completely, enter the open stope moderately until its rear limit (8,0 to 12,0 meters), enter only by previous technical supervision and no entrance at all.

CASE STUDY: SCORE ANALYSIS TO LHD USE IN N200 LEVEL OF IPUEIRA 6A

Figure 10 – Case study for one stretch of GL201 and GL201A drifts in Ipueira Mine 6A.

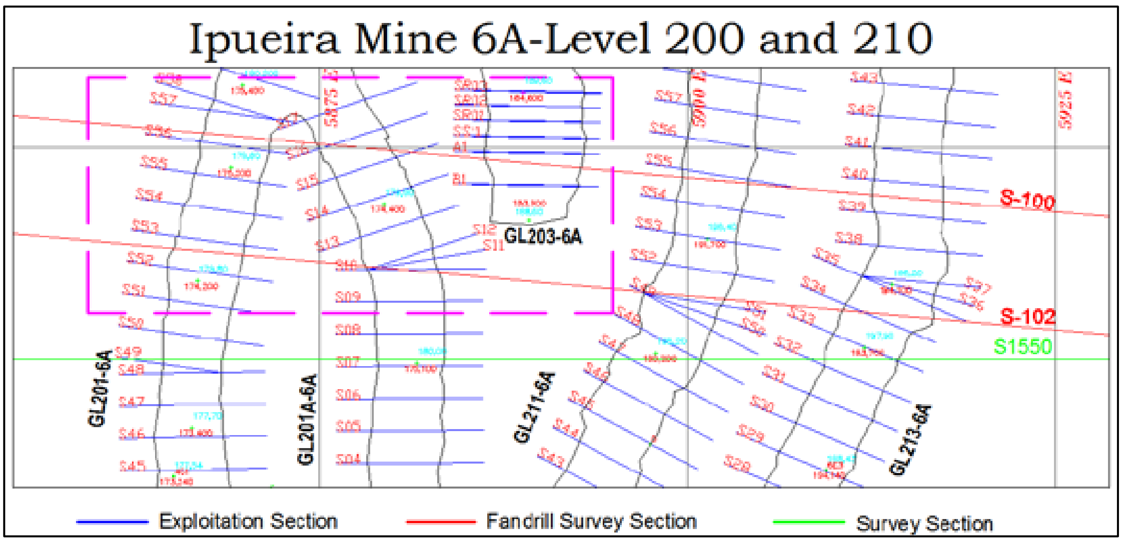


Chart 1 – Determination of Operational and Technical Risk Scores

Risk Evaluation of LHD's During Remotely Controlled Operation Inside the Open Stope						
Stretches Analysis: Section S/1550 - Survey/ Fandrill Section S-100 and S-102.			**Current Version: May/2015**			
Study location: GL201-6A e GL201A-6A						
I – Scores Determination Riscos		**Parameter Analysis**	***Local***	***N210-6A***	***IP-6A***	
Item	**Description**	**Condition**	**Status**	**Points**	***Scor***	
1	Open stope roof rock mass quality and width of 5 to 10 m.	RQD or RMR > =75; Good condition		10		
		RQD or RMR from 50 to 75; or unknown value. Medium condition		5	***5***	
		RQD or RMR < 50; Very bad condition		-10		
2	Hydraulic radius versus rock mass quality (N')	Self-supporting. Good		15		
		Stable with cables. Medium		7,5	***7,5***	
		Collapse. Caving Roof type. Very good		-15		
3	Cabling reinforcement conditions: spacing and length	Spacing/length > 80% recommended, or self-supporting. Good		15		
		Spacing/length from 50% to 80% of recommended. Medium		7,5	***7,5***	
		Spacing/length < 50% of the recommended. Very bad		-15		
4	Observation of the Roof State Detachment	No perceptive detachment (Visual or CMS). Good		15	***15***	
		Moderate detachment. Medium		7,5		
		High detachment/intermittent. Very bad		-15		
5	Proximity of nearby open stopes	Over 50 meters. Good		10	***10***	
		Between 50 and 25 meters. Medium		4		
		Less than 25 meters. Bad		0		
6	Open Stope Depth	Average depth, between 200 and 400 meters. Good		5		
		High depth (from 400 to 500); or low (from 150 to 200). Medium		3,5		
		Very high (greater than 500 m) or too low (lower than 150 m). Bad		0	***0***	
7	Average height of the last 4 last fandrills, including the current one	Lower than 15 m. Good		5		
		Between 15 and 20 m. Medium		2,5	***2,5***	
		Higher than 20 m. Bad		0		
8	Overall Conformation of the roof camber	Arched roof. Good		5		
		Plane roof. Medium		2,5	***2,5***	
		Convex roof. Bad		0		
9	Exploitation Speed	Last four blasts in 5 days. Good		10		
		Last four blasts between 5 and 8 days. Medium		5	***5***	
		Last four blasts after 8 days. Bad		0		
10	Elapsed Time after current blasting	Less than 2 days. Good		10	***10***	
		From 2 to 4 days. Medium		5		
		More than 4 days. Bad		0		
				Total	***65***	
		Maximum Score	**100**			
		Average Score	**50**			
		Minimum Score	**-55**			

Chart 2 - Determination of Incentive Score

Item	Description	Condition	Status	Punctuation	*Score*
1	Remaining ore reserve after non-remote extraction	Reserve >=50 t. Very Attractive		50	
		Reserve between 30 and 50 t. Attractive		40	
		Reserve between 20 and 30 t. Average attractiveness		30	***30***
		Reserve between 15 and 20 t. Uninviting		0	
		Reserve < 15 t. Not attractive		-50	
Incentive Score Value			**Total**		***30***

Chart 3 - Determination of Total Score

Item	Description	*Score*
I	Risk total score	*65*
II	Incentive Score	*30*
Total		***95***

IV - Recommended Action

Total score	Decision making of the LHD remotely controlled use in open stope	Status	*Current*
Higher or equal to 100	Very favorable / Recommended Access		
Between 80 and 100	Favorable / Recommended Access		***X***
Between 80 and 60	Medium / Recommended Access with Supervision		
Between 60 and 40	Unfavorable/ Not Recommended Access, necessary authorization to access		
Less than 40	Very unfavorable / Access Denied		

CONCLUSION

The study demonstrated along these five years, that the geomechanical and geotechnical team of FERBASA are maintaining the program with success, because during this time there was not any considerable risk regarding the physical integrity of the LHD and its operator. The geotechnical description and the treatment given to the open stope stope roof rock mass is very important to enhance the cabling project precision as well as the block detachment risk evaluation during exposing the LHD remotely controlled.

The Potvin's empirical method (1988) was applied in a non-conventional manner to scale out cables spacings and lengths, presenting significant importance, to the hydraulic radius analysis and in the obtained results of cables spacings varying from 2,5 x 2,5 m to 1,8 x 1,8 m, according to the rock mass quality and open stope extent with height of 8 m and 15 m, from the drift stope roof. The cable lengths in open stopes roofs and in their sides may vary between 5,0 m, 7,0 m, 9,0 m until 12,0 m.

The instruments use to rock mass control and monitoring when there is detachments, such CMS and others, shall improve significantly the cabling project and risk evaluation. Some suggestions and new studies are being developed to a better evaluation system, with data input of "3D Terrestrial Laser Scanner RIEGL VZ-400", drifts geotechnical mapping, fandrill survey information and others.

The blasted ore exploitation with LHD's exceeded all expectations, thanks to proximity of the material pile in relation to the drift, stope roof stability and good fragmentation.

The cable system installed to stabilize the open stope roofs and dilution control (Lima *et al.*, 2008) also promoted a general stabilization of the drifts, acting together with the bolts and shotcrete, prevented the appearance of wedges and provided the preservation of the vaults near the open stopes.

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ROCK MASS CHARACTERIZATION THROUGH CORE SURVEYS ANALYSIS IN IPUEIRA 6A MINE LEVELS N200 AND N210 SOUTH PANEL, ANDORINHA-BAHIA

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ROCK MASS CHARACTERIZATION THROUGH CORE SURVEYS ANALYSIS IN IPUEIRA 6A MINE LEVELS N200 AND N210 SOUTH PANEL, ANDORINHA-BAHIA

ABSTRACT

The main purpose of this work was to establish a geotechnical classification study in Ipueira Mine 6A South Panel in the levels 200 and 210 belonging to the group Cia Ferro Ligas da Bahia - Ferbasa. The survey was conducted using as a basis the rock mass classification systems, RMR (Rock Mass Rating) created by Bieniawski (1989) and RQD (Rock Quality Designation) proposed by Deere et al. (1967). The idealization of the study was to determine the rock mass characteristics located in the ore body as well as the upper and lower regions, limited to 20 m above and below. The results allowed to previously estimate the kind of treatment and containment necessary to the rock mass for the development of galleries and establish containment parameters for the open stoping during the extraction phase.

KEYWORDS

Rock Mechanics, RMR, RQD

INTRODUCTION

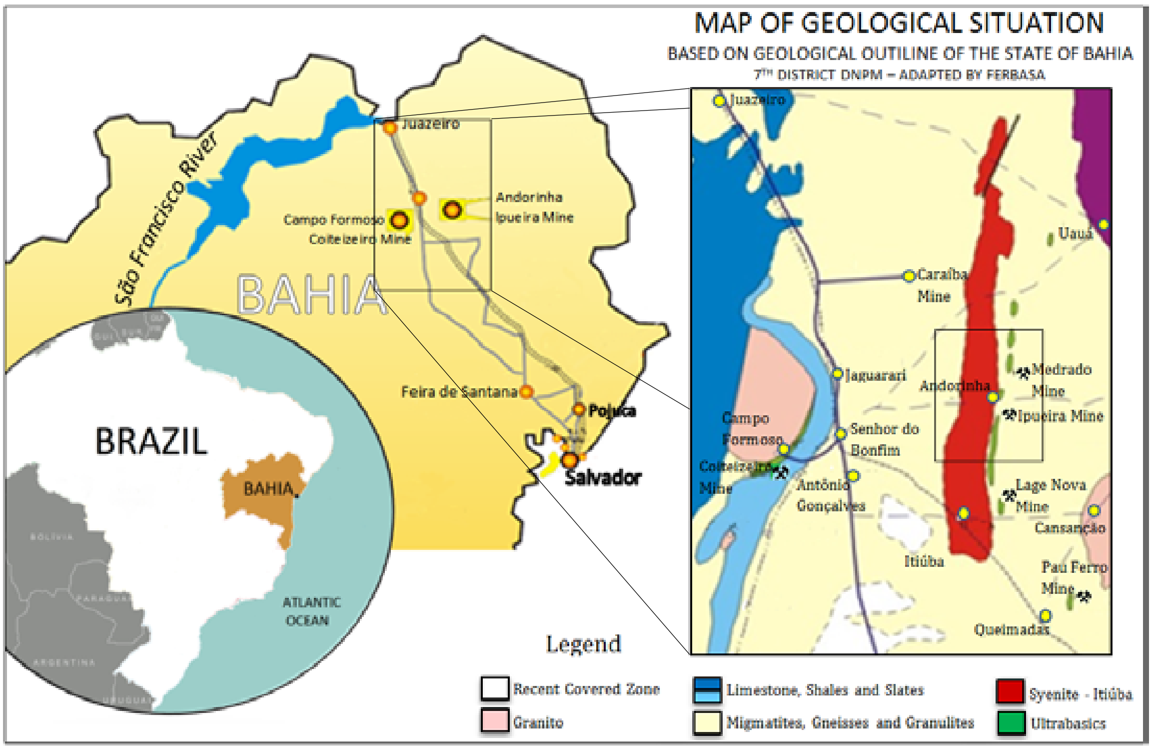
During the initial steps of an underground excavation project, which consists of the viability stage and projects design, in many cases occurs an absence of detailed information concerning the rock mass, stresses and hidrological characteristics. Pointing this out, the use of rock mass classification systems may be considered serviceable. On the other hand, one or more rock mass classification systems may be used as well to build composition and rock mass characteristics image, providing initial estimations of containment methods and allows to appraise the rock resistance properties. (Hoek, 2007).

The methodology developed to this work was defined by Souto (2013), at the South panel of Ipueira 6 at the levels N200 and N210, which allowed by some criteria the characterization and classification studies of rock masses, analyse survey core samples in regions next to the projected galleries, at the top, in and at the bottom of the ore body. These samples were collected, analysed, measured, and tested by point load resistance on core samples, to estimate the treatment efficiency and the containment of the analyzed levels, in order to plan and prepare the necessary actions to medium and long-term.

STUDY FIELD ASPECTS

The Ipueira Mine, where this reasearch was conducted, belongs to the FERBASA Group, located at the North-Northeast of Bahia state, Andorinha city – Brazil. By the highway network from Salvador until Ipueira Mine, final destination, totalizes 447 km, on coordinates 39º45’56" West longitude and 10º21’51" South latitude. The Medrado and Ipueira Mines are part of a chromitiferous district of Jacurici valley and near this Sill can be found many bodies of mafic-ultramafic of Jacurici valley with 100 km in extension from a North-South track, mineralized in chrome. They are embedded bodies in granulitic and gneissic-migmatitic rocks of the São Francisco Cráton basement, from little to medium extensions, creating the so-called chromitiferous district of Jacurici River Valley (Silva, 1998).

Figure 1 – Location of Ipueira Mine and Regional Geological Draft.

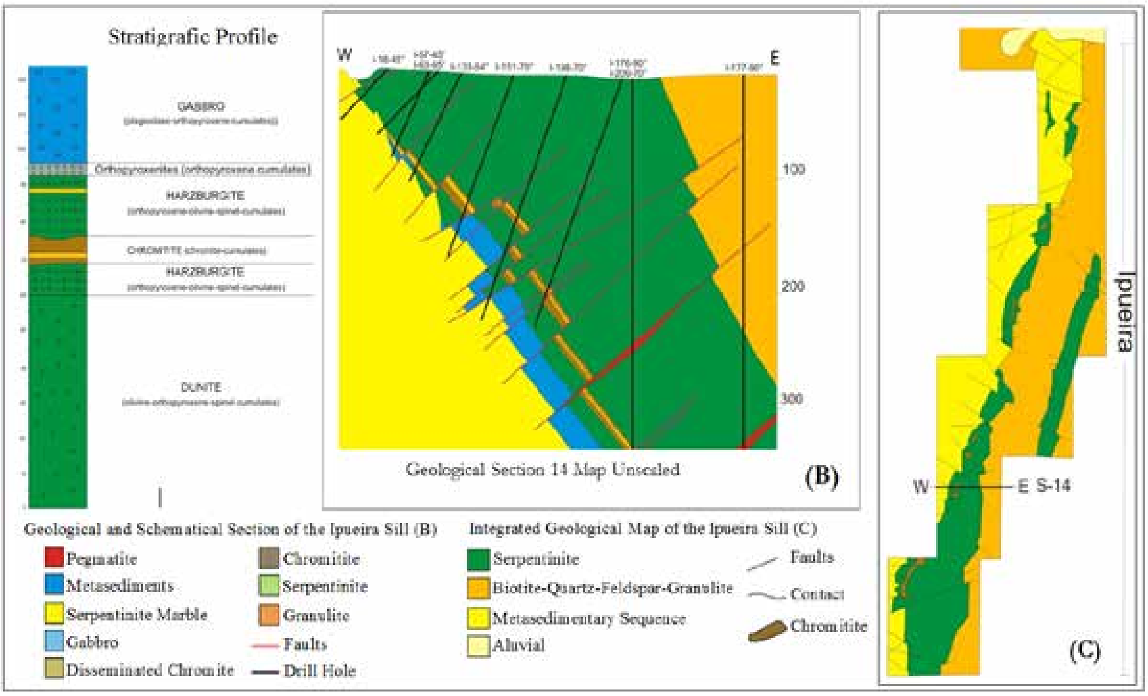


Source: Internal archives modified from Ipueira Mine and Geology Division/ FERBASA (2013).

The Sill is positioned among marbles, at the stratigraphic top, and granulites on the base. In the region of the studied mines this stratigraphic sequence is found inverted, with the marbles positioned at the basement and granulites on top.

According to Marinho *et al*. (1986) the following units are present in the ultramafic sill (Figure 2): plagioclase-orthopyroxene cumulates (29 m); orthopyroxene-espinel cumulates (2 m); orthopyroxene-olivine-espinel cumulates (33 m); chromite cumulates (7 m); and olivine-orthopyroxene-espinel cumulates (60 m). A complex discontinuity system crosses all lithologies (Marinho et al., 1986). This failures intercut the ore body, forming blocks commonly varying in width from 2 to 20 meters as transversal as longitudinal ways. In relation to the ore what predominates is a massive chromite lump type, with 30 to 40% of Cr_2O_3, representing about 80% from the ore, and can be found the disseminated ore type as well (Mello et al, 1986). The mineral produced is the chromite, which the commercialized products are *lump* (fraction above 2 ½ inches) and the chromite sand (fine fraction).

Figure 2 – Geological map integrated Medrado-Ipueira.



Source: Internal archives modified from Geology Division/ FERBASA (2013).

The exploitation works today reach 500 m deep. All Ipueira mines are interconnected. Each mine has a main ramp which access to many mineralized levels. The access to the South and North exploitation Panels in each sublevel is made by access ramps opened transversally West-East way.

EXPLORATION METHODOLOGY

Sampling

The location of this study was in the Ipueira 6A South Panel at the levels N200 and N210. Thus, 23 surface surveys holes were chosen at the sections S/1.425 until the section S/1.625, totalizing 1.061,75 m of core surveys analyzed, see Table 1. These survey holes were initiated at the surface until the lower horizons, placed in exploration sections in East-West way, spacing 18 m from one another and a burden of 25 m approximately. The drilling was done by diamond rotative survey with a Christensen CS-14 Atlas Copco type, able of drilling up to 1.200 m. The samples were organized in benchs and distributed in advance ascending order. The core surveys diameters studied were NQ and BQ of approximately 46,7 mm and 36,5 mm, respectively.

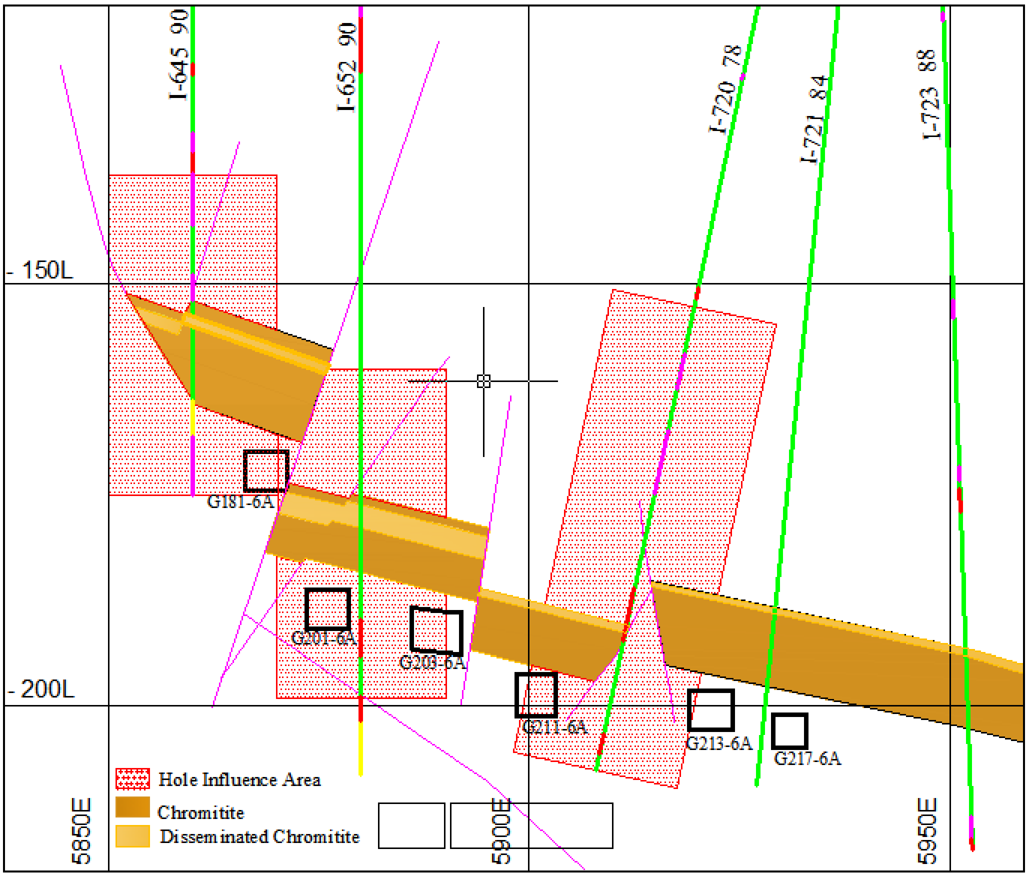
Table 1 – List of analyzed holes to study N200 and N210 from Ipueira 6A levels.

Hole	Tilt Angle	Section	From (m)	To (m)	Length (m)	Diameter
I-652	90°	S/1425	128,15	172,50	51,35	NQ
I-720	78°W	S/1425	139,70	183,35	43,65	BQ
I-721	84°W	S/1425	139,40	181,85	42,45	BQ
I-723	88°E	S/1425	143,85	187,60	43,75	BQ
I-646	90°	S/1450	126,80	162,05	35,25	NQ
I-688	90°	S/1450	136,45	175,40	38,95	NQ
I-727	90°	S/1450	145,40	201,65	56,25	BQ
I-647	90°	S/1475	127,30	175,75	48,45	NQ

I-732	77°W	S/1475	143,40	197,20	53,80	BQ
I-736	88°W	S/1475	151,55	206,45	54,90	BQ
I-649	90°	S/1525	123,85	169,90	46,05	NQ
I-695	90°	S/1525	136,85	181,75	44,90	BQ
I-747	82°W	S/1525	178,30	230,85	52,55	BQ
I-650	90°	S/1550	122,35	125,45	38,30	NQ
I-651	82°W	S/1550	119,25	157,05	37,80	NQ
I-755	65°E	S/1550	153,25	215,20	61,95	BQ
I-653	90°	S/1575	113,75	147,55	33,80	NQ
I-696	90°	S/1575	131,40	176,00	44,60	BQ
I-762	90°	S/1575	175,80	219,55	43,75	BQ
I-654	90°	S/1600	123,90	169,70	45,80	NQ
I-699	90°	S/1600	139,40	193,25	53,85	BQ
I-655	90°	S/1625	123,00	163,80	40,80	NQ
I-697	90°	S/1625	176,10	224,90	48,80	BQ

Observing the structural characteristics of the characterized rock mass lithology, it was decided to analyze only the drill core between 20,0 m above and 20,0 m under the ore, near the regions of the galleries in the aforementioned levels, as demonstrated in Figure 3. Some holes presented lengths under 20 m, at the layer under the ore body due to the exploration information necessity, so, it was considered only the known value.

Figure 3 – Analysis region and hole influence area.



Source: Internal archives from the Geology and Geotechnical Division/ FERBASA (2013).

The holes survey had as main criteria the preference by vertical and subvertical holes, because their stresses due to the deepth are higher, the holes which were near the drive galleries at the N200 and N210 were chosen as well. In reason of unavailable surveys holes to study the section S/1.507 could not be sampled.

Input Materials

To this study were utilized materials such as: field spreadsheet, point load test portable equipment - PLT, measuring tape and caliper, geological hammer, Notebook with Microsoft Excel and AutoCAD softwares.

Data Processing and Considerations

<u>Selection of the galleries to be characterized:</u> The galleries were analyzed in ground plan, located at the South Panel at the N200 and N210 levels. Doing so, it was stablished to characterize the drive galleries: GL 201, GL 201A, GL 203, GL 211, GL 213 and two galleries still in Project phase, GL 215 and GL 217.

<u>Holes location at the ground plan:</u> To each exploration section, a transversal view was done to visualize the position of each hole in realtion to the galleries. From these data with the Autodesk AutoCad 2009 aid, the holes were plotted at the ground plan view with their influence area.

<u>Geomechanical/Geotechnical influence area:</u> FERBASA's Geology Sector adopts to its surveying works an square influence net (20 m x 25 m). Thus, based on this methodology an equivalent influence area was stablished to estimate the rock mass geothecnical and geomechanical conditions in a given region from the collected core surveys holes information.

<u>Geomechanical and geotechnical parameters calculation:</u> After the field work and with the necessary data put at the spreadsheet, the Microsoft Excel 2010 software was used to determine the RMR, RQD and other indexes, calculating all of them.

Procedures to RMR Determination

A drilling advance is understandable as the distance that the drilling equipment advanced inside the rock mass, until it was necessary to interrupt the operation to withdraw the rock sample (core survey). The procedures to obtain the five parameters are shown next (Table 2).

Table 2 – RMR Classification System (modified - Bieniawski, 1989).

A. CLASSIFICATION PARAMETERS AND ITS WEIGHTS									
Parameter			**Values Range**						
1	**Undamaged Rock Resistance (MPa)**	Point Load Index	>10	4-10	2-4	1-2	Lower values see compressive strength stress tests		
		Unixial Stress Resistance	>250	100-250	50-100	25-50	5-25	1-5	<1
	Weight		***15***	***12***	***7***	***4***	***2***	***1***	***0***
2	**RQD (%)**		90-100	75-90	50-75	25-50	<25		
	Weight		**20**	**17**	**13**	**8**	**3**		
3	**Discontinuity spacing**		>2m	0,6-2m	0,2-0,6m	60-200mm	<60mm		
	Weight		***20***	***15***	***10***	***8***	***5***		
4	**Discontinuity condition (see Table E)**		Highly rough surface, with no alteration, closed and hard wall.	Slightly rough surface and with low alterations, opening <1mm. Hard wall	Slightly rough surface and much altered, opening <1mm. Soft wall	Grooved surface or filling width <5mm or opened joint of 1-5mm	Filling width with clay material (soft) >5mm or opened joint >5mm		
	Weight		***30***	***25***	***20***	***10***	***0***		
5	**Undergroung water action**	Infiltration flow by 10 m tunnel (l/m)	Nill	<10	10-25	25-125	>125		
		(Water pressure in the joint) / (major main joint)	0	<0,1	0,1-0,2	0,2-0,5	>0,5		

		General conditions	Completely dry	humid	wet	dripping	Rich flow
		Weight	***15***	***10***	***7***	***4***	***0***

- Point Load Tests Procedures (Figure 4)

The tests were performed with an Enerpac/Telemac model JHA-73 equipment, 2 tests per drilling advance were determined, and only 3 tests when presenting differences among the results to have close values. To two or more lithologies present in the manouvers, it was determined to register the dominant lithology (higher length in the drilling advance) as the unique in the interval.

Figure 4 – Examples of point load tests.

Source: Internal archives from the Geomechanical Division/ FERBASA (2013).

- RQD Determination Procedures

RQD calculation was done from the equation below (Deere, 1989), which consists in adding the core survey pieces of a higher length than 10 cm inside a drilling advance and divide the result from this addiction by the length of the manouver/advance based on the classic analysis.

$$RQD = \frac{\Sigma \, Length\ of\ core\ pieces > 10\, cm}{Total\ length\ of\ core} \times 100$$

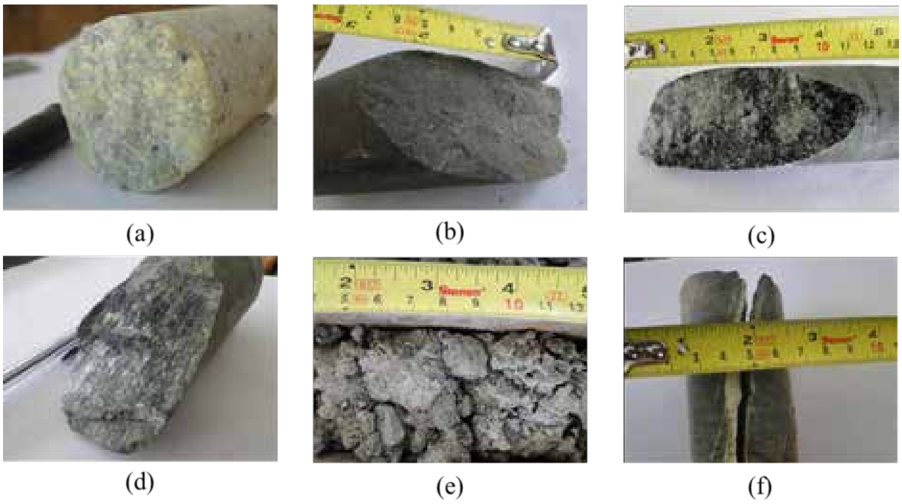
- Discontinuity Average Spacing Estimation Procedures

In each core survey drilling advance must be observed the natural fracture present in core surveys measuring and classifying for each interval. The average spacing calculation among fractures in meters is done by dividing the length of each gap in meters by the numbers of fractures. Each result is attributed a weight according to Table 2. In fault zones, with a greater number of fractures, it was given minimum rating to the correspondent average spacing parameter among fractures.

- Discontinuity Condition Estimation Procedures

Five discontinuity characteristics must be analyzed and assign a weight, after add them verify Table 2 to choose a gap in a given class and consider (Brown, 1981): walls discontinuity roughness; discontinuity's aperture; discontinuity wall alterations; and the discontinuity walls fillings with low cohesion material (Clay clay, highly altered rock, sand or silt). These parameters indexes are 30, 25, 20, 10 and 0, in Figure 5 are shown a few examples of discontinuities and their respective indexes.

Figure 5 – Discontinuity plans examples.



Source: Internal archives from the Geomechanical Division/ FERBASA (2013).

In Figure 5, the index "25" [item (a)] is related to rogh discontinuities, slightly altered and a filling minor than 1 milimeter. While the index "20" [item (b)] differs from the previous one for presenting low roughness and be moderately altered. The index "10" [item (c) and (d)] is adopted to the cases where be evident polished/grooved/smooth surface fractures with a filling lower than 5 milimeters or strong alteration presence. The index "0" [item (e) and (f)] is associated to the presence of a filling higher than 5 mm or highly altered rock.

- Determination of Water Condition in Rock Mass Procedure

For the present work study area, attempting to simplify the study it was decided to attribute a pattern index to all RMR calculations. According to Lima (2011), through visiting to undergroung mines He could stablish that in major locations prevail a "dry" condition (index 15), "humid" (index 10) and in a few specific ones "wet" (index 7). So, it seems reasonable to consider temporarily the value 10 to general water condition, what can be reviewed after reports consulting about the mine hydrogeology.

- Discontinuities Orientation Determination Procedures

In general, such parameter is not used in the calculation due to the difficulty to be determined in studies through core surveys, in zones with many discontinuity plans variations and in áreas where still there is no development activities, what is common once the RMR is a classification system frequently applied in project phases. Nevertheless, in this reasearch, some galleries are developed. Considering this, many visits were realized to the Ipueira 6A N200 and N210 galleries, and could be observed the discontinuities direction in relation to the drift axis, the way in relation to the drift advance and the dip. 101 observations were performed, the visiting field results is in Table 3.

Table 3 – Determination of discontinuities direction

Discontinuity direction and way	Dip	Observation Numbers
Perpendicular direction to the tunnel axis – Tunnel opening in dip way	45°-90°	17
	20°-45°	6
Perpendicular direction to the tunnel axis – Tunnel opening in inverse dip way	45°-90°	21
	20°-45°	1
Parallel direction to the tunnel axis	45°-90°	42
	20°-45°	13

No relation to direction	0°-20°	1

Assuming a weight to each observation (refering to “tunnels”), with a weighted average of the correction factor equals to “- 6,94”, to the RMR calculation it was wise to use na index equals to “- 5” as a correction factor.

Gathering and Organization of Data Analysis

The results obtained were organized by hole and section to make available information that allowed to classify and characterize the South Panel levels N200 and N210, the higher region to the ore body and the production zone (corresponding to the values weighted average). These data were organized in spreadsheet as a table, as shown in Table 4. In Table 4, average values of RMR, RQD, Uniaxial Compressive Strength (MPa), drilling lengths, by hole and by section, quantified and with their respective evaluation were obatained.

Table 4 – Example of data obtainment about rock mass quality.

HIGHER ROCK MASS IN RELATION TO THE ORE BODY

Section	Hole	RMR	RQD (%)	Uniaxial Compressive Strength	Advance (m)	Evaluation (RMR)	Evaluation (RQD)
S/1425	I-652-90°	42,4	36,0	96,1	21,3	Regular rock	Poor rock
	I-720-78°	43,4	45,8	96,5	22,0	Regular rock	Poor rock
	I-721-84°	34,8	38,5	89,0	22,3	Poor rock	Poor rock
	I-723-88°	48,1	59,6	127,7	20,8	Regular rock	Regular rock
	Section	42,1	44,8	102,0	86,3	Regular rock	Poor rock
S/1450	I-646-90º	49,6	50,8	159,4	19,3	Regular rock	Regular rock
	I-688-90º	45,4	40,5	133,0	21,6	Regular rock	Poor rock
	I-727-90°	43,6	32,2	126,0	18,3	Regular rock	Poor rock
	Section	46,2	41,3	139,4	59,1	Regular rock	Poor rock

RESULTS AND DISCUSSIONS

The information presented to rock mass qualification and characterization of Ipueira 6A levels N200 and N210 were the RMR (*Rock Mass Rating*) and the RQD (*Rock Quality Designation*). To a better rock mass quality evaluation a label was created, which unifies some classes determined by RQD and RMR, see Figure 6, from their analogy, a simplified classification was created, where if a given rock mass with 50 value in the RMR index would be classified as “regular” and if the RQD is 40 it would be classified as “poor”, unifying the classes as suggested, the new classification will be as “regular to poor”. With this it was established a label, shown at the left side from the Figure 7 and 8, that identifies the rock mass quality by a color scale, such subtitle was applied to each map presented in this work.

Classification of Levels N200 and N210 South Panel of Ipueira 6 Mine

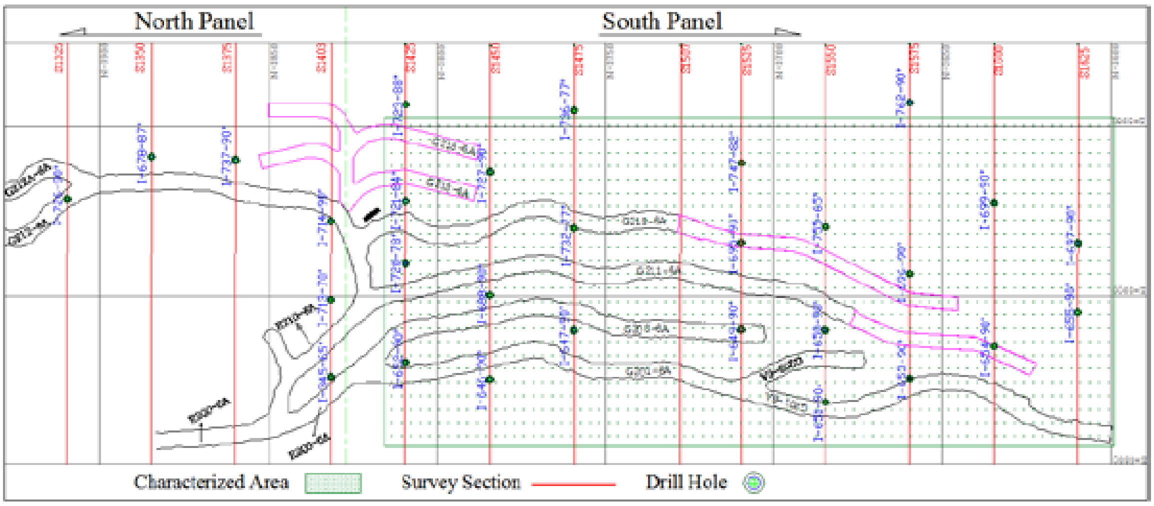
Table 5 presents the general average to the Ipueira 6 South Panel N200 and N210 levels, indicating a regular rock mass what qualifies the panel as good to development and exploitation. In general it is possible to assure that the studied region presents a good efficiency with a systematic containtion with shotcrete and steel cables during the level development. At the exploitation phase with this type of rock mass above the ore, it is possible to have ROM exploitation with a waste/ore relation with no great dilutions.

Table 5 – South panel characterization and classification.

RMR	RQD (%)	Uniaxial Compressive Strength (MPa)	Advance (m)	Evaluation (RMR)	Evaluation (RQD)
51,3	56,2	150,4	1,061.75	Regular Rock	Regular Rock

The Figure 6 below illustrates the South Panel below the crosshatched area in green color and on the left, the beggining of the North Panel. The galleries in violet color refer to sectors in project phase.

Figure 6 – South Panel levels N200 and N210 6A.



Source: Internal archives from the Geology and Geotechnical Division/ FERBASA (2013).

Superior Rock Mass in Relation to Ore Classification

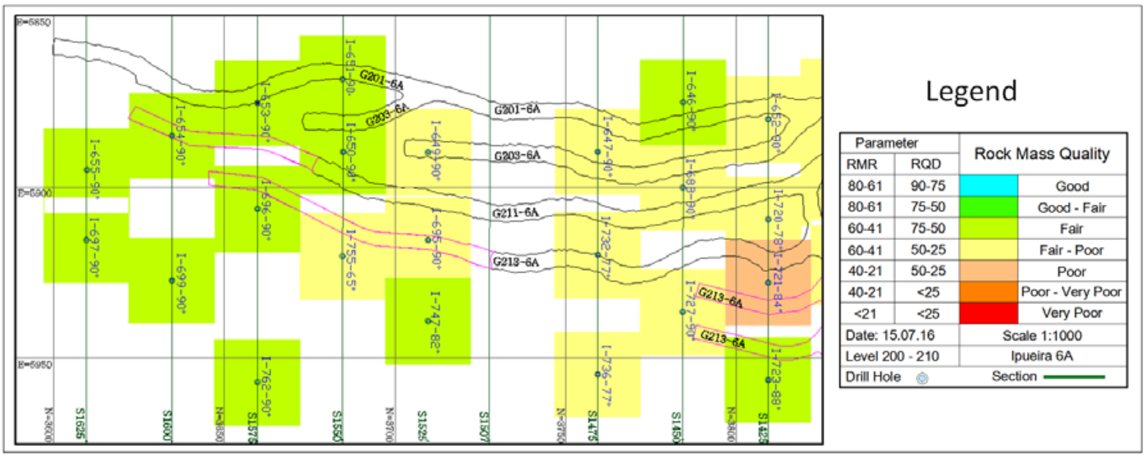
Based on the data presented in Table 6 and visualized in Figure 7, which shows the rock mass quality at the superior region in relation to the ore. The Table 6 has a summary of the rock mass characteristics to each exploration section. Joining both knowledges theoretical and practical of rock mass classifications by geomechanical/geotechnical groups of FERBASA, it is possible to assure that a "regular" rock mass has in average a medium to high efficiency at the rock mass stability during the exploitation phase of medium and low term, while a "regular to poor" rock mass presents a medium to low efficiency at the same expoitation phase. It is important to stress this rock mass containment and stability can improve with the installation of steel cables (5,5 m e 7,0 m), in the wall rock above the ore body.

In Table 6 and Figure 7, it is possible to divide the South Panel in two regions with distinct properties, in which, S/1.425 until S/1525 sections indicate a "regular to poor" rock mass, while S/1.550 until S/1.625 sections the rock mass is "regular" allowing to prevail the high efficiency containment of the rock mass and low waste/ore dilution in the ROM. The rock mass shows improvement in its geomechanical/geotechnical properties in North-South way, evidencing that the RMR remains into the "regular" strip and the RQD raises until the rock mass classification changes, and the Uniaxial Compressive Strength also raises in North-South way. Indicating a rock mass less fractured and more resistant.

Table 6 – Higher rock mass in relation to the ore body characterization and classification.

Section	RMR	RQD (%)	Uniaxial Compressive Strength (MPa)	Advance (m)	Evaluation (RMR)	Evaluation (RQD)
S/1425	42,1	44,8	102	86,3	Regular rock	Poor rock
S/1450	46,2	41,3	139	59,1	Regular rock	Poor rock
S/1475	48,1	38,9	140	59,4	Regular rock	Poor rock
S/1525	47,0	43,7	144	60,1	Regular rock	Poor rock
S/1550	51,3	55,5	150	59,2	Regular rock	Regular rock
S/1575	52,9	60,8	151	63,7	Regular rock	Regular rock
S/1600	51,1	60,2	150	41,6	Regular rock	Regular rock
S/1625	53,5	67,4	154	41,4	Regular rock	Regular rock

Figure 7 – Rock mass higher in relation to the ore body.



Source: Internal archives from the Geology and Geotechnical Division/ FERBASA (2013).

Rock mass classification in production stage

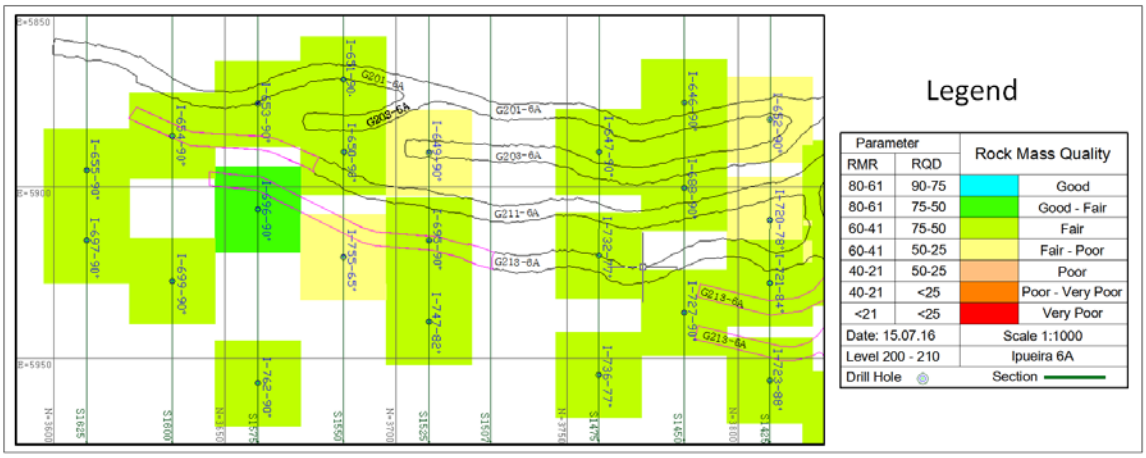
This classification combines the information inside the interval in the ore body and in the 20 m below the ore body, in order to complete, the geomechanical and geotechnical information as to the galleries development in the ore body as in the waste rock and the exploitation phase, naming so production zone.

Table 7 – Rock mass in production zone characterization and classification.

Section	**RMR**	**RQD (%)**	**Uniaxial Compressive Strength (MPa)**	**Advance (m)**	**Evaluation (RMR)**	**Evaluation (RQD)**
S/1425	48,6	50,6	136	94,90	Regular rock	Regular rock
S/1450	51,2	57,8	154	71,35	Regular rock	Regular rock
S/1475	51,9	60,6	158	97,75	Regular rock	Regular rock
S/1525	54,2	59,6	167	96,70	Regular rock	Regular rock
S/1550	51,1	53,3	159	78,90	Regular rock	Regular rock

S/1575	56,2	67,8	173	58,45	Regular rock	Regular rock
S/1600	48,5	62,0	140	58,10	Regular rock	Regular rock
S/1625	55,8	68,5	161	48,20	Regular rock	Regular rock

Figure 8 – Production zone (Influence area).



Source: Internal archives from the Geology and Geotechnical Division/ FERBASA (2013).

Considering the data in Table 7, that exposes the rock mass quality at the production zone, and based in theoretical and practical knowledge of the geomechanical/geotechnical FERBASA group, it can be assumed that a “good to regular” rock mass is very favourable to development and exploitation, and a “regular” and “regular to poor” rock mass is less favourable to the development and exploitation. As shown in Table 7 and Figure 8, the South Panel demonstrates itself favourable to development and exploitation, once all sections had their rock mass classified as “regular”.

There is evidence that a few sectors need special treatments in respect of containment during the development and exploitation phases. The RMR remained Constant in “regular” rock class and the RQD raised gradually in North-South way. The rock resistance to Uniaxial Compressive Strength did not present relation to the rock mass quality improvement.

CONCLUSION

The data analysis obtained compared the technical viability to the development and exploitation of Ipueira 6A Mine South Panel, at the point of view of geomechanical and geotechnical study of levels N200 and N210. It could be observed that despite the ore body is fit in the “orthopyroxene-olivine cumulate and olivine-orthopyroxene cumulate” and analyzing the presented data, it is evidente that the rock massa above the ore body is, in general, of poor quality located at the production zone, in and below the ore body.

Previous studies suggests that is possible to prevail the containment treatment type to the rock mass, with shotcrete and steel cables, before the development. And at the exploitation phase, to prevail stretchs with more or less dilution of waste/ore in the ROM, allowing to estimate the efficiency of the steel cables used at the roof of the stope based on the rock mass quality, as a temporarily containment method to control the ROM dilution.

The study had a great importance, once it enabled the analysis and estimation of stretchs with possible geotechnical problems, providing support data to future containment studies of the studied áreas, however, it suggests to correlate the RMR system with the Q system, being able to compare afterwards with the Q value obtained *in* loco at the drift through the present discontinuity analysis. And obtain detailed exploration information using the Fandrill with a top hammer system to add geotechnical information athe the mineralized production Panel.

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SCORPIO N – REMOTELY CONTROLLED MOBILE ROBOT AND THE POSSIBILITY OF COST SAVINGS AND PROVIDING SECURITY

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SCORPIO N - REMOTELY CONTROLLED MOBILE ROBOT AND THE POSSIBILITY OF COST SAVINGS AND PROVIDING SECURITY

ABSTRACT

Project Scorpio N is the newest version of unmanned martian rover allowing to conduct researches,as well as collect samples and transport the samples. Scorpio N was created with an idea of exploration geology, mining geology and ore, lode or deposit exploitation in tough Earth and extraterrestrial conditions. Despite of tasks mentioned above, rover also enables real-time data transmission, performing laser scanning and space model of scanned object. It could be used as an support in work for mining engineers, geologists and geodetic surveyors. Mechanically construction could be split into three main components – driving module, robotic arm and measurement unit for soil sample examination. Driving module creates a base of whole rover, while other components are modular and can work independently. Scorpio's driving module is a 4-wheel modificated version of rocker-bogie suspension used for ex. in NASA rovers. Thanks to differential bar dependence of suspension and equivalent load distribution for each wheel is provided. Specially designed tires made from Cordura® material filled with styrophone granules co-operate with suspension and provide traction and damping efficiency. A key component for rover's functionality is the robotic arm. It has 5 degrees-of-freedom what in connection with differential steering of vehicle assures great operational possibilities. In measurement unit, soil sample could be tested for expected parameters and data could be passed to the operator. Currently works are focused on installing laser scanner, which could be used for rapid delineation of XYZ coordinates thanks to laser measurement. Creating "point cloud" allows to generate three dimension model of scanned object. Scorpion N is remotely controlled mobile robot used for research purposes, but also industry purposes are available. Thanks to adequate construction solutions Scorpio rover possess high terrain efficiency and due to robotic arm the object manipulation ability. Project also regards to innovations at space engineering in Poland and around the world.

KEYWORDS

martian rover, exploration geology, mining geology, prospecting lode and ore, intelligent mine, rocker-bogie suspension, robotic arm, gripper, differential steering, Cordura tires, laser scanner

INTRODUCTION

The Scorpio 4 project is a new approach to the subject of human-operated mobile scientific platforms, called rovers. Project Scorpio has been established in 2011 and since then it is constantly developing and optimizing their constructions. From the basis Scorpio 4 is planned to be a semi-autonomic human-operated vehicle, which means that the team had to verify most of the assumptions connected to professional constructions of rovers. This point of view made the team think about building a rover which could be more ergonomic and easily adaptable but still being able to perform some complicated science or human-assistance related tasks. The newest version of Scorpio N had been built as a result of knowledge and experience combination. Project Scorpio rovers are taking top positions in international competitions in which they participate. In 2013 Scorpio 3 took second place at University Rover Challenge [2] in Hanksville in United States of America. In 2014 Scorpio 4 won European Rover Challenge [3] in Poland while Scorpio N took third place in University Rover Challenge 2015. Scorpio 4 is older generation of Scorpio N

SCORPIO'S MECHANICS

Mechanically the construction could be split into 4 main subassemblies – Driving Module, Electronic Bay, Robotic Arm with Gripper and Ground Drill. Driving Module together with Electronic Bay are making up the base of the rover while other subassemblies are modular and are able to work independently. That makes Scorpio rover universal mobile scientific platform.

DM – Driving Module

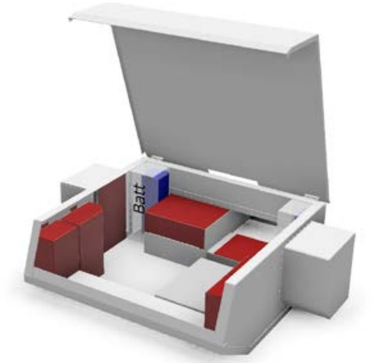
The Scorpio 4 rover Driving Module is based on rocker-bogie type system widely known and used for ex. in NASA rovers operating on Mars[4]. The system was modified to match the competition requirements and project statements. To match weight limits and simplify the suspension, the rover is designed to use only four wheels[1]. The differential suspension is a wheel-to-frame connection system providing absolute dependence between wheels. Unlike the most popular spring-supported suspension systems the differential suspension relies only on a geometrical load distribution providing each wheel relatively equal normal force during regular driving. The system used in Scorpio N construction consists of two longitudinal rockers connected centrally to Main Rotational Joint, which is a main axis supporting the whole body. Due to the fact that the vehicle center of mass is unlikely to be situated along the axis, there would have been unbalanced momentum of force causing the body to flip if no additional point of support was added. As the momentum needs to be balanced, the Differential Bar System is added to support the body in the second axis. The differential bar consists of two rods connected to Rocker Pushers on one side and to the bar on the other. The bar is connected to the frame centrally using a slide bearing, which allows it to rotate. During driving, the stem gives an advantage of reducing the body pitch angle (to half the angle of rocker pitch as the wheels drive over an obstacle) and distributing the wheel-related forces to the whole suspension system. The rover is supposed to operate at higher speed range than assumed in the original design of rocker-bogie type of suspension. The special attention has to be given to the energy distribution and dispersion provided by its components. The best way of improving energy dispersion of the suspension was to design a way to heighten the damping coefficient of the connection between the ground and motors being the heaviest elements of the suspension. That means that special attention had to be given either to wheel rim or tire design [6]. After few tests the idea of high-damping tires made of Cordura® fabric and filled with styrofoam granulate were developed. Tires completed the idea of rocker-bogie 4-wheel suspension

construction.

Figure 1 – Driving Module of Scorpio N

MEB – Main Electronic Bay

Main electronic bay had been designed in purpose of locating all of the electronic devices needed for



basic rover's functions. Thanks to the components used it is dustproof and waterproof (IP54). MEB is made of aluminum and due to it's shape and specially emitted places for each subassembly it became a radiator and is passively cooling whole construction. After desert tests it appeared to be very effective.

Figure 2 – Main Electronic Bay of Scorpio N

RA – Robotic Arm

The key component to the functionality of our vehicle is a manipulator with a gripper. The manipulator has 5 degrees of freedom, which combined with differential steering of the rover enables vast operating abilities. Rotation at the base of the manipulator creates an optional 6th degree of freedom. Application of aluminum alloy, the same that is used in aviation industry, reduces weight, at the same time guaranteeing satisfying endurance and stiffness of the robotic arm. Optical encoders installed at each axis increase precision of movement of the manipulator. Every undesired or unexpected movement is countered by motors or actuators. Further improvement was achieved by using inversed kinematics, due to which the gripper moves in linear segments instead of arches. Specially designed gripper, in which a screw transmits turning motion into linear motion, is capable of holding and grasping objects of different shape. Proper stiffness assures huge lifting capacity. Shape of the gripper endings can be modified if needed for further specialization of the construction. In order to avoid any possible damage of the motors the measurement of amperage was also installed. While encountering resistance grasping force can be adjusted, which enables grasping fragile objects.

Figure 3 – Robotic Arm of Scorpio N

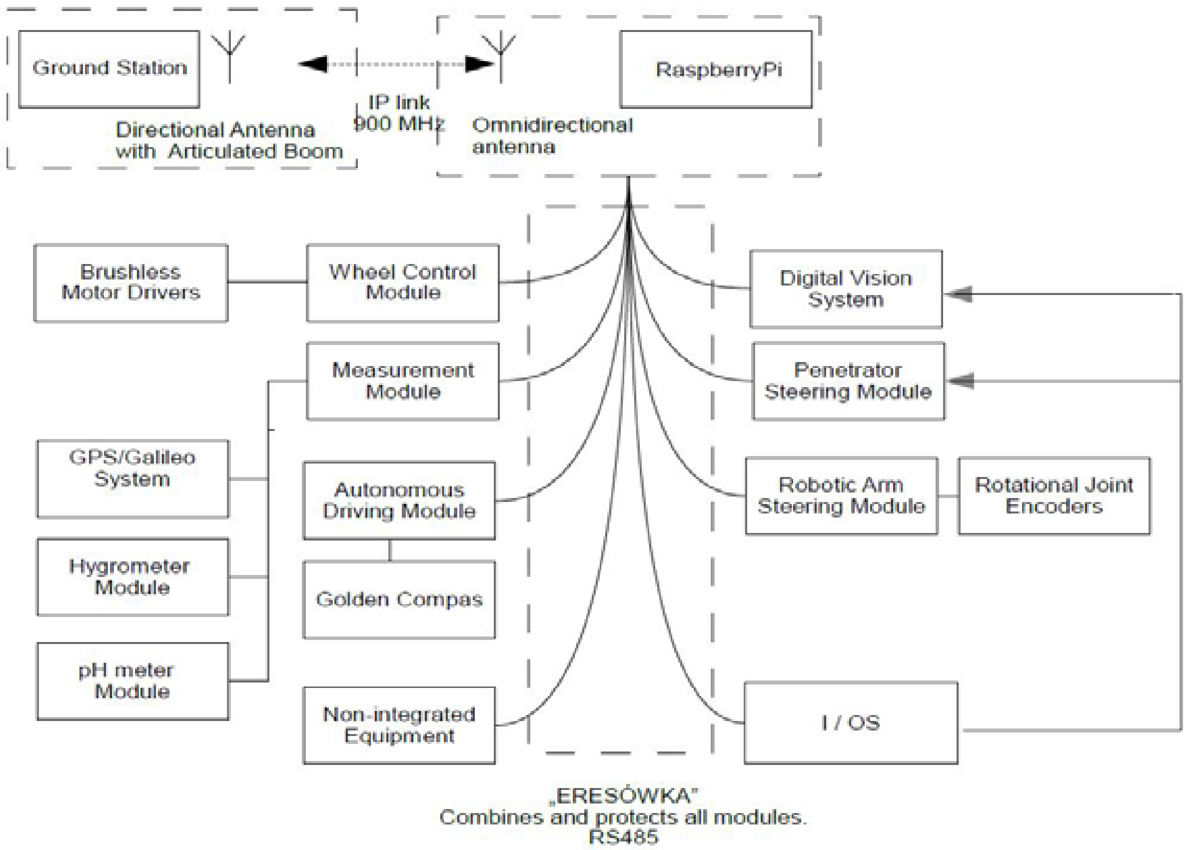
Drilling Unit – DU

Due to the Science Task the drilling module was designed (figure 4). It stands out for its ability to isolate a ground sample from any depth up to 8 cm. It is possible thanks to element (1) coupled by a screw-nut mechanism with a motor so that it can move along the drill. Drilling process can be divided into few stages. At the first one, whole module is lowered until bottom support (2) touches the ground. It is followed by powering next motor (3) driving a screw and drill attached to the screw by a freewheel (4). It means that one motor is responsible of drill's rotation and vertical movement. Due to that, motors number was decreased, so the overall mass is lower, but the drill's pitch (vertical movement with 360 degrees revolution) is determined. During the drilling process, ground is transported vertically through sleeve and and dumped outside it. When desired depth is reached, element 1 is lowered on the sleeve, so that ground can be sampled into the box (5). After required sample mass is achieved, drill is taken out of the ground and then the whole module is lifted. Additional function of this module is the capability to regulate its inclination angle. It allowed to decrease module's dimensions. Moreover, with angle regulation, sample can be removed from the box and in the "ride mode" module can be moved to a position, where it doesn't decrease rover's clearance. During the design process, the most essential aspects where low overall mass and minimum sampling

time of 5 minutes.

Figure 4 – Scorpio N Drilling Unit

SCORPIO N ELECTRONICS



Scorpio N electronics architecture

Figure 5 – Scorpio N electronics architecture diagram

Base-Rover and Autonomous INteraction System (BRAINS)

The system consists of a computer platform which controls several modules located on Scorpio N rover. Scripts controlling rover operation are memorized on a flash drive in a RaspberryPi platform Using an additional outer WiFi card the module sets a network (named 'Scorpio'), to which any capable device can be connected from the base stand. There is no need to download and install any additional software to drive the rover as the whole programme needed is set in the network location. The computer inside the rover, using a USB-to-RS485 converter, controls data transfer in the bus as a 'master' device, what is electrically accomplished using a RS485 Communication Module.

Scorpio N data transmission

Commands as well as vision are transmited via 900 MHz frequency. It provides stable connection and better bandwidth. The system is able to operate on 8 cameras at the same time, which is more than enough for easy maneuvering. Our rover's radio communication is based on transceivers Nanostation Loco M900 made by Ubiquiti firm, running in ISM band on frequencies varying between 902 and 928 MHz. They are capable of working in 5Mhz, 10MHz and 20MHz bandwidth and own an input power regulator in range of 1 to 30dBm.Transceivers also include two internal directional antennas in MIMO technology with 7,5dBi gain and one place for one external antenna. To upgrade we equipped it with antenna switch to provide access for our own antennas with better gain or better radiation characteristic .As a result we can switch between internal and external antennas depending on situation and need. In the base station we used directional antennas, named Yagi, with 10 and 20 elements. They are attached on the rotor which, due to GPS and compass placed both in Scorpio and in base station, is able to direct the antennas straight to the rover . This technology helps to make the link transparent (from the network point of view) for all of the equipment on the rover as well as in the station. Everything mentioned above provides us with possibility to communicate on the distance of dozen or so km and the bandwidth of approximately 50Mb/s. This way we are able to capture smooth, undisturbed vison in great quality from more than one camera at the same time which significantly helps our rover.

Scorpio N application

Project Scorpio N is the newest version of unmanned martian rover allowing to conduct researches as well as, collect and transport samples. Scorpio N was created with an idea of exploration geology, mining geology and ore, lode or deposit exploitation in tough Earth and extraterrestrial conditions. Despite of tasks described above, rover allows also real-time data transmission, performing laser scanning and space model of scanned object. It could be used as a support in work for mining engineers, geologists and geodetic surveyors. Mine rescue or mines rescue is the specialised job of rescuing miners and others who have become trapped or injured in underground mines because of mining accidents, roof falls or floods and disasters such as explosions caused by firedamp. Scorpion N can be useful in such incidents or technical support rescue. Scorpio N can used to rescue miners trapped by various hazards, including fire, explosions, cave-ins, toxic gas, smoke inhalation, and water entering the mine. In addition to the tasks related to research and monitoring in mines and industrial Scorpion N can be sent anywhere man cannot go. Scorpion N is remotely controlled mobile robot used for research purposes, but also industry purposes are available. Due to adequate construction solutions Scorpio rover possess high terrain efficiency and thanks to robotic arm object manipulation ability. Project also refers to innovations at space engineering in Poland and around the world.

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SIMULATION OF PRODUCTIVITY PARAMETERS APPLIED TO ELABORATION OF MINING PLANS

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ABSTRACT

One of the main functions of a monthly mining plan is to define the ROM mass, waste volume and the location of mining polygons. Mining plans applied to open-pit mining method need to adopt hourly productivity parameters for haulage and loading operations. The most common source to provide this data is consulting a historical database. However, the adoption of historical data as a parameter may not accurately represent the productivity capacity of fleet dedicated to implementing the plan. The switching between rainy and dry periods implies adjustments in the operation, which causes oscillations in productivity of haulage and loading operations. This study aims to simulate the main operational parameters through multivariate equations that explain the production cycle of these operations. To obtain the equations were applied techniques and multiple regression based on a database obtained via operational management system of a mine. The simulation model is supported by high correlations among variables. Thus, the simulated indicators reached values very close to the real ones. This adherence and high correlations validate the model. Therefore, the application of this tool ensures high feasibility of monthly mining plans.

KEYWORDS

Monthly mining plan, productivity capacity, simulation

INTRODUCTION

Simulation techniques applied to mining projects are widely used in industry because support operational decisions. On account of the possibility to reach reduced costs and risks, these tools have been developed over the last decades and can be applied to mine planning (Nader et. al, 2012). In most operations that use the open-pit mining method, the mining plan leads the drilling, blasting, loading and haulage. Thus, the mining plan shall set targets compatible with the system capacity. Bozorgebrahimi et al. (2003), claim to be necessary to select a group of key variables able to build a model that explains a particular process. One of the main responses obtained from a simulation model is the performance estimation of a certain process.

Monthly mining plans must provide the locations of mining faces, quality and material quantities. As this mining plan will be implemented by a given fleet of equipment, it is necessary to establish a feasible performance target. The conventional method of establishing productivity targets is a background query. However, these reports can inform only the productivities performed in previous months and do not consider the variability of cycle times, average haulage distance and other operational variables. This practice can threaten the viability of the mining plan because adopts a performance obtained in a different operating condition of the current. Thus, the use of background reports can lead to situations in which the targets are too bold or too simple to be accomplished.

Considering the monthly mining plan as the most important technical guidance for mining operations, it is necessary that the targets are achievable. Rodovalho and Cabral (2014) performed estimations of hourly productivity that consider a database built during one month. The current study aims to generate equations able to estimate the productivity parameters of loading operations and mining haulage of a mine with high adhesion in relation to productivity performed. In this regard, a group of variables that have strong influence on the operations productivity will be evaluated. The studies adopt a database built during four months and were carried in a large open-pit mine in the state of Minas Gerais, Brazil. Using techniques of multiple linear regressions, equations were generated for each load equipment and haulage fleet in operation. A fleet management system was used to record the behavior of the operating variables and build the database. The main contribution of this study is to develop more realistic mining plans. It is also important to discuss the results of a simulation that uses larger database. In addition, the simulation model can be an excellent management tool because it allows the mapping of deviations or process failures (Rodovalho et. al., 2016).

METODOLOGY

The development and implementation of simulation models may require high costs. Even with high proficiency in software applied to the simulation, a meaningful collection of data is needed. This way, the model must generate output data capable of supporting an adequate analysis of the process. This action depends on the identification of variables that influence the process. The method used to obtain the equations that explain the processes studied and compose the simulation model is the multivariate linear regression. The use of this tool in the development of monthly mining plans represents a novelty able to increase the quality of short-term plans (Rodovalho and Cabral, 2014).

Data Collection

Data collection occurred in a large open-pit mine in Minas Gerais, Brazil. In this same mine also was the execution of a mining plan prepared according to the methodology described in the present study. The database meant to evaluate the behavior of the main variables, related to the mining process, includes the previous four months before the plan. Both parameters behavior analysis and equations generation were executed by using this database. The studied period corresponds to the dry season in the region. Rodovalho and Cabral (2014) studied the same season but considered one month to perform the multivariate analysis. In addition, the authors considered the previous three months to perform the parameters behavior analysis. The present study uses the same database to generate the equations through multivariate linear regression and perform the parameters behavior analysis. Furthermore, a comparative analysis will be performed to evaluate the impact of database size.

To collect data in real time was used a fleet management system. This system also organizes the reports and stratifies the database for each of the variables studied. Also it is possible to stratify the information in shifts, days, weeks and months. For the haulage fleet the following variables were evaluated: cycle time (CT), operational delay (OD), queuing time (QT), loading time (LT), maneuvering time (MT), payload (P), average haulage distance (AHD), operational moment (M) and the ratio of haulage distance when the truck is loaded and empty (RFD). Regarding the load equipment the following variables were evaluated: cycle time, operational delays, AHD, downtime (DT) and operational moment.

Variable Selection

Each of the cited variables in the previous section must be analyzed in a statistical tool that classifies as correlation with the response variable. In this study the response variable is the hourly productivity and statistical tool is represented by the stepwise forward and backward regression method. This method performs some rounds of correlation evaluating between each predictor variable and the response variable. In each round it is possible that variables are included or excluded. The selection ends when it identifies a group of predictor variables that hold the greatest correlation with the response variable. This study includes this type of analysis for eight shovel hydraulic excavators, four large mechanical loaders and three haulage fleet.

Table 1 shows the matrix correlation of excavator number one which has been applied the stepwise regression analysis. The report informs the group of variables that have more influence in hourly productivity of the excavator 1. The signs before the coefficients indicate direct or reverse proportionality in relation to the response variable.

Table 1 – Stepwise regression for excavator 1.

Response	Hourly productivity				
Step	1	2	3	4	5
Constant	1892	1704	1761	1837	1999
Variables	**Coefficients**				
AHD	-278	-604	-561	-549	-423.4
M		0.2274	0.2411	0.2574	0.2301
OD			-311	-309	-314.2
DT				-1732	-791
LT					-1981.7
R^2_{adj} (%)	11.12	84.51	90.49	91.57	92.5

Productivity Equations

After the variable selection step, for each transport fleet and loading equipment, will be possible to build equations using multiple linear regression. The hourly productivity equations are obtained by applying the stepwise regression method. Charnet (2008) states that the adjusted coefficient of determination (R^2_{adj}), measures the quality of the regression and the capability of the equation to explain a particular process. Table 2 shows each equation generated for loading operations with their respective adjusted coefficient of determination. Table 3 lists the equations for the haulage fleets. Blank fields in tables 2 and 3 indicate that the variable has been dropped or does not have significant influence in a given process. Analysis of tables two and tree show that the adjusted determination coefficients are satisfactory. This information indicates that the equations have high possibility to explain the studied process.

Table 2 – Loading Machines equations for hourly productivity estimations

Loading Machines	Coefficients of equations							
	Constant	AHD	M	DT	OD	CT	LT	R^2_{adj} (%)
Excavator 1	1999.2	-423.4	0.2301	-791	-314.2		-1981.7	92.5
Excavator 2	2399.8	-350.7	0.1799	-4592.6	-332.96		-5721	93.9
Excavator 3	1981.1	-406.4	0.2386	-2271.2	-362.3	54.2	-4502.5	90.2
Excavator 4	1739.5	-331.2	0.1885	-1656.4	-306.8	180	-1092.5	92.3
Excavator 5	1394	-399.5	0.2597	1896.3	-331.4	12.3	3260.2	96.8
Excavator 6	1041.9	-249.1	0.2457	144.2	-209.9		-93.4	92

Excavator 7	1255.9	-271	0.2267	260.2	-258.4		-276	90
Excavator 8	1497.5	-85.9		-8287.2		889		88.6
Loader 1	1553.9	-385.4	0.2305	-497.5	-297.2			91.9
Loader 2	2302.4	-88.2	0.02	-10411.1	-251.59	395.68	-13190.6	94.5
Loader 3	2074.8	24.6	-0.0238	-13437.5	-285.7	-55.6	-13436.4	92.7
Loader 4	2649.1	-23.6	-0.0083	-12746.9	-261.6		-18302.9	91

Table 3 – Haulage fleet equations for hourly productivity estimations

Coefficients	**Haulage fleet**		
	Fleet A	Fleet B	Fleet C
Constant	253.9	346.6	398.7
M	0.2938	0.2944	0.2654
OD	-4.47	-2.8	-4.89
CT	-189.2	74.36	18.95
QT	-388.3	-60.63	141.5
MT	790.8	216.2	11.2
LT	-161.3	212.2	
P	0.8027	-0.1875	-0.1523
AHD	-80.352	-103.8	-102.7
RFD	-9.7	21.384	5.22
R^2 adj (%)	96.1	95.2	93.4

CASE STUDY

The equations presented in the previous section were applied to the development of a mining monthly plan in a large mine. All stages of this study were conducted in an iron open-pit mine. The operations occur in shifts of six hours for 24 hours a day with no production stoppage during the year.

The ore production demands and waste removal are defined in the first stage of mining plan development. These demands include the ore feeding to the crusher, waste removal and other flows of budget. The performance indicators provided in the budget are used for setting the volume for each mining face. This information is used to draw the haulage profiles and calculate the AHD. Therefore, the volumes obtained for each mining face are preliminary.

In the studied mine there are twelve active loading points. Each mining face can be composed by ore and waste. The ore volume can supply the crusher or strategic stock piles. The waste is disposed in one of the actives waste piles. The mass calculation of each mining face uses the indicator OEE (Overall Equipment Effectiveness). This indicator is provided in annual budgets and is calculated by multiplying mechanical availability, operational usage and hourly productivity of the fleet. However, before the development of the mining plan these masses will be adjusted according to the results obtained in the simulation. The AHD is generated by the distance between each mining face to the active destinations. This distance is weighted by the mass of each flow. Table 4 presents all the parameters to replace in the equations described in Table 2. Table 5 shows all the parameters that should be replaced in the equations described in Table 3.

Table 4 – Operational parameters for loading equipments

Loading Machines	**Operational parameters**						
	AHD (km)	M (t. km/h)	DT (h)	OD (h)	CT (h)	LT (h)	HP (t/h)
Excavator 1	4.85	10010.2	0.0376	0.556		0.053	1934.8
Excavator 2	4.2	9944.2	0.0384	0.447		0.042	2150.9
Excavator 3	2.81	7068.7	0.0279	0.594	0.018	0.043	2053.5
Excavator 4	4.34	9842.1	0.0291	0.556	0.028	0.051	1886.2
Excavator 5	2.99	6388.4	0.0282	0.6405	0.029	0.053	1871.9
Excavator 6	3.92	5234.6	0.0475	0.5616		0.073	1231.7
Excavator 7	4.03	7581.8	0.0384	0.8056		0.062	1666.6
Excavator 8	4.09		0.0396		0.329		1110.2
Loader 1	2.79	5162.2	0.0392	0.5243			1493.2
Loader 2	2.15	2725.7	0.038	0.7854	0.1881	0.041	1093.8
Loader 3	1.98	2506.4	0.0285	0.7548	0.1683	0.042	887.3
Loader 4	1.98	2681.9	0.0381	0.5202		0.040	1190.7

Table 5 – Operational parameters for haulage fleet

Operational parameters	**Haulage fleet**		
	Fleet A	Fleet B	Fleet C
M (t.km/h)	828.86	1148.46	1401.35
OD (h)	9.48	8.2	10.29
CT (h)	0.181	0.237	0.311
QT (h)	0.0224	0.0249	0.0298
MT (h)	0.01995	0.01908	
LT (h)	0.03588	0.04004	0.041
P (t)	141.93	176.65	234.31
AHD (km)	2.21	2.85	3.43

RFD	0.837	0.892	0.891
HP (t/h)	350.9	380	346.6

The equation (1) is related to excavator 1. This equation represents the way that the values should be replaced. It must consider the results obtained in the rounds of multiple linear regression analysis. The result is provided in t/h and represents the hourly productivity planned for excavator 1 during the month. The same process should be repeated for the other equipments.

$$HP = 1999.2 - 423.4 \cdot AHD + 0.2301 \cdot M - 791 \cdot DT - 314.2 \cdot OD - 1981.7 \cdot LT \qquad (1)$$

The results of hourly productivity estimation for all equipments allow calculation of the mass for each mine face. This process ensures that the mass of each mine face is compatible with the actual capacity of the equipments during that month. Therefore, it is possible to design the mining monthly pushbacks compatible with the handling capacity of each mine.

DISCUSSION OF THE RESULTS

After implementing the plan is possible to evaluate the degree of deviation between the estimated hourly productivity and actual productivity achieved in the month. Furthermore, it is possible to evaluate the deviation for each load equipment or haulage fleet. Table 6 shows the estimated hourly productivity, the actual values and the deviation between both values. Analyzing this table, there is low variation for loading and haulage operations. However, some loading equipments have high variation. The excavators 5, 7 had many mechanical and operational failures. Each failure occurred between short periods of time and caused instability in the production process. Depending on these specific failures, there was high deviation for these machines. On the other hand, the loader 3 achieved productivity higher than estimated because this equipment supported the feeding of crusher. The result was assigned to operation in closer load points. Regarding other loading equipment and haulage fleets, the deviation was satisfactory. The analysis performed by Rodovalho and Cabral (2014) showed variations of up to 26%. In this study the maximum variation was less than 15%. This result is attributed to the use of larger databases. A larger volume of data makes the simulation more accurate and reliable.

Table 6 – Hourly productivity results and deviations related to the estimative

	Estimate (t/h)	Real (t/h)	Deviation (%)
Excavator 1	1934.8	1952.9	0.9%
Excavator 2	2150.9	2124.6	-1.2%
Excavator 3	2053.5	1996	-2.9%
Excavator 4	1886.2	1861.9	-3.8%

Excavator 5	1871.9	1686.7	-11%
Excavator 6	1231.6	1258.7	2.1%
Excavator 7	1666.6	1521.8	-9.5%
Excavator 8	1110.2	1119.6	0.8%
Loader 1	1493.2	1478.5	-1%
Loader 2	1093.8	1195.6	8.5%
Loader 3	887.3	1036.8	14.4%
Loader 4	1190.7	1182.1	-0.7%
Loading Machines (Total)	**1717.8**	**1768.3**	**2.9%**
Haulage fleet A	350.9	349.2	-1%
Haulage fleet B	380.1	366.1	-4%
Haulage fleet C	346.6	357.3	3%
Haulage fleet (Total)	**362.2**	**360**	**-1%**

CONCLUSION

This work established aim at generating productivity equations for loading operations and haulage with reduced variation. Methodology application for loading and haulage fleets achieved the objectives. Evidence of compliance is low deviation between estimated and actual. This result can be assigned to the database. The use of a larger database provides greater accuracy and reliability to the simulation results. However, the simulation model showed high deviations for excavator 5 and 7 for the loader 3. These negative deviations are justified by mechanical and operational failures unplanned. This type of event is considered an outlier to the simulation model and it shows the complexity of translating an industrial process in mathematical equations. However, the model can be considered as a tool to become a mining plan more realistic and suitable with production capacity of a mine. Management decisions can also influence the results. The allocation of the loader 3 in stocks near the crusher was not planned and justified the increase of productivity.

The techniques and statistical tools used are affordable. There is availability of various packages that are able to perform these analyzes or generate similar equations. Many packages can be used for free. Thus, future work can be developed by adapting the methodology of this study. Research can move forward from application to other mine settings or develop analysis covering quarterly, annual and multi-year horizons.

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SIMULATION OF STRUCTURAL DISCONTINUITIES IN MINING PHYSICAL MODELS FROM GEOLOGICAL ANALOGUE MODELS AND DOUBLE SCALING

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ABSTRACT

In this study we present a brief overview of the analogue modeling methodology carried out in geological and physical modeling work applied in mining. The most important aspects that support Scaling Theory are also presented, as well as the materials used in this type of models considering scale similarities. The most widely used materials are summarized and their limitations are discussed. Based on this theoretical framework and previous experiences in analogue modeling, we suggest that it is possible to construct a rock massif containing a structural arrangement generated in a cortical scale analogue model, thus generating analogue model of mining process considering natural preexisting structures. We refer to this process as "double scaling", since the second assembly modeling will vary the mechanical properties of the material, due to modification of the scale. Finally, this study summarizes the data acquisition systems used in published works, and highlight the importance of generating data acquisition systems, to measure the real-time surface and internal deformation of analogue models, allowing researchers to identify how the variables influence real geological and mining processes.

KEYWORDS

Analogue, physical, model, geology, mining, scale, materials, data acquisition.

INTRODUCTION

Analogue modelling is a powerful tool developed since the XIX century that helps the interpretation of geodynamic environments, through the simplification and research of isolated parameters of the studied process.

Since the “Scaling Theory” developed by M. King Hubbert, this methodology has been considered a quantitative tool than merely a descriptive one. Scaling Theory indicates that there must be a geometric and dynamic similarity with nature. The results of the modelling can be compared with natural examples when scaled experiments are carried out. Numerous advances have spurred a wide range of studies using this technique, employing diverse analogue materials for scale representation of lithospheric layers and various assemblies for the simulation of different deformational environments. The introduction of computer and numerical modeling techniques in the data analysis and processing, has allowed a significant advance in the acquisition of information, its interpretation, and comparison to natural prototypes.

The mining process models cannot be extrapolated directly to scale, therefore, the delivered results do not imply an exact behavior of reality but instead a good approximation, thus contributing to a better understanding of the phenomenon, as long as the limitations are considered, and the study variables are well recognized. This study suggests that there are materials with rheological and mechanical properties that allow scaled analogue modeling of geological and mining processes, respecting the geometric, kinematic and dynamic similarities. To this end, the mechanical characteristics of most of the materials used in analogue and physical modeling are presented, and a "double scaling" process is proposed to preserve structures (in situ fracturing) prior to the analogue model of rock massif.

Moreover, we propose, as a requirement, the generation of a data acquisition system, to measure the real-time surface and internal deformation of analogue models. Tomography, sound waves and several numerical algorithms have been developed to implement data acquisition systems for internal deformation measurement. To measure the surface deformation, some researchers have developed PIV (Particle Image Velocimeter) algorithms, which enable to recognize the instantaneous displacement of surface particles of geologic analogue models, whose software's will be tested for profile deformation.

A BRIEF STATE OF ART IN ANALOGUE MODELING

Analogue modelling of geological processes has been performed since the XIX century. The first experiments using this methodology were carried out by Hall in 1815, who through the construction of a modeling table, was able to support the hypothesis that natural folds in rock strata were the product of horizontal compressive stresses. During that century, much of the progress made by naturalists in analogue experimentation (Figure 1), highlighted the role of compressive stresses in the formation of mountain ranges (e.g. Cadell, 1889).

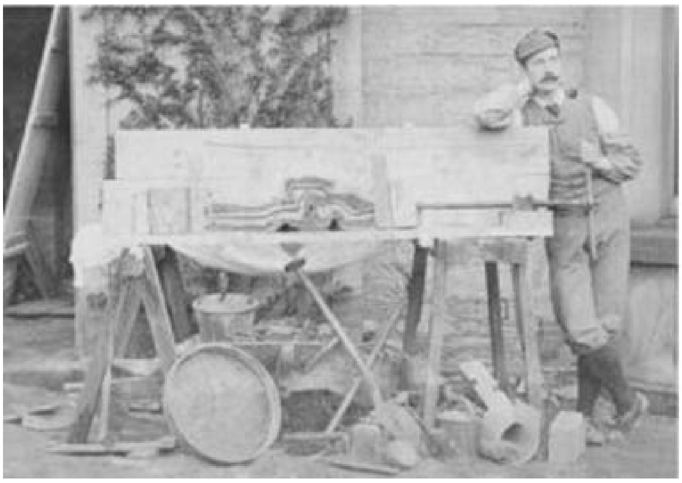

Figure 1 – Photograph of Cadell and his thrusting analogue experiments (source: Geological Survey of Great Britain's Memoir).

Several and diverse experimental setups were developed during the XX century in order to study various prototypes of natural processes. A fundamental advance in the art of modeling was the introduction of the Scaling Theory (Hubbert, 1937). Since then, more than several hundred studies have enabled a wide range of assemblies of different tectonic environments, particularly those related to deformation of the upper levels of the lithosphere in response to natural stresses. Examples include analyses of the influence of erosion and sedimentation, intrusion, reactivation of preexisting structures, folding, rifting and basin analysis, fold-and-thrusts belts and accretionary wedges, among others phenomena (Davis et al. 1983; Vendeville et al. 1987; Mulugeta, 1988; McClay, 1989; Cobbold et al., 1993; Bonini, 1998). The use of this technique has been complemented with technology that enables data acquisition and the use of several materials to simulate the different behaviors of lithospheric rocks. Currently, analogue modeling is further complemented by numerical modeling, softwares of data analysis, 3D digital reconstruction of the deformed volumes, and other techniques. The great volume of analogue model studies in geology, has enabled significant progress in the study of the employed materials (Rossi & Storti et al., 2003; Eisendstadt & Sims, 2005; Holland et al, 2006; Graveleau et al., 2011; Gomes, 2013) as well as assembly methods; case in point, is the proof that the physical handling technique of granular materials (eg. sand deposition technique in layers) can greatly affect their friction coefficient (Krantz, 1991; Lohrmann et al., 2003; Panien et al, 2006; Maillot, 2013).

Through the use of analogue modeling, it has been shown that the spacing and position of fractures plays an important role in their reactivation (Sassi et al., 1993), and that a non-coaxial angle between the compressional and extensional stresses is necessary to produce an inversion of high dip angle structures, with a near optimum angle of 15º (Brun and Nalpas 1996, Gartrell et al, 2005). It is important to mention that these models allow researchers to consider discontinuities and rock anisotropy as a variable during the deformative process.

In addition, there has been much research and other physical models applied to mining during the last century. Some studies developed in sublevel caving have focused on gravitational flow of the *in-situ* fragmented material (Janelid, 1972; Gustafsson, 1998; Power, 2004), while those related to block caving have opted for physical modeling to scale extraction processes, due to the high cost (in time and money) of direct trials (Janelid, 1972; Alvial, 1992; Gustafsson, 1998; Halim, 2004; Castro et al., 2007). In experiments with sandboxes, it has been observed that the gravitational flow of the fragmented material tends to form a very similar ellipsoid to that of the flow of granular material into bins and hoppers; as a results it has been homologated to the equations and variables governing these flows (Kvapil, 1965, 1992; Janelid & Kvapil, 1966). However, some authors have suggested and corrected the Kvapil’s ellipsoid theory, by considering new variables and their corresponding scaled materials and model assembly (McCormick, 1968; Marano, 1980; Heslop & Laubscher, 1981; Heslop, 1983; Laubscher, 1994, 2000; Gustafsson, 1998).

SCALING THEORY

If the mechanical properties of the rocks, and space-time dimensions of a geological phenomenon are known, it is possible to determine the experimental procedure of the analogue model that is been simulated (Hubbert, 1937, 1951; Mandel, 1963). In order to reliably reproduce the mining-geological phenomena; the geometric, dynamic, and kinematic variables must be scaled, establishing a proportionality constant between the model and the natural prototype.

The geometric similarity is related to the length (L), area (L^2), volume (L^3) and the curvature (L^{-1}) of the natural phenomenon studied. The ratio between the natural length and the model defines the length scale factor ($L'=L_m/L_n$), and may range from 10^{-6} to lithospheric scale, and 10^{-3} at the regional level. The kinematic similarity refers to the time (T) required to generate a change in the position or in the form of two geometrically similar phenomena. Experimentally, the scale factor of time is defined as $T'=T_m/T_n$, and this factor also scale the speed ($V'=L'/T'$) and acceleration ($A'=L'/T'^2$) of phenomena. Particularly, in geological processes, the time it takes to deform the Earth's crust is on the order of thousands to millions of years, while in analog modeling these years are reduced in a few hours or days.

The dynamic similarity requires geometric and kinematic similarity, and that the prototype and model show proportional forces acting on a unit mass (M) per volume (V). Based on mass and density ratios of the material model and natural rock ($M'=M_m/M_n$, $V'=V_m/V_n$ and $\rho'=\rho_m/\rho_n$), the scaling factor for the force ($F'=M'\cdot A'$) and stress ($\sigma'=M'\cdot A'\cdot L'^{-2}=A\cdot\rho'\cdot L'$) is experimentally defined. For simulations of Earth’s processes, if we assume a unit area, average density of 2800kg/m^3, cortical thickness of 30km and g=9.81m/s^2, then the vertical stress result is approximately 824 MPa at the base of the crust, meaning that laboratory scale models should reach 824Pa at a depth of 3cm (with $\rho' = 1$ and $L' = 10^6$) (Davy & Cobbold, 1991).

From the above equations, and assuming that gravity is identical between the model and the geological phenomenon ($A_m=A_n=G$), we obtain that $A'=1$ and $L'=T'^{-2}$. In addition, considering that the friction coefficient is dimensionless, it must be equal to the natural rock and the modeling material. Finally, assuming that the rocks follows the Mohr-Coulomb failure criterion, the cohesion of the model material must be scaled, and since it has the same units as stress, cohesion has the same scale factor ($C'= L'\cdot A'\cdot\rho'$).

Scaled analogue materials

To define the materials to employ in the modelling, an analysis should be performed by considering the mechanical characteristics required to represent the material of the natural phenomenon. To this end, in first instance it is necessary to define the material to be studied, creating a primary complexity on whether it is considered the intact rock, the rock massif or the blocks of rocks and the interaction among them.

Analogue models implicitly assumes that physically discontinuous material (e.g. sand) can model continuous media (e.g. granite). This assumption is one of the main criticisms on analogue modeling, because if the sand is scaled to represent modeled rocks, the sand grains have an extremely large size, as the space between them (Weijermars et al., 1993). Despite the differences in particle size, and the porosity between the two media, granular materials are chosen by the similarity of the mechanical properties, even if the deformed materials generate larger scaled shear zones than naturals (Mandl, 1988). Furthermore, it is possible to assume that the granular materials are continuous media if the scale of the phenomenon studied is much larger than grain size (Weijermars et al., 1993).

Analogue models that study the crust, usually assume a scaling factor of $L'=10^{-4}$-10^{-6} (i.e. 1cm equals 0,1-10km). Assuming cohesion of rocks of C=0-100MPa, and similar densities between the rock and the material (i.e., $\rho'\approx1$ and $A'=1$), these models would require cohesive materials with C=0-100Pa, similar to soils values (Beniawski, 1989). Since the friction coefficient is dimensionless, this parameter must be, if possible, similar to the studied phenomenon. The material that fulfill the mechanical requirements of above is the sand, since its deformation is not sensitive to the rate of deformation as clays (Eisenstadt & Sims, 2005), or has a high friction coefficient as gravels (Castro et al., 2007). Cohesion of sand is less than 500 Pa and its friction coefficient range from 0.4 to 1.1, although values are susceptible to the variation of roundness, sphericity, homogeneity and composition of the grains, among other factors (Mair et al. 2002, Anthony & Marone, 2005, Eisenstadt & Sims, 2005; Klinkmüeller et al, 2016).

Physical models applied to mining, unlike geological studies, are analyzed on a smaller scale ($L'=10^{-2}$-10^{-3}, i.e. 1cm equals 0.1-10m) and therefore in fragmented and non-cohesive media. The size of the fragmented rock is 1-10m, and multiplying by the scale factor, sand-gravel material for simulation is required at a size of 0.001-0.1m. The most commonly used material is sand (Kvapil, 1965; McCornick, 1968; Yengue, 1980, Marano, 1980; Susaeta, 2000); however, since sand does not have the geometry of the extracted blocks in mining operations nor a typical friction coefficient above 0.84, some authors propose that the gravel is the best material to simulate caving extraction processes, if a smaller scale than 1: 100 is used (Halim, 2004; Castro et al., 2006).

Scaling mechanical properties

Since the mechanical properties of the rocks studied in geological-mining models have a similar behavior on a large scale, most of the studies have used the same materials, independent of its mains objectives. Some of the materials commonly used as analogues to the rock in the models are: sand, clay, glass beads, gravel, silica powder, hemihydrate powder ($CaSO_4{\cdot}H_2O$), PVC powder, glass powder and fragments. The average values of the mechanical properties of these materials are shown in Table. X, and described below, in particular, sands and gravels.

Table 1 - Physical properties of different granular materials used in analogue modelling.

Material	µ	ϕ (°)	C (Pa)	References
Clay	0,5-7	25-35	0-100	Sims (1993), Eisenstadt and Sims (2005).
Sand	0,58-0,84	30-40	0-150	Krantz (1991), Schellart (2000), Lohrmann et al. (2003), Panien (2006), Gomes (2013), Klinkmüeller et al. (2016).
Castor sugar	1,14	48,8	247	Schellart (2000).
Gravel	0,8-11	41-48	0-50	Castro et al. (2007, 2014).
Al-Microballs	0,46	24,7	6	Rossi & Storti (2003).
Si-Microspheres	0,44	23,9	1,5	Rossi & Storti (2003).
Glass microspheres	0,4-0,7	22-33	0-50	Schellart (2000), Santimano et al. (2015), Klinkmüeller et al. (2016).
Tapioca pearls	0,74	36,5	39	Grotenhuis et al. (2002)
Hemihydrate powder	0,71	35,37	62	Holland et al. (2006).
PVC powder	1,35	53	<5	Graveleau et al. (2008, 2011).
Silica powder	0,4-1,6	40-58	0-5000	Galland et al. (2006), Graveleau et al. (2011).

Based on a bibliographic compilation, and the identification of different materials and their mechanical properties, used in published papers regard this methodology, we recognize that the most widely type of sand used in different geological models is the quartz sand (Ellis & McClay, 1988; Davy & Cobbold, 1991; Gutscher et al, 1998; Yamada et al, 2006, among others). The average size used was <500µm and the measured density of the sand depended on its composition and size, obtaining values between 1-2,8g/cm^3. The typical friction angle used by different authors, even those that do not refer to their measurement, was 30°, but the vast majority, even those that measured experimentally, was found in the range of 30°-40° (µ=0.58-0.84). Much of the articles considered that the sand had no cohesion (<5Pa), but however, the value measured by many authors was <150Pa and, in exceptional cases, values exceeded 1000Pa (Rossi & Storti, 2003, and the references therein).

Gravels were used primarily in works related to the physical modeling of gravitational flows and mining caving (Castro et al, 2007, 2014; Halim et al, 2008; Trueman et al., 2008; Fuenzalida, 2012; Pineda, 2012; Orellana, 2012; Irribarra, 2014). The average sizes used was between 6-8mm, with a friction coefficient of µ=0.8-1.1 (φ=41°-48°) and negligible cohesion (<5 Pa). Peters (1984) and Powers (2004) showed, using physical models, that the size of the gravel had a small influence on the geometry of the displacement area in simulations of caving extraction method, independent of the scale model.

DOUBLE SCALING PROPOSAL

From the above background, we highlight the fact that some mining physical models have assumed the mechanical properties (cohesion and angle of internal friction) and the geometric scaled dimensions of a rock massif, like a continuous medium simulated with sand (e.g. sandboxes experiments). In cases where the fractured rock was modeled, it has been scaled the blocks of rock following the geometric similarity (e.g. gravels representing blocks). When interpretations involve the behavior of the rock massif, the results of physical models should be treated with care, because there has not been scaled the effects of natural fractures, where orientations, spacing, frequency, roughness and friction of walls, among others, could significantly affect the model results.

This article proposes that the use of granular materials is a good tool for studying similar models, provided that its selection consider scaling system variables, therefore representing the rock massif characteristics (e.g. in situ fracturing) in case that they have an important role in the outcome of the natural prototype. To this end, we propose the use of tools and assemblies used in geology (scale models of earth's lithosphere), in order to generate "natural" pre-existing structures to study the mining process. These structures can be developed in continuous isotropic media (e.g. sand) and define sets of structures with predefined orientations, provided that the deformation process is scaled respect to its geological variables and the conditions of thickness and stresses that enable the development of them. Once the model that represents a certain thickness of crust contains these structures, the representation of the mining process should consider a smaller length scale; which will change the material properties (e.g. cohesion); for this stage it is proposed to modify this property "in situ", in order to increase the cohesion of the system without losing the "natural" structures previously developed. While the orientations of the structures can be preserved, spacing and frequency will be modified to the new scale, so that this process must be solved mathematically, prior to physical testing. Subsequently, it must increase the cohesion of the material by using gelatins (Payrola et al., 2012; Bureau et al., 2014), or other materials, that once dries, permit brittle behavior of the model.

In summary, a dry sand, with particle size X, cohesion Y, friction coefficient Z, will be representative of the fragile crust, only if the scale similarities have been respected. Using a scale of 1:1000 (or 1:500), we can consider the system as an isotropic and continuous medium, in order to generate structures that will give discontinuity for the second model, in which the material must increase its cohesion in Y´ at the new scale. Bureau et al. (2012) showed, by the study of sedimentary intrusions, that sand increases its cohesion and friction coefficient after adding gelatin to a concentration of 1g/L, up to 490Pa and 0.08 respectively. We propose that through a good preliminary analysis and control of the concentration of gelatin prior to analogue modeling at the second scale, it will allow the analogical modelling of rock massif behavior (considering intact rock and fractures) for mining process. Performing this double scaling process does not imply necessarily the use of the structures as a variable of the model; in fact, it is possible to test the model with continuous material in order to isolate the variable under study (e.g. width tunnel, size of extraction point, rock column volume, stress direction, etc.), testing the selected variable with a predetermined arrange of set structures, or even modifying the structural arrangement mentioned above.

DATA ADQUISITION

In most of the analogue models in the literature, data acquisition is performed superficially both in plan and profile; this can be done during the deformation process to generate a display of the evolution of the deformation. Surface deformation can be measured through the collection of data, photos and videos related to grid deformation (Figure 2).

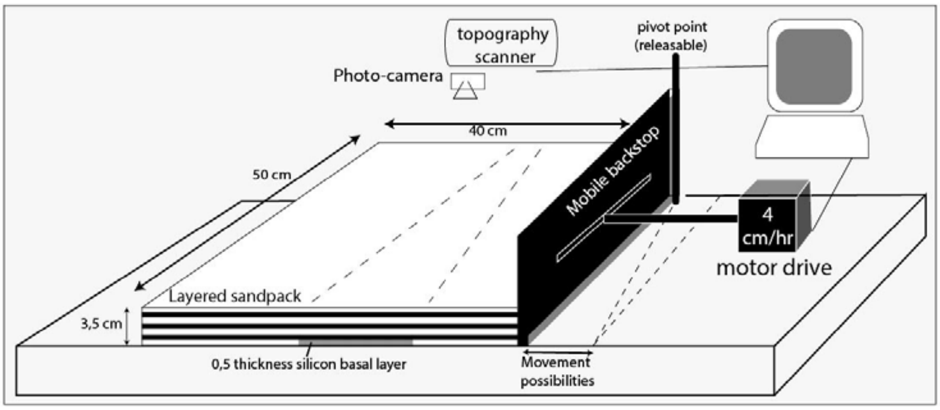

Figure 2 – Set-up example of a geological analogue model.

Technology has allowed for the recognition of areas with topographic changes (rise or subsidence) by surface scanner (Le Guerroué & Cobbold, 2006; Nilforoushan et al. 2008; Wu et al. 2009); there have also been algorithms developed to visualize the motions of points through time, so as to recognize the areas where the deformation is concentrated; these algorithms enable the generation of time-resolved particle image velocimetry (PIV) softwares that can be used to study multiple parameters of a flow pattern (Thielicke, and Stamhuis, 2014). Profile deformation of a geological analogue model has been made by direct visualization of the process when the assembly has a visible face (fixed wall parallel to the direction of applied stress); generally through photographic shoots allowing for reproduction at different speeds, since model processes can range from very slow to very fast. Profile deformation visualization is also enable by cross sections, once the model is finished, provided it does not alter the deformation generated during the modeling. This is a very important stage in the case of 3D modelization, as a series of transverse cuts can allow for a reconstruction of a virtual volume of deformation (Figure 3).

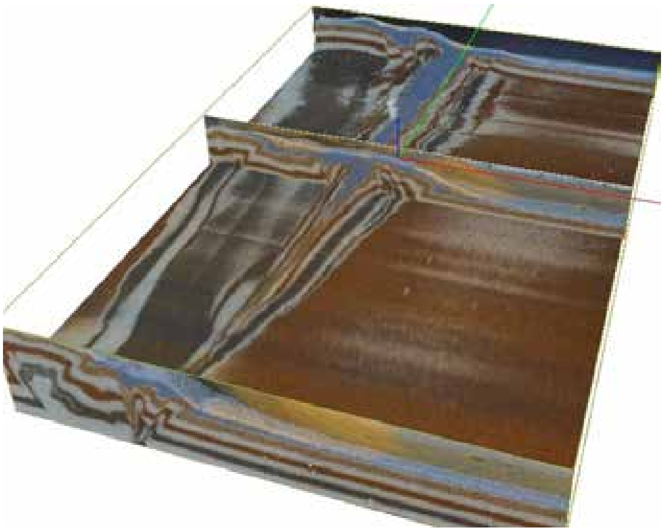

Figure 3 –Virtual Volume constructed by sliced final state of model (c. 5 mm), perpendicular to the principal structure's trend.

These reconstructions, as a virtual volume of deformation, provide a visualization of the model as a block diagram which can be analyzed; however, the latter will only show the result after the entire deformation process, unless the model has been stopped in some previous phase. The ability to record a three-dimensional reconstruction of the model, before being manually sectioned, could allow for a far more detailed study of 3-D structures. To this end, it has been used computerized x-ray tomography analysis (Coletta et al, 1991) and ultrasonic seismic recording of 3-D fault systems in sand analog geological models (Sherlock, 1999; Sherlock and Evans, 2001). Both techniques deliver a far more detailed interpretation of the structures, but they have not been widely developed due to their cost and other limitations.

DISCUSSION

Geological analogue modelling has been developed for almost two centuries and has allowed researchers to study countless variables that affect the results of the deformation of a large number of case studies, through generating varied assemblies representative of different tectonic environments. This wealth of production has generated, at the same time, efforts to properly acquire as much data as possible during the deformation process. However, much of the internal information is lost due to the high costs of implementing adequate data acquisition systems.

The superficial data of geological models is generally acquired using appropriate methods, and some authors have developed algorithms capable of filtering and recognize areas where deformation is accumulated by plan images. In mining, except in cases where there is interest to study the subsidence produced in a process, the data are acquired by studying layered profiles or by monitoring internal markers.

By performing a bibliographic review of physical models used in mining, we noticed that most of them correspond to specific case studies. Although these studies can be used to extrapolate generic cases of flows (e.g. gravitational flow), they are limited by the scale factors, the properties of the rocks present at the site, and in some cases by the geometry of the orebody. This indicates there must be special care taken when generating interpretations involving the behavior of rock masses, where the structural arrangement can play an important role in the process results.

While the most widely used material in geology is sand, since it allows a good representation of the behavior of the earth's crust, studies have been carried on with other materials, allowing for the modelling of different lithologies or rheological behaviors. The use of a discontinuous material (sandpack, for example) is justified by assuming that the material is representative of a continuum, as long as the scale of the phenomenon studied is much larger than grain size. Most of the mining physical models have also used sand, since they are especially focused on the study of gravitational flows. In some cases, fractured rock models have been simulated using scaled blocks sized (e.g. blocks of gravel to 1:30 and 1:100 scales); with the use of gravel of different characteristics, even gravels of the same exploited rock. In these cases, material flows can also be performed, but care must be taken to not make interpretations beyond the limitations of the model; keeping in mind that inhomogeneities, internal changes in the size, and shape of grain or blocks into the volume material can affect the results. In addition, the effect of natural fractures (joints or faults) cannot be considered in these models, even though the orientations, spacing, frequencies, friction walls, etc, could significantly affect the results, given that they have not been used as model variables.

We believe that it is important to take advantage of the virtues of both methodologies (geological analogue modeling and physical modeling of mining processes), so as to construct a more representative assembly of rock mass, in cases where the natural fracturing corresponds to a variable of the model. To this end, we propose a double scaling process, that first it will enable to produce a structural arrangement through an analogue crustal scale model, that later will be increase in cohesion to be analyzed in the new scale. Throughout the process we must have proper data acquisition systems, regarding the following: material choice, essays of the mechanical behavior and deformation during and after the end of modelization.

CONCLUSIONS

Through a literature review of works carried out by both analogue and physical modeling, we have a clear idea of the vast number of objectives (and consequently, of variables), set-ups, materials, and data acquisition systems used by researchers all over the world. In order to reliably reproduce the geological mining phenomena; geometric, dynamic, and kinematic variables must be scaled, to establish a proportionality constant between the model and the natural prototype.

The most commonly used material, in both physical an analogue models, is sand. Physical models applied to mining, unlike geological studies, are analyzed on a smaller scale ($L'=10^{-2}$-10^{-3}). However, since sand does not have the geometry of the extracted blocks in mining operations nor does it possess a typical friction coefficient, some authors propose that gravel is the best material to simulate caving extraction processes, only if a scale smaller than 1: 100 is used.

Analogue models implicitly assume that physically discontinuous material can model continuous media. Irregardless, the geological analogue models allow us to consider discontinuities and rock anisotropy as variables during the deformative process. But nevertheless, when interpretations involve the behavior of the rock mass, the results of the physical models should be treated with care, because the effects of natural fractures have not been scaled. We suggest that it is possible to construct a rock mass containing a structural arrangement generated in a cortical scale analogue model. To this end, we propose the use of tools and assemblies used in geology (scale models of earth's lithosphere), in order to generate "natural" pre-existing structures. The representation of the mining process should consider a smaller length scale; which will change the material properties (e.g. cohesion). For this stage it is proposed to modify this property "in situ", in order to increase the cohesion of the system without losing the "natural" structures previously developed. We refer to this process as "double scaling".

In most of the analogue models in the literature, data acquisition is performed superficially both in plan and profile, and can be done during the deformation process. Profile deformation can also be visualized by cross sections, once the model is finished. This is a very important stage in the case of 3D modelization because it allows for the reconstruction of a virtual volume of deformation. The ability to record internal information of the model before it is manually sectioned, could enable a far more detailed study of 3-D structures. Data acquisition systems have been implemented in order to achieve these ends, but they have not been widely developed due to their cost and other limitations.

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TECHNOLOGY ROADMAP ON TAILINGS MANAGEMENT AND NEW DISPOSAL

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TECHNOLOGY ROADMAP ON TAILINGS MANAGEMENT AND NEW DISPOSAL

ABSTRACT

Technology Roadmap (TRM) becomes essential to the planning of R & D of a company, since it is presented as a diagram to outline future goals and technological skills needed to achieve the "final stage". Moreover, it is a tool that integrates long and short term aspects of portfolio management activities to optimize the allocation of company resources. Due to the relevance for the future of mining, a TRM on tailings management and new disposal was developed in order to identify the need for technological development and the skills required to reduce or eliminate the need of dams in the medium / long term; identify new market niches and integrate R & D activities and operational improvement in the short view, medium and long term. The project involved experts from Vale, academia and industry to discuss and build a future vision. As a result, 26 possible niche markets and 35 technologies to be developed in the short, medium and long term were identified.

KEYWORDS

Technology Roadmap, tailings, dam, niches, technology, portfolio R&D, iron ore.

INTRODUCTION

For some time, Vale has been concerned with the management of iron ore tailings and its impact for the future of mining. Several projects had been developed to reduce the generation of tailings pulp, establishing new forms of its disposal and allocate new uses for tailings. Vale uses open innovation with several universities and research centers in Brazil and the world as well as has an R&D Portfolio focused on this strategic subject. Despite the significant number of projects, it was identified that there was a need to create a structured vision of the future on tailings management.

According to Robert Phaal (2004), "technology roadmapping is a flexible technique that is widely used within industry to support strategic and long-range planning. The approach provides a structured (and often graphical) means for exploring and communicating the relationships between evolving and developing markets, products and technologies over time. It is proposed that the roadmapping technique can help companies survive in turbulent environments by providing a focus for scanning the environment and a means of tracking the performance of individual, including potentially disruptive, technologies. Technology roadmaps are deceptively simple in terms of format, but their development poses significant challenges. In particular the scope is generally broad, covering a number of complex conceptual and human interactions." The Figure 01 shows an example of technology roadmap.

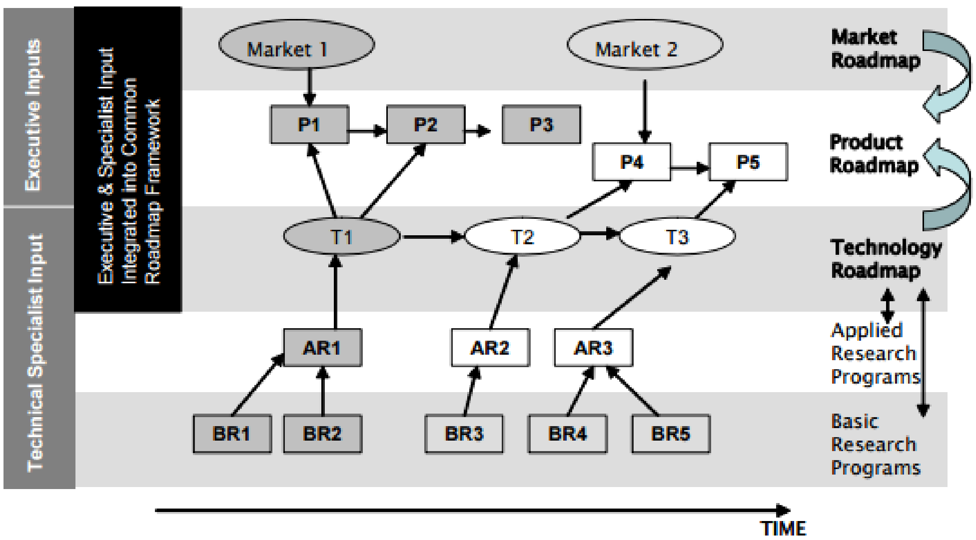

Figure 01: Roadmapping Captures Both Strategic and Tactical Aspects (Irene J. Petrick, 2008)

A Technology Roadmap has the following functions in a company:

- "Vision Document": a document to communicate the technology strategy in the organization and its implications for technological development in the medium and long term.
- "Strategic Framework": diagram to chart future technological goals and the skills necessary to achieve the "final status".
- "Tactical Plan": plan to promote specific interactions between existing products and help achieve short-term revenue targets.
- "Portfolio Priorization Tool": tool to integrate all aspects of long and short-term portfolio management activities to optimize the allocation of resources in the company.

These approaches enable the definition of technological routes to achieve the desired scenario and will enable the deployment of technological intelligence work in R & D vision. Strategic planning technology guided the choice of topic to be developed in the roadmap. According to the priorities for the coming years and the projected trends in future scenarios, it was decided that tailings management would be the technological focus of the work.

The tactical objectives of this work were:

- Identify need for technological development and the skills required to reduce or eliminate the need for dam in the medium / long term.
- Identify niche markets for tailings
- Integrate portfolio management of R & D and Operational Improvement in the short view, medium and long term.

METHODOLOGY

According to Irene J. Petrick (2008), "Roadmapping helps different individuals in the company discuss their view of the possibilities. Because it is an information organizing framework, a roadmap quickly helps these different individuals find linkages, common ground, and differences of opinion." Considering this important feature of the roadmap, Vale invited experts from different areas (environment, geotechnical engineering, process engineering, operation, other mineral commodities, mineral exploration, researchers) and experts from research centers, companies and universities to build the work. The goal was to build an environment of complementary skills and diverse experiences to promote constructive discussions under different points of view.

The roadmap consists of layers designed in previously defined time horizons. Trends and market drivers have been discussed and studied. The time horizon was divided into frame blocks: past / present (2016), short-term (2017-2018), medium term (2019-2023) and long term (2024-2030).

In this study, the following layers were defined in accordance with the business characteristics and objective expected for the work: drivers and trends, market niches, processes/technologies, R&D and Resources. The table 01 represents the roadmaps layers and time horizons.

Table 01: Roadmap layers and time horizons

Layers	2016	Short Term (2017 - 2018)	Medium Term (2019 - 2023)	Long Term (2014 – 2030)
Drivers and Trends				
Market Niches				
Processes / Technologies				
R&D and Resources				

The stages of the project were organized in three main phases: data preparation and review of public sector information, conducting technical workshops and compilation and dissemination of results.

Data Preparation and Review of Public Sector Information

A literature review on possible drivers that can influence the future of tailings management in mining was carried out. Legal and social aspects indicate that there will be a tendency to be greater restrictions dams than today. This is a factor that directly impacts the future of mineral production and continuity of operations, particularly in the Brazilian state of Minas Gerais.

Drivers and trends are mostly political, economic, legal and social aspects which may restrict the building of dams and force mining companies to dispose tailings in alternative ways to continue to maintain their current operating license. Some of the more important factors that were considered are listed below.

- Production of iron ore will double in volume by 2030 according to the projections of the leading companies. This represents a significant increase in the volume of tailings produced.
- Requirement to promote sustainable production in the mining sector
- Encouragement of investment in R&D for using mineral tailings in other industries.
- Payment for use of water resources and further water usage restrictions
- Participatory environmental management

Technical Workshops

A technical workshop for each roadmap layer was conducted: market niches, processes and technologies, R&D and Resources. Experts from different areas of Vale (environment, geotechnical engineering, process engineering, operation, other mineral commodities, mineral exploration, researchers), Companies, Universities and research centers were invited to discuss the matter. For each layer, were expected the followed objectives, as showed on the table 02.

Table 02 – Roadmap Layers and objectives

Layers	Objectives
Market Niches	Which industry can consume iron ore tailings in its processes?
Processes / Technologies	Which processes / technologies must be developed to achieve lower generation of waste, greater efficiency of the recovery process, with less need for water and alternatives for disposal of tailings?
R&D and Resources	What R & D projects should be developed, which technologies should be acquired, what skills should be developed to enable the routes defined in layers of niches and processes?

During the workshop, experts were encouraged to generate ideas freely and, at the end, questions were discussed and alignment was promoted.

Compilation and Dissemination of Results

All ideas were organized in blocks according to the time division and the defined layers. The table 03 resumes the final result of all workshops.

Table 03: Roadmap Results

Layers	2016	Short Term (2017 - 2018)	Medium Term (2019 - 2023)	Long Term (2014 – 2030)
Drivers and Trends	• Production of iron ore will double in volume by 2030 according to the projections of the leading companies. This represents a significant increase in the volume of tailings			

produced.

- Requirement to promote sustainable production in the mining sector
- Encouragement of investment in R&D for using mineral tailings in other industries.
- Payment for use of water resources and further water usage restrictions
- Participatory environmental management

Market Niches	2 niches	10 niches	7 niches	7 niches
Processes / Technologies	11 technologies	9 technologies	9 technologies	6 technologies
R&D and Resources	R&D Planning and Portfolio defined			

RESULTS

26 market niches (figure 02) and 35 technologies (figure 03) have been identified in the short, medium and long term. The identified market niches will be analyzed from technological, economic and social points of view for setting priorities for project development. Related technologies and processes will be evaluated to determine if they are already in use in the iron ore industry, the degree of novelty for the sector and which the maturity of each technology is. From this analysis, will be possible to prioritize technological routes to be developed and what resources are needed for this development.

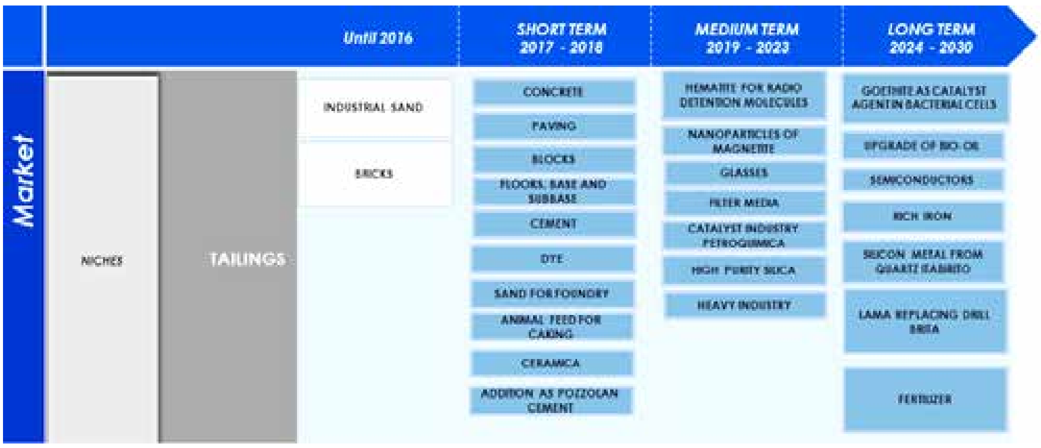

Figure 02: Market Niches

Each niche will be studied, considering the economic, technological and social aspects. After the data collection, correlation analysis will be conducted for prioritization of niches and preparing action plans for applying waste into new markets.

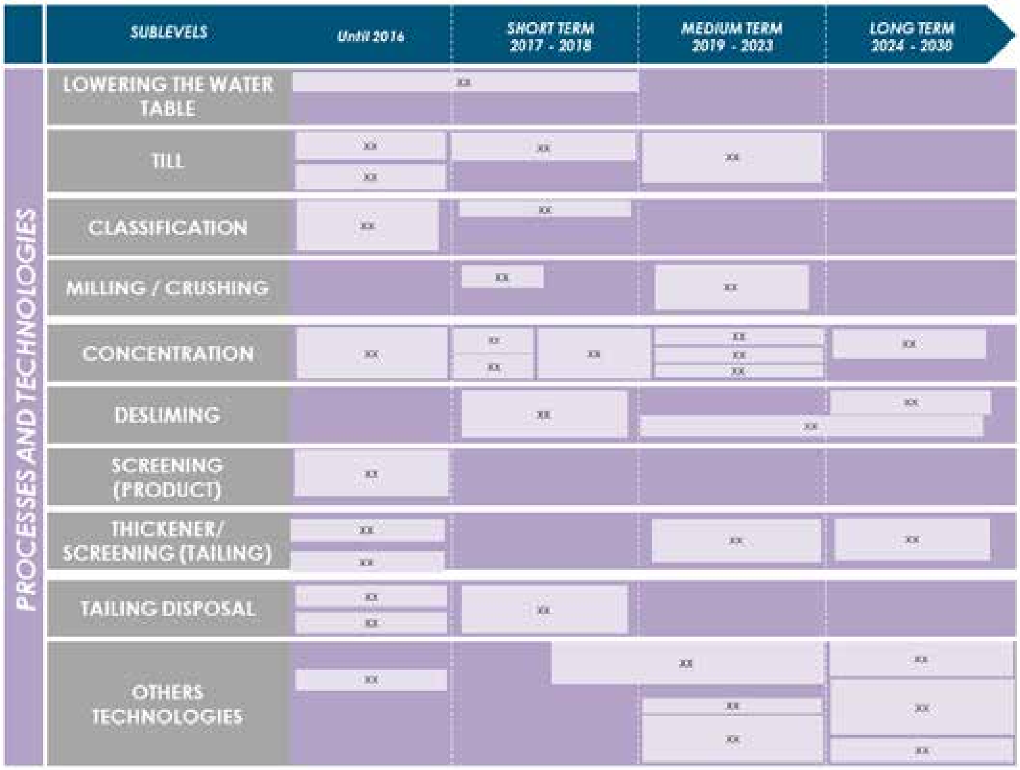

Figure 03: Technologies and Processes

While the technology layer and processes involve studies that are under confidentiality agreements do not allow the ideas generated to be published, it is worth emphasizing that analysis of technologies and processes to reduce the generation of tailings were also done to the blasting and mining phases. It is known that a properly performed blasting and selective mining process reduces the need for more efficient processing and tailings generation.

CONCLUSION

The study revealed that there is a range of opportunities to be developed and studied to enable the disposal of tailings for other purposes. In addition to new destinations for the tailings, there is also another range of opportunities for development of technologies in the process to reduce the generation of tailings. Acting on both fronts, generating less waste and new allocations, it is believed that there will be a significant reduction in the current dependence on dams.

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THE CYCLONE OF THE FUTURE

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THE CYCLONE OF THE FUTURE

ABSTRACT

The presence of unwanted coarse material in flotation feed streams negatively impacts both recovery and throughput of concentrator plants. The detection of this coarse material is a long- standing problem in mineral processing that has been poorly addressed by existing instrumentation systems. This problem has been overcome by the development of two new systems that detect in real-time the presence of coarse material in the overflow stream of individual hydrocyclones. They both use sensors mounted to the overflow pipe of the hydrocyclone and provide a robust and maintenance- free system measurement that enables corrective actions through operator intervention or automatic control strategies. One system is non-invasive and detects very coarse material 6mm or greater in size. The other system uses a wetted sensor and detects coarse material down to a lower size limit of approximately 100 µm. Both systems have been commercially deployed. The systems will be described in detail. Validation data and typical plant data will be shown.

KEY WORDS

Hydrocyclone, particle size, throughput, recovery, real-time, individual hydrocyclone, maintenance-free.

INTRODUCTION

In mineral beneficiation involving comminution and subsequent flotation, mineral recovery is strongly dependent on the proper particle size being delivered to flotation. The optimum particle size must be neither too fine nor too coarse. This paper presents a solution to the challenge of maintaining the optimal particle size in flotation feed. In general, reducing the amount of coarse material can significantly improve recovery and throughput of a plant. Due to the process and equipment designs, this coarse material challenge typically comes in two forms. The first form involves the unwanted delivery of very coarse particles, several millimeters or larger in size, to the flotation circuit. This is often caused by specific events, such as broken trommel screens on mill discharges, various hydrocyclone classifier malfunctions, or excessively high hydrocyclone feed density. The second form of coarse material challenge involves the unwanted delivery of coarse particles that are only slightly above the target size for the flotation feed, which would typically be in the 100 um to 200 um range. This is usually caused by poor control of the grinding process or deficiencies in hydrocyclone classification. The first challenge of very coarse material exists to varying degrees in many plants, while the second problem is a generic problem that exists in most plants.

Two related solutions based on novel instrumentation technologies are presented that have been developed to address these long-standing challenges. They involve robust sensors that are mounted on the overflow pipes of individual hydrocyclones to detect the presence of coarse material in real-time. These systems provide real-time overflow product size information that enables immediate corrective action by operators or various control room strategies.

Both solutions have been commercially deployed and will be described in detail, including the specific problem each addresses, system designs, installation and maintenance, validation data, and real plant data examples. The convention used in this paper for particle size is as follows: “pebbles” are particles 6mm – 12mm and larger in diameter; “sand” is particles 100um – 200um and larger in diameter.

OVERSIZE MONITORING SYSTEM (OSM)

Problem Statement – OSM

Many operators place a premium emphasis on asset management, which becomes necessary when increasing operating efficiency. However, the need to extract maximum value from the grind plant means that maintenance intervals will be stretched as long as possible. Additionally, maximizing the return on equipment investment requires operating the plant at the highest throughput possible without compromising safety, excessive equipment wear or reduced recovery. This presents a wide variety of operating conditions as one asset approaches the need for maintenance while others are new or recently refurbished. Hydrocyclones are one such asset that will report extremely coarse material, such as pebbles, to the overflow when not operating as designed. Pebbles reporting to the overflow are usually passed directly to the flotation system. Pebbles in the flotation feed reduce the economic performance of the concentrator by reducing valuable mineral recovery, reducing volumetric efficiency in the flotation cell, and in some cases by blocking the flow path in the flotation cells leading to partial or complete plant shutdown. It has also been noted that pebbles will damage equipment downstream of the grind circuit or cause blockages in pipelines and thickeners.

Detection of pebbles in the consolidated overflow from a hydrocyclone battery via acoustic sensors or traditional particle size monitors suffers from reduced sensitivity, slow update rates, and an inability to discern which hydrocyclone is passing the pebbles. Determining the exact source of the oversize material can be complicated and time consuming for a busy operations crew and usually has to be manually performed. While troubleshooting, the oversize material continues to report to the flotation circuit, resulting in considerable disruption to the flotation circuit until the offending hydrocyclone is taken offline.

With support from Rio Tinto, CiDRA has developed and commercially deployed a new technology for monitoring individual hydrocyclone overflow lines for pebbles and the associated increase in P80. This technology detects pebbles (6-12mm range and larger) passing through the hydrocyclone overflow. By monitoring the overflow as opposed to the underflow, these pebbles are detected irrespective of the cause, whether it is due to a plugged apex, a roping condition, certain operating conditions, damage to the hydrocyclone, or wear in the hydrocyclone. This technology enables the operators to greatly reduce the length of time in which pebbles pass through the overflow (pebble events). It allows the plant personnel to identify hydrocyclone damage or excessive wear and to ascertain whether or not a grind circuit is the cause of the pebble event.

System Design – OSM

One of CiDRA's core competencies is the measurement of acoustic information through the wall of a pipe (Gysling, Loose & van der Spek, 2005 [2], and O'Keefe, Maron & Gajardo, 2007 [3]). CiDRA has used its expertise to develop the CYCLONEtrac OSM system. It has been observed that pebbles have a significant probability of striking the inside of a hydrocyclone overflow pipe as they pass through such a pipe. Even in the presence of rubber liners, sufficient mechanical energy in the form of acoustic waves is transferred from the striking particle through the liner and into the wall of the pipe. The OSM system uses CiDRA's proprietary distributed acoustic sensor mounted on the outside of the pipe, which is tightly mechanically coupled to the pipe without couplant. The particle-induced acoustic wave travels through the pipe wall and is converted by CiDRA's proprietary sensor to an electrical signal. A unique algorithm extracts this very low level signal from the background noise to identify a pebble event. The system reports the rate of these pebble events and sounds an alarm when rate thresholds are exceeded. By using a distributed acoustic sensor as opposed to a point sensor, the probability of detecting a pebble passing through and striking the inside of the overflow pipe is greatly increased.

Installation and Maintenance – OSM

The installation process consists of 1) cleaning and possibly sanding of the pipe to remove material buildup, paint splatters, and other high points; 2) wrapping the sensor band around the pipe and cinching it tight using its captive screws; 3) covering the sensor band with a water tight cover and attaching the sensor band cable to the preamplifier built into the sensor cover; 4) mounting transmitters and power/communication junction boxes; 5) connecting power/communication lines between the sensor heads and the transmitters and between the transmitters and the power/communication junction boxes; 6) supplying power to the junction boxes; and 7) connecting an Ethernet line between the junction boxes and the CYCLONEtrac computer, which would typically be placed in the control room. Pictures of the installed system are given in Figure 1. The installation process is non-intrusive and allows for the hydrocyclone battery to remain operational during installation.

Figure 1 - Installed CYCLONEtrac OSM System at Kennecott with transmitter and junction box (upper left picture), sensor head (upper right picture), and a completely instrumented hydrocyclone battery (bottom picture)

Due to the non-intrusive nature of the instrument, no regularly scheduled maintenance is required nor are there any inherent wear mechanisms present in the instrument. The sensor head is reusable and may be reinstalled when an overflow is replaced. No calibration or zeroing is required when reinstalled.

Data Validation – OSM

The CYCLONEtrac OSM system has been commercially deployed at Rio Tinto Kennecott's concentrator in Utah, USA since 2010 (Cirulis & Russell, 2011[1]). The system has been fully validated and adopted by the plant. Three years of plant operation were used to validate the performance over both normal and abnormal operating conditions, including many pebble and sand events. During this time trends for individual hydrocyclones and the consolidated trend of each battery of hydrocyclones were used to adjust thresholds and alarms, thus balancing sensitivity against false alarms. Proper setting of thresholds has allowed the operators to effectively control the separation process via manual intervention. While the system displays the trend of various sizes of coarse material, the minimum material size for repeatable detection of individual particles is approximately 6 to 12 mm (Fig. 2).

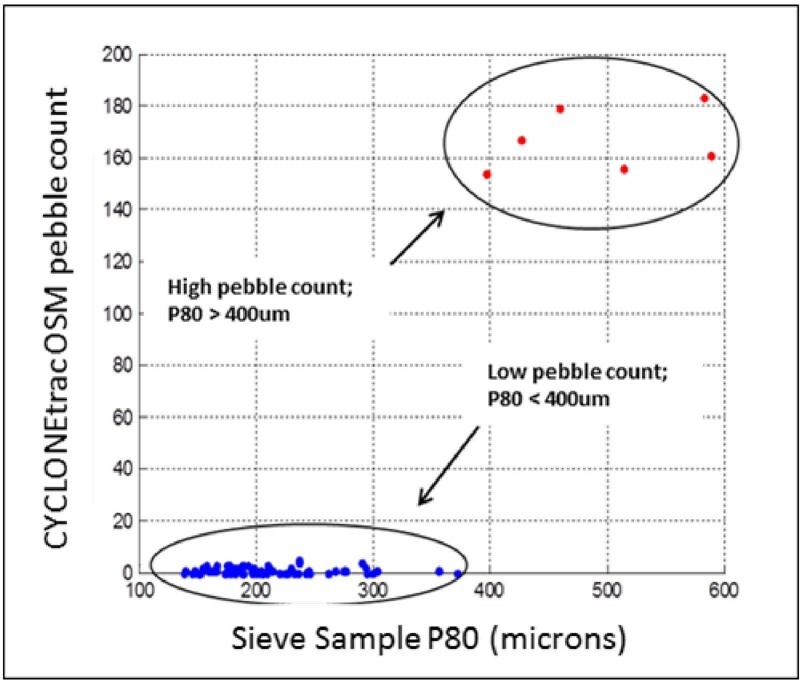

Figure 2 - During field validation, 78 sieve samples were analyzed and compared to OSM pebble counts. The shift in the P80 due to the increased mass flow of extremely coarse particles corresponds to pebble events. In this figure the red circles are periods in which pebble events were occurring, and the blue circles are periods in which the system indicated no pebble events were occurring.

Control Scheme – OSM

With the CYCLONEtrac OSM system now fully deployed and integrated into concentrator operations, operators receive actionable information. This information provides the operator with the ability to determine if there is an issue with the performance of an entire battery or with an individual hydrocyclone. An HMI (human-machine interface) in the control room displays trends on the aggregated performance of each hydrocyclone battery.

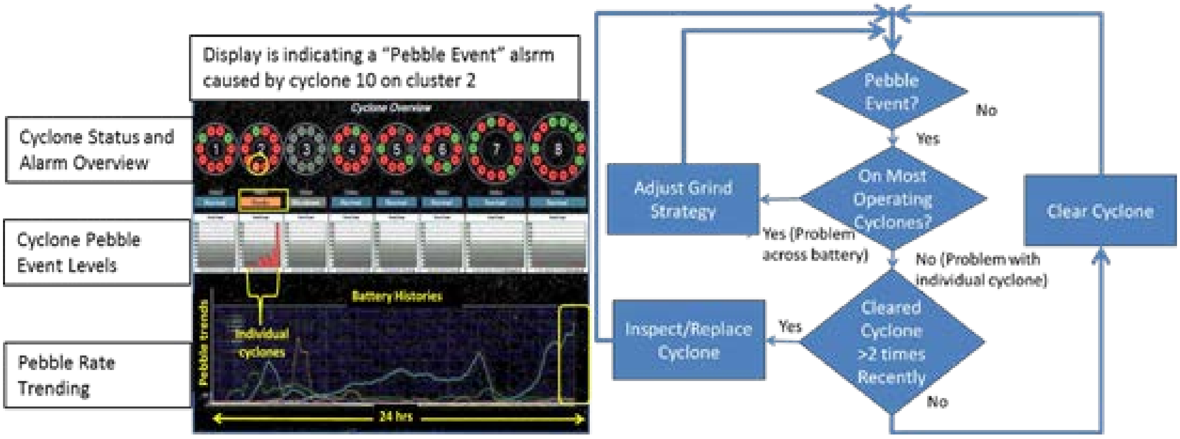

Figure 3 - Left, display is indicating a "Pebble Event." Right, sample flow sheet.

The operator discriminates between the two types of events by consulting the event level trends for the individual hydrocyclones in that battery. If a single hydrocyclone is indicated as the cause (Fig. 4), then the offending hydrocyclone may be isolated, and an alternate is placed in operation. In other cases, multiple hydrocyclones are responsible for the rapidly increasing large particle event trend of a hydrocyclone battery. In this case the operator can adjust operating parameters (e.g. hydrocyclone feed density) to return the system to steady state (Fig. 5).

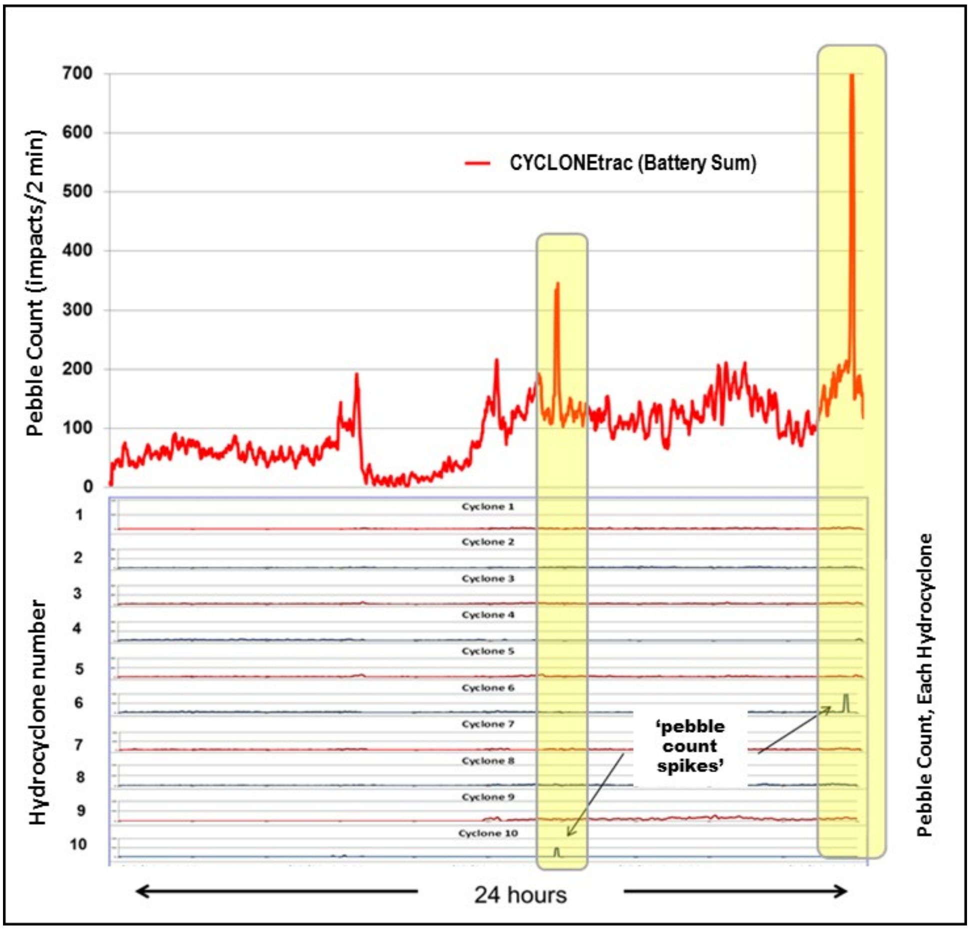

Figure 4 - Single hydrocyclone causes combined overflow of the battery to display a spike in the coarse material trend.

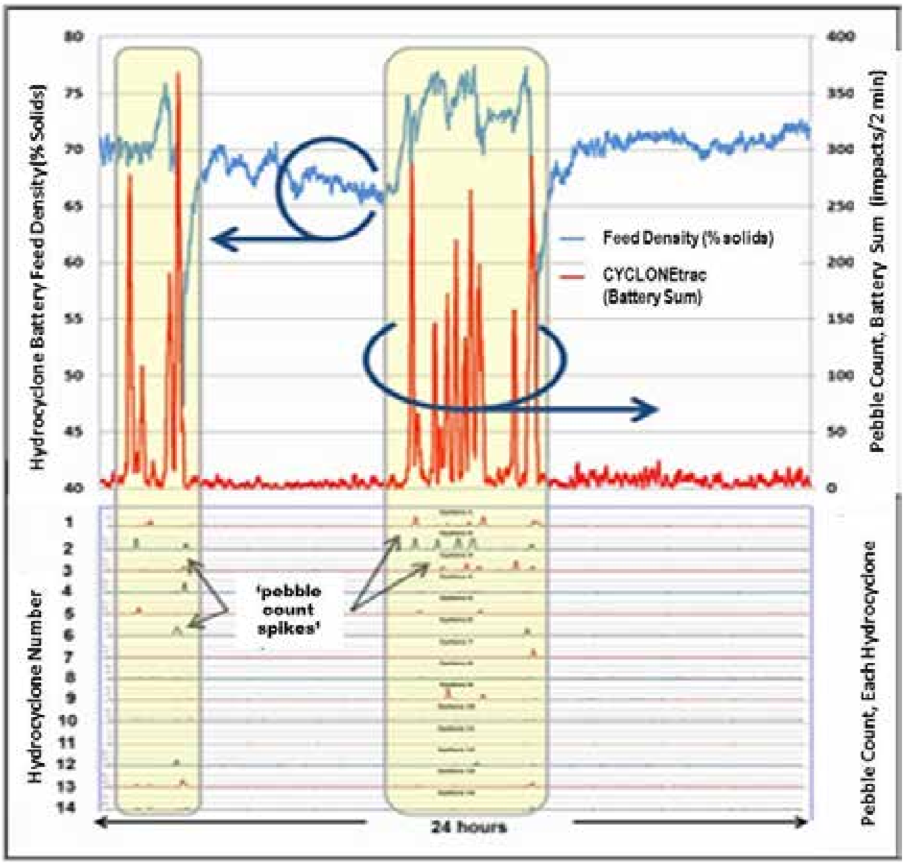

Figure 5 - Multiple hydrocyclones cause the combined overflow of the battery to display spikes in the coarse material trend, which is coincident with increases in feed density to the battery.

Over several years of commercial operation at Kennecott the OSM system has proven to be valuable in detecting excessive amounts of pebbles in the hydrocyclone overflow stream, enabling immediate corrective action, thereby helping to prevent any serious blockages in the rougher flotation or tailings circuits.

PARTICLE SIZE TRACKING SYSTEM (PST)

Problem Statement – PST

Valuable mineral recovery is strongly linked with the particle size distribution of the material delivered to the flotation circuit. Recovery of liberated and middling +150um material is significantly lower than that of -150um material. This is in part due to decreased mineral liberation and limitations in the ability to recover coarse particles by flotation.

The ability to make value based decisions around throughput and recovery relies on the ability to measure grind size. In order to achieve optimal throughput and recovery, the flotation feed grind size must be controlled and stabilized in real-time. Currently, there are three methods for determining grind size at Kennecott with varying levels of accuracy and frequency. These are: lab sieve analysis on rougher head samples, on-line sampling of the hydrocyclone battery consolidated overflow, and Marcy© Scale procedure.
The sieve screening on samples of the rougher head feed is considered the most reliable measurement of the

particle size distribution that is being presented to the flotation rougher cells. At Kennecott the rougher head feed is a combination of multiple ball mill hydrocyclone overflows and, therefore, does not represent the performance of any individual ball mill circuit. Thus, the rougher head stream samples cannot be used in a ball mill control strategy for real-time particle size control. Additionally, the sampling and processing time results in a 24-hour delay of results. This delay makes it difficult to use the grind size information for real-time process control and decision making.

The on-line sampling systems and ultrasonic particle size monitors were installed at Kennecott on each hydrocyclone battery in 2004. These systems periodically draw a sample from the consolidated overflow of the hydrocyclone overflow. The sample is then conditioned, and particle size is measured using an ultrasonic particle size monitor. Since installation, the instruments have proven to be maintenance intensive and, as a result, utilization has dropped significantly.

The Marcy Scale procedure, based on a procedure outlined by Wills (1988) [4], is used by the Kennecott operating crews to get an indication of the grind size at a moment in time. The procedure is relatively quick to perform; however, it is subject to sampling and procedural errors, resulting in inaccurate particle size measurement. Further, the manual nature of the procedure prevents it from being used for automatic process control.

In order to achieve optimal grind process control, a real-time accurate measurement of particle size is needed. At Kennecott a CIDRA CYCLONEtrac Particle Size Tracking (PST) system has been installed to serve this purpose. The CYCLONEtrac PST system offers the advantage of real-time particle size tracking on individual hydrocyclone overflows. Multiple particle size measurements are statistically processed to form a robust indication of the particle size generated by the ball mill circuit. The system does not require a stream sampling system and is low maintenance.

System Design – PST

The CYCLONEtrac PST system consists of sensor assemblies, junction box, and a control room computer. The sensor assembly is made up of a ruggedized probe that is in contact with the overflow stream and an integrated electronics package that is protected by a sealed metal enclosure. The probe itself is coated with an extremely hard layer for wear resistance. As the slurry stream hits the probe, it effectively “listens” to the impacts of individual particles. The impact response is processed by the on-board electronics in order to derive the particle size distribution in the slurry stream. The sensor assembly is powered by 24V and communicates to a junction box using Modbus protocol.

Figure 6 - Left, CYCLONEtrac PST sensor. Right, CYCLONEtrac PST sensor installed on pipe

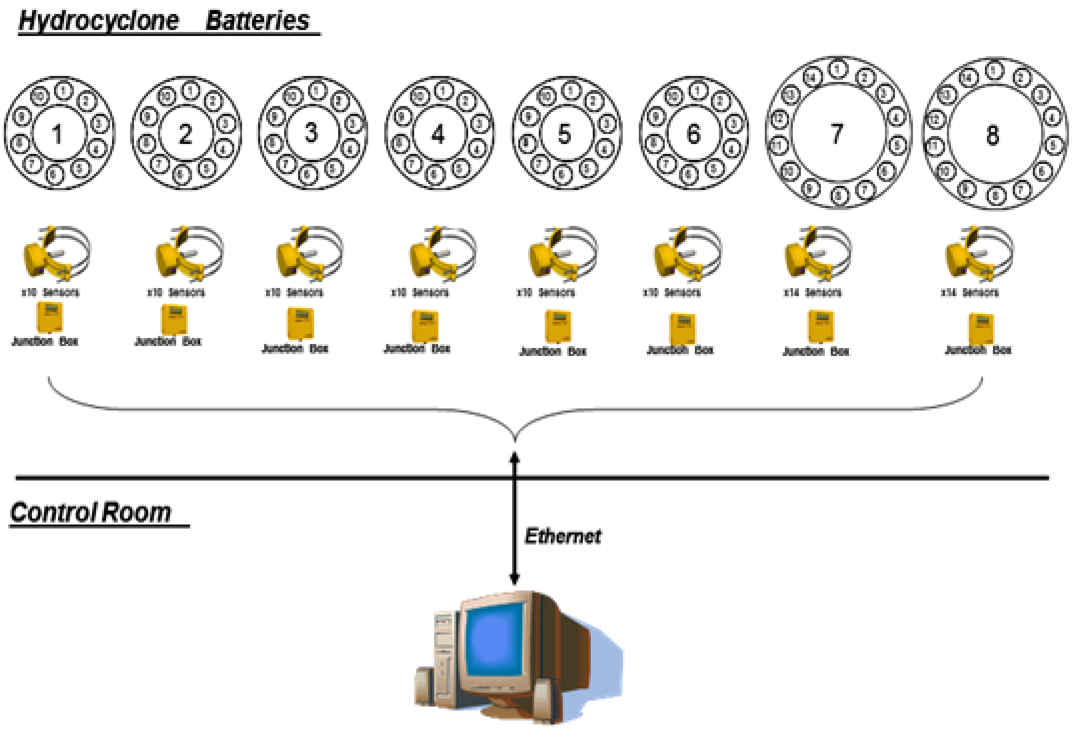

Figure 7 - Schematic of CYCLONEtrac PST system installed at Kennecott

Each junction box can interface with up to 16 sensor assemblies, providing both DC power and communications. The junction box takes Modbus communications from each sensor assembly and translates that into information over an industrial Ethernet network to a computer in the control room. The control room computer collects the measurements from each device and then passes the measurement to the Kennecott Distributed Control System (DCS) via an OPC tunnel.

Existing overflow pipes are modified using a portable magnetic drill. The PST probe is then inserted through the hole and secured in place with a simple pipe tap style saddle. If a PST unit is not available a plug saddle unit is available so the overflow pipe can be returned to service with limited impact to hydrocyclone availability. Typically, a hydrocyclone battery has a number of available hydrocyclones that are not in use. This allows installation of the PST devices without grind circuit downtime.

Installation of a PST unit on an already drilled overflow pipe can take as little as 10 minutes. Once a PST is installed, there is virtually no impact on overflow operation. There are two reasons for this. First, the probe occupies a very small surface area compared to the cross sectional area of the pipe. Second, the cylindrical shape of the probe creates minimal flow disturbance.

System Validation – PST

After the PST system was installed at Kennecott, a sampling campaign was undertaken to validate the performance of the system. CiDRA and Kennecott personnel worked closely to bump the hydrocyclone and grind circuit operating conditions over a range of grind sizes. During the validation campaign more than 130 samples were collected from individual hydrocyclone overflow streams. Sieve analysis was performed on the samples and the results compared to the output of the PST system. The particular particle size distribution feature of interest at Kennecott is the percent of material over 150um (100 mesh.) As stated earlier, valuable mineral recovery drops significantly for grind size that is greater than 150um. As such, the PST system was tailored to provide a direct real-time indication of the percent by weight of the stream that is +150um.
Figure 8 shows the real-time signal from the PST system with the validation sieve samples overlaid. During

the validation campaign the sampling variability was determined to be ±3.1% absolute. This variability is indicated on Figure 8 by the error bars. Figure 9 shows all 130 validation samples comparing the sieve analysis percent +150um to the PST readings. The validation campaign has demonstrated that the PST system is capable of predicting the percent +150um with ±6.3% absolute uncertainty. With consideration for sampling variability and sieve analysis precision, the results of the validation campaign give CiDRA and Kennecott confidence that the PST system will provide a real-time grind size measurement that can be used for value-based decision making and process control.

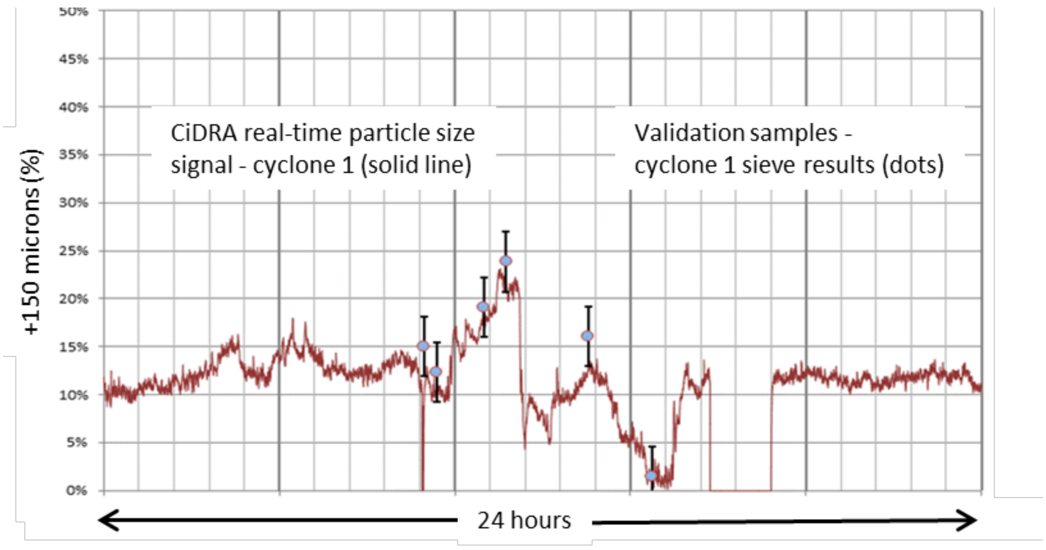

Figure 8 - Hydrocyclone 1 CiDRA signal vs. validation sieve samples

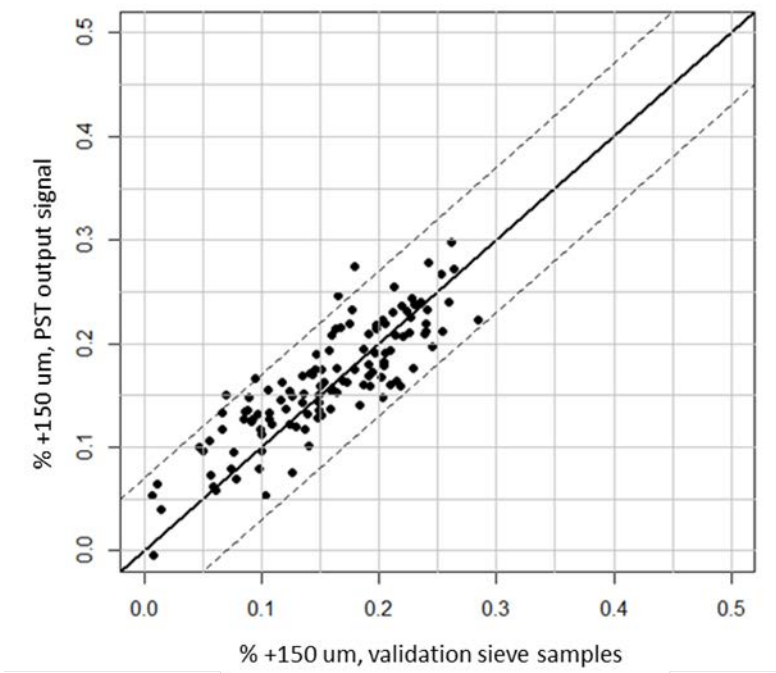

Figure 9 - Percent Mass Fraction +150um, PST output vs. validation sieve samples

Control Scheme – PST

Real-time grind size optimization requires three critical components. First, key drivers in the process must be measurable in real-time. This criterion is now fully met with the CYCLONEtrac PST system. Second, the process must be stabilized using closed loop control strategies. Third, the process must be driven to an optimal setpoint.

At Kennecott the ball mill and hydrocyclones are in a closed loop circuit. The primary drivers of hydrocyclone efficiency are feed density and operating pressure. However, throughput, ore hardness, and recirculating load are key variables for grinding efficiency. To address the second critical component of grind size optimization, a control scheme has been developed that uses the real-time PST measurement to stabilize hydrocyclone overflow +150um (100 mesh). The basis of control relies on manipulating hydrocyclone feed density within other circuit constraints. Figure 10 shows grind size stability under automatic control. The natural variability of grind size is shown by the PST signal on the left of Figure 10. The grind size is driven to a setpoint by observing the PST signal and manually adjusting the feed density. Without automatic control the grind size fluctuates while the density remains constant. Finally, the right portion of the graph shows the grind size stability under automatic control. The control system automatically adjusts the density setpoint to maintain grind size at setpoint.

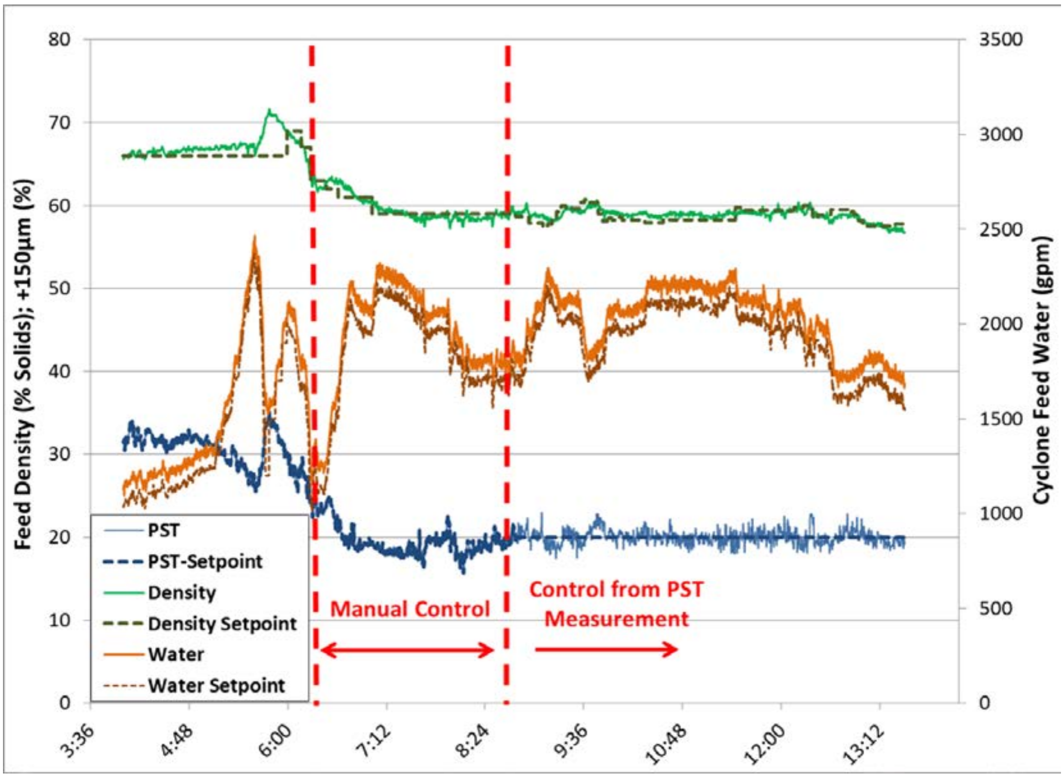

Figure 10 - Manipulating density for PST control

The closed loop grind size control scheme can now be used to maintain a target ball mill power draw and recirculating load to achieve optimum grinding efficiency, thus preventing ball mill overload and hydrocyclone roping conditions. Testing has proven that it is possible to maintain throughput and ball mill target power draw while reducing grind size. Figure 11 shows how the SAG mill tonnage remains constant while the hydrocyclone battery overflow stream percentage +150um drops significantly at the same time maintaining the target power draw.

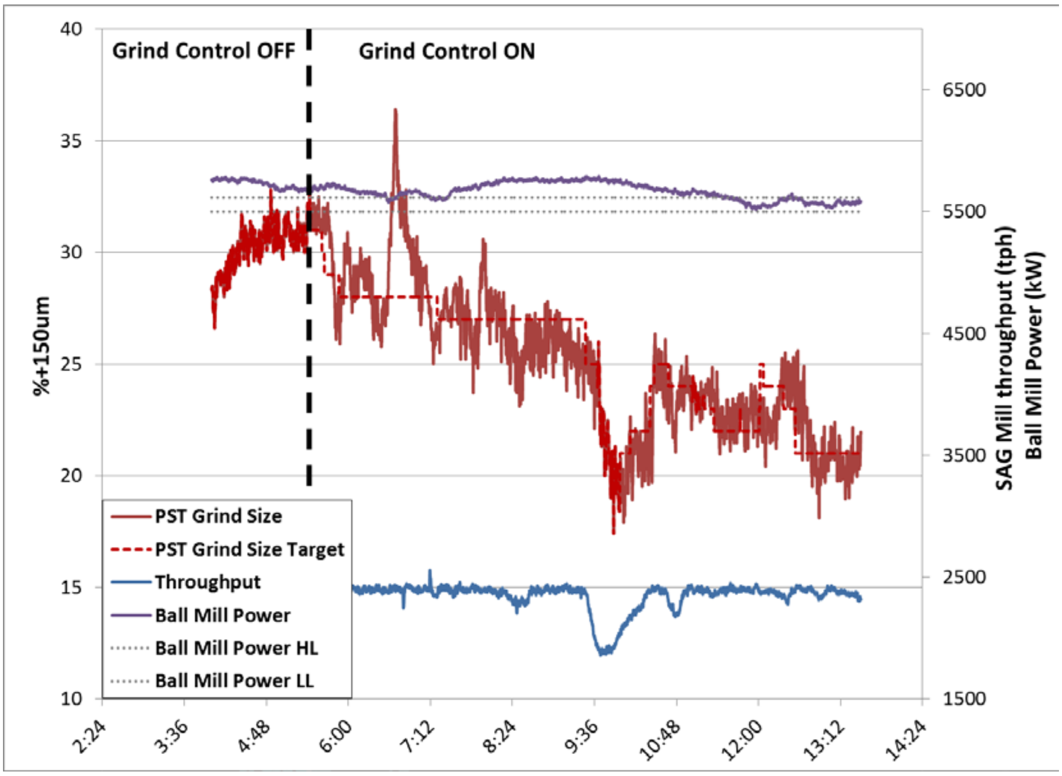

Figure 11 - Reduction of grind size with constant TPH and target ball mill power

While this first generation control scheme is relatively simple, the PST system is proving to be very effective in enabling grind circuit control at Kennecott and will ultimately assist value based decision making with respect to throughput, particle size, and recovery trade-offs.

CONCLUSIONS

Two novel systems have been presented that provide solutions to the long-standing challenge of coarse material in hydrocyclone overflow streams. Both solutions, OSM and PST, are complete, turn-key system based on novel instrumentation technologies that have been developed and validated in partnership with Kennecott at the Copperton concentrator over several years. The real-time information provided by the OSM system has been used to develop operating practices that have eliminated flotation circuit shutdowns due to flotation cell blockage. Likewise, the real-time information provided by the PST system has been used to develop first generation closed loop control logic for controlling % +150um flotation feed to a finer grind size while maintaining target grind line throughput. Further control system development is underway to optimize ball mill grind efficiently, balance plant-wide milling, and to make value based decisions between throughput and mineral recovery. Plant data have been presented showing operational examples that demonstrate how the systems are used to deliver value.

ACKNOWLEDGEMENTS

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THE FRAMEWORK FOR INNOVATION AS A DRIVER FOR THE EU-MINING AND RAW MATERIALS INDUSTRIES

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ABSTRACT

Innovation and thus R&D are key for the future industrial development of the European Union. Besides others the aim of the EU is a re-industrialization – the goal is that industry's share of GDP should be 20% by 2020. Nevertheless there have to be the appropriate framework conditions to support R&D and thus to foster innovation, not only, but especially within the EU mining and raw materials industries. The paper will show that neither an appropriate raw materials policy, nor supporting research and innovation programs will be sufficient to reach innovation if there is not as well an appropriate legal framework ensuring a level playing field compared to the competitors of the EU companies which are producing around the globe.

KEYWORDS

Innovation, mining, raw materials industries, European Union, Roman Stiftner, World Mining Congress

INTRODUCTION

Modern life today is totally unconceivable without minerals – infrastructure, renewable energies, eco-mobility, computers, mobile phones etc. – all are heavily dependent on the availability of mineral raw materials.

Mining and processing of mineral raw materials has a long tradition within the EU and especially also in Austria. Although, Europe has huge reserves of sand and gravel and reserves of numerous industrial minerals, concerning metals, however, despite of its numerous excellent recycling facilities, Europe unfortunately is importing most of its demand (http://www.bmwfw.gv.at/EnergieUndBergbau/PublikationenBergbau/Documents/WMD2015.pdf)

Today mining and the raw materials industries are modern, innovative, high-tech sectors that ensure employment and are an inevitable basis for the whole economy of the EU-member states respecting the environment and taking utmost care of health and safety of our employees. The future development of these industries, however, is heavily dependent on research and development.

Innovation and thus R&D are therefore key for the future industrial development of the whole European Union. Besides others the aim of the EU is a re-industrialization – the goal is that industry's share of GDP should be 20% by 2020 (http://ec.europa.eu/growth/industry/policy/renaissance/index_en.htm).

Therefore there have to be the appropriate framework conditions to support R&D and thus to foster innovation. Neither an appropriate raw materials policy, nor supporting research and innovation programs will be sufficient to reach innovation if there is not as well an appropriate legal framework ensuring a level playing field compared to the competitors of the EU companies which are producing around the globe.

The EU Raw Materials Initiative – based on Resource Availability and Recycling

Due to emerging constraints in the access to raw materials within the EU as well as on the world market, which are crucial for the growth and the competitiveness of the EU, the Commission adopted in 2008 and re-newed in 2011 its so-called "Raw Materials Initiative" (http://ec.europa.eu/growth/sectors/raw-materials/policy-strategy/index_en.htm). Its goal is a strategy to tackle the upcoming challenges with a three-pillar approach ensuring:

1. Fair and sustainable supply of raw materials from global markets;
2. Sustainable supply of raw materials from sources within the EU;
3. Resource efficiency and recycling.

It includes all raw materials necessary for the European industry except materials from agricultural production and energy raw materials.

Discussing the topic with external stakeholders and the general public, it has to be repeated again and again, that there is no "resource depletion" of mineral raw materials. Most of the mineral resources are abundantly available around the globe, but they are unequally distributed. Nevertheless for the further growth of an economy the access of the industries to these raw materials is paramount.

On the other hand we have to keep in mind that due to different constraints resource efficiency and recycling will be an inevitable part of the future raw materials policy. They will, however, not be sufficient for a sustainable supply of the world with raw materials in the future. Thus, mining will remain a necessary part of the supply chain. Therefore the EU chose the above 3 pillars-approach for their raw materials initiative.

Research and Development - the Inevitable Factor of Future Growth

Innovation and R&D are part of all of the three pillars-approach of the EU. To name only few of them, Horizon 2020, the European Innovation Partnership on Raw Materials, the EIT Raw Materials or SPIRE PPP are supporting R&D within the European Union.

Horizon 2020

Horizon 2020 (https://ec.europa.eu/programmes/horizon2020/) is the biggest EU Research and Innovation program ever with nearly €80 billion of funding available over 7 years (2014 to 2020). The program follows a challenge-based approach, focusing on three main themes: excellent science, industrial leadership and societal challenges.

In order to tap the full potential of primary and secondary raw materials and to boost the innovation capacity of the EU raw materials sector a number of challenges along the entire raw materials value chain will be addressed in the Raw materials part of the Societal Challenge 5: "Climate action, environment, resource efficiency and raw materials".

The calls in the 2016-2017 program focus on securing the supply of minerals and metals through sustainable innovative production technologies for primary and secondary raw materials. They focus on new solutions for sustainable production of raw materials, addressing issues like sustainable selective low impact mining, new technologies for the enhanced recovery of by-products or new sensitive exploration technologies. Calls for raw materials innovation actions address the issues of intelligent mining on land, processing of lower grade and/or complex primary and/or secondary raw materials in the most sustainable ways and sustainable metallurgical processes. Other parts of the 2016-2017 program concentrate on raw materials policy support actions and raw materials international co-operation or the Era-Net Cofund on Raw Materials.

Also a comprehensive set of Research and Innovation actions of the European Innovation Partnership (EIP) on Raw Materials will be funded by Horizon 2020. Concrete targets include the launch of up to 10 pilot actions to promote technologies for the production of primary and secondary raw materials, finding substitutes for at least three applications of critical and scarce raw materials, building up the EU raw materials knowledge base, creating better framework conditions for raw materials in the EU, and establishing proactive international cooperation with the third countries.

European Innovation Partnership on Raw Materials

European Innovation Partnerships (EIPs; http://ec.europa.eu/research/innovation-union/index_en.cfm?pg=eip) are one approach to foster research and innovation within the EU.

An EIP is driven by special challenges with a focus on societal benefits. It reaches over the whole value chain of research and innovation and brings together stakeholders at EU, national and regional level. Their goal is to coordinate and thus streamline existing initiatives as well as new ones whenever necessary.

One of the five EIPs' that are active at the moment, is the European Innovation Partnership on Raw Materials. Its aim is to bring the European Raw Materials Initiative into action and thus to help to raise the re-industrialization of Europe to 20% per 2020 by securing the access to raw materials as well as a sustainable supply of the EU economy with raw materials.

The EIP on Raw Materials developed a so called Strategic Innovation Plan with 95 actions – technological as well as legal. They reach from R&D to changes in the policy framework and help implementing Best Practices as well as creating a knowledge base about raw materials. Last but not least they include also International cooperations. The EIP-actions might be financed by the specialized section of H2020.

Partners from the entire EU, from different areas like e.g. academia, industry and public authorities and other interested stakeholders, and the whole value raw materials value chain, define, propose and execute so called "commitments" that contribute to the aims of the EIP in the different action areas of the Strategic Innovation Plan. Therefore, the EU Commission had its first call for commitments in 2013, which lead to 80 commitments in this area. The second call was performed in 2015/16.

European Institute of Innovation & Technology - EIT Raw Materials

The EIT Raw Materials (http://eit.europa.eu/eit-community/eit-raw-materials) was established on 9. December 2014 as a Knowledge and Innovation Community (KIC), integrating multiple disciplines along the knowledge triangle of business, education and research. This KIC will cover the whole raw materials value chain: sustainable exploration, extraction, processing, recycling and substitution. It is a strong European network of 115 partners - big industrial players as well as small innovative companies. They all have one ambitious aim: turning raw materials into a major strength in Europe.

Mining is completely different now to how it was 20 years ago. Now we talk about intelligent mines that require completely new competencies for our industry.

A KIC partner has obvious advantages:

- be part of a very strong consortium with good European coverage from Western to Eastern Europe.
- the opportunity to find synergies due to the diversity of partners, resulting in a high concentration of ideas for the development of innovative technologies and new materials.
- participate in many research programs and initiatives led by EU agendas, like Horizon 2020.

Sustainable Process Industries Through Resource & Energy Efficiency Public Private Partnership - SPIRE PPP

Innovation actions with relevance to raw materials are also targeted by the calls under SPIRE PPP (http://www.spire2030.eu/). SPIRE is the abbreviation for Sustainable Process Industries Through Resource & Energy Efficiency. The specific legal entity A.SPIRE AISBL was founded through the joint efforts of the chemical, steel, engineering, minerals, non-ferrous metals, cement, ceramics and water sectors in 2012.

SPIRE aims to deliver new solutions to improve resource and energy efficiency in the process industry and along the value chain by 2020. SPIRE addresses the whole value chain – from different types of feedstock, including renewable feedstocks, through industrial transformation into intermediate and end products.

Legal Framework Conditions for the EU Industries –the Example of the EU Climate and Energy Policy

Besides of innovative ideas in these industries the companies do also have to face very high levels of environmental and health and safety protection. These often differ enormously from the constraints competitors outside the EU are confronted with. Nevertheless EU and Extra-EU-producers are competing for the same customers on the world market. We therefore urge for a level-playing field around the globe in legal obligations, which is not the case nowadays in many areas. To give only one example: the EU Climate and Energy Policy.

The EU Climate and Energy Package

The European Union aims to limit the increase in global temperature to a maximum of 2 degrees Celsius. As part of the efforts required by developed countries as a group the EU has decided a long term target: by 2050, the EU aims to cut its greenhouse gas emissions substantially – by 80-95% compared to 1990 levels. In order to reach this very ambitious goal the EU passed the EU Climate and Energy Package with several intermediate goals. At the end of October 2014 the 28 Members-States decided the "2030 EU Climate and Energy Package" containing three objectives:

1. Emission target: The reduction of emissions of greenhouse gas by 40% by 2030 compared to 1990 (only binding target at the European Union)
2. Renewable energy target: 27% share of renewable energy sources in final energy consumption against 14% today with flexibility given to Member States.
3. Energy efficiency target: 27% reduction of final energy consumption compared to business as usual through improved energy efficiency.

These targets are mandatory for the member states of the European Union. So far the EU has managed to reduce its emissions of greenhouse gas by over 18% since 1990 – due to economic crisis and the small amount of economic growth in the last years.

For the European industry the EU Climate and Energy Package means in the long run to replace fossil fuels by regenerative and renewable sources and to improve its energy efficiency. Many energy intensive industries have greenhouse gas emissions resulting from fossil fuels as energy resource and from process emissions. Fossil fuels used for energy resource can be replaced by renewable energy sources in many cases, but will be difficult for high temperature furnaces. 'Process emissions' means greenhouse gas emissions other than combustion emissions occurring as a result of reactions between substances or their transformation for use as product or feedstock and cannot be avoided or reduced. For example the steel-industry and the refractory industry have high shares of process emissions which cannot be reduced.

EU – Emission trading system

The EU – Emission trading system is an instrument to reduce the greenhouse gas emissions and to reach the long term emission targets. There is a cap for emissions from power stations and other fixed installations in the 28 EU Member States and the three EEA-EFTA states (2013 the cap was set at 2,084,301,856 allowances – 1 allowance for 1 ton CO2 emission). To achieve the target of a 40% reduction in EU greenhouse gas emissions below 1990 levels by 2030, set out in the 2030 framework for climate and energy policy, the cap will need to be lowered by 2.2% per year from 2021, compared with 1.74% currently during phase 3 of the EU ETS (2013-2020).

It is estimated that during phase 3 around at least 48% of allowances will be auctioned. The other allowances are handed out for free, including those for the modernization of the electricity sector in eight Member States, and is based on the existing 'carbon leakage' list – which will be reviewed and reduced for the period 2021 onwards.

Industry sectors and sub-sectors deemed to be exposed to a significant risk of 'carbon leakage' receive a higher share of free allowances because they face competition from industries in third countries which are not subject to comparable greenhouse gas emissions restrictions. But at the end the steel industry, classified at very high risk of carbon leakage, has to buy more than 50% of emission allowances in 2030 which leads in additional costs (Billions of EUR) and eroding incomes and funds for investments.

Energy intensive industries are facing three main difficulties in the European Union:

1. high energy and labor costs in the EU
2. high energy costs, increasing the share of renewables will increase the energy cost
3. increasing costs for greenhouse gas emissions

Therefore energy intensive productions in the European Union are facing a steadily increasing cost pressure and steadily increasing cost gaps compared to productions outside the European Union. Sectors with high shares of process emissions without any “brake through technology” with sharply decreased process emissions – e.g. the refractory industry – are particularly badly affected by the steadily increasing costs for greenhouse gas emissions.

European Steel industry: overcapacities and cheap imports from China and Russia – Does the Steel industry have a future in Europe?

The world faces a huge oversupply of steel - currently only two-thirds of the steel being produced is actually being used. How did this come about?

China's annual steel production has surged in the past decade from about 100 million tonnes to around 820 million tonnes per year, more than half of the global steel production. China is now by far the world's biggest steel producer, today the EU steel production accounts for about 170 million tons. Austria, which produces almost 8 million tonnes a year, is a minor player in terms of absolute output, but is specialized in high-quality, high value steel products (https://www.worldsteel.org/statistics/statistics-archive/yearbook-archive.html).

China's dramatic economic growth - since liberalisation started in the 1980s - has been one of the key drivers in the global steel market. With China's market slowing over the last years, their steel producers have been looking for export markets, such as the EU.

Additionally over the last two years the dollar and the euro have gained substantially against the ruble, and the price for Russian steel dropped by a substantial margin on the world market. The Russian economy is in recession, which means lower domestic consumption and more tonnes to export (http://www.finanzen.net/devisen/euro-russischer_rubel-kurs).

Continuing low commodity prices and subdued prospects for economic development in the EU result in particularly low steel demand and low market prices for steel products.

At the same time EU steelmakers are facing high costs for labour and energy added by increasing costs for EU climate change policies. The European Economic development suffers since the financial crisis in 2007/08 – the result is an overcapacity of 40 million tons in the European steel industry.

The European steel industry has seen significant automation and computerisation and is not as labour-intensive as it used to be. Most of the European steel plants occupy an excellent position for iron/steel technologies in international comparisons of innovation and competitiveness performance. A robust industrial base is essential for Europe's economic growth, preservation of sustainable jobs and our competitiveness on global markets.

EU trade policies and measures facilitating imports of cheap steel products have the consequence of an increasing import pressure and continuing of distort competition on the European Steel market. On the long run the steel production in the EU will decline, the security of supply is in risk will cause incalculable impacts on the downstream industry.

EU Conflict Minerals Regulation – Responsible Sourcing

Regarding the supply of raw materials from global markets, new EU legislation on responsible sourcing might be introduced soon. This EU legislation proposes to set up an EU system for supply chain due diligence certification of responsible importers of tin, tantalum and tungsten, their ores, and gold originating in conflict-affected and high-risk areas. The aim is to help to reduce the financing of armed groups and security forces through mineral proceeds in conflict-affected and high-risk areas and to provide transparency. Many EU companies acting globally are in most cases already complying with existing international certification schemes. For these companies it is very important that the new EU Regulation recognizes their existing efforts on responsible sourcing and does not impose additional bureaucratic burdens for EU-industries only.

CONCLUSIONS

The EU has decided on an elaborated raw materials initiative. It has put in place a highly sophisticated research and innovation program. But the missing level playing fields concerning legal obligations as e.g. the energy and climate policy of the EU will of course make it very difficult to reach the very challenging goal of re-industrialization as innovation and R&D are the basis of the future industrial development of the EU mining and raw materials industry. Any measures increasing cheap imports not sufficiently covering European production costs will have further negative impacts on the existence of the raw materials sector or even lead to a disappearance of (parts of) the sector from the European market. This might lead to an interruption of the supply of raw materials on the European market and further dependency on extra-European sources, which might not always be available. A cessation of raw materials availability within Europe will therefore have severe negative consequences for the down-stream industries as well and therefore be counter-productive for the goal of re-industrialization.

Therefore we urge for an adaptation of the legal obligations, e.g. the energy and climate, environmental and health and safety policy – also for the benefit of our environment and our employees. E.g. in the world climate policy an international agreement on the level playing field might be an option to give the same chances and opportunities to all producers around the globe.

THE PATH TO MINING EXECUTION EXCELLENCE

The Benefits of Operational Stability

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THE PATH TO MINING EXECUTION EXCELLENCE

The Benefits of Operational Stability

ABSTRACT

The mining industry is at a tipping point in its history, in need of innovation to ensure it remains sustainable and able to better weather challenging economic times. That is why there has been a growing call from leading industry executives over the past several years to bring philosophies and technology proven in other industries into mining.

The pursuit of operational stability will see the mining industry move to formalizing and standardizing their business processes, driven by the need for conformance, predictability, safety, cost management and productivity. Achieving operational stability will require robust planning practices; optimized scheduling; and strict adherence to plan, but mines that can become stable will benefit from reduced costs and improved performance in production.

This paper will discuss how miners can deploy software prevalent in the manufacturing industry to achieve operational stability. It will demonstrate how synchronizing operations across multiple departments and between sites will provide: (1) visibility into the current status of the work underway in the mine; (2) control over business processes and faster reaction times to unforeseen circumstances; and (3) optimize the sequence of mining and support tasks and their resource assignments. It will then examine how this sets the stage for the "Virtual Mine".

The ultimate end goal is the deployment of "A Virtual Mine", or digital clone of the mining operation. The Virtual Mine is a platform for innovation by being the environment to try out innovative, perhaps challenging ideas that would never be explored in real life due to risk or cost of failure. It offers immersive, 3D simulation which allow all ideas, small and large, to be explored, so they can be proved and improved upon before being implemented in the real world.

KEYWORDS

Operational Stability, Productivity, Short Interval Control, Lean Manufacturing, Virtual Mine

Operational Stability is the First Step to Transforming the Enterprise

Mining company profits are being pressured by the uncertainty that remains in the global economy and by a cost and profitability curve that has grown tight over the past twenty years. This is why it is critical Operational Stability – the predictability of expected mine production, costs, and performance levels – be attained. Getting there requires mining and plant processing activities function at higher levels of productivity and efficiency so that conformance plan is always realized.

The quickest avenue to improved Operational Stability begins with reducing the variability in the planning and execution of mining and processing. This requires comprehensive planning, optimized scheduling, and disciplined work management.

Operational stability increases throughput, reduces waste and associated costs, and ensures production and quality targets are met. The key lies in harnessing operational data. While "big data" may be produced in mining in terms of volume, it must become visible, analyzable, and it must be made actionable to executives, mine management, and frontline workers. If it is, the path to mining execution excellence, and eventually business agility, is paved. As we will see, enabling technology is one of the most important requirements to begin the journey. In Figure 1, "Three Examples of Benefits Realized with Operational Stability", we see several real world outcomes that manufacturing and mining companies have achieved.

Figure 1 – Three Examples of Benefits Realized with Operational Stability

Establishing predictability in operations is the first step towards transforming mining businesses in a meaningful way. Without control over operations, attempts at becoming agile may not deliver the desired value. If mining businesses do not understand how healthy their operating processes are (including their inputs, plans, equipment, labor, and supporting activities), *and how well they are functioning in the now*, they will continue to waste resources (capital, equipment, labor, and even the mineral assets). Mining operations that continue to operate in the dark, without information fed back into processes, will continue to see variability in operating efficiency, and planned versus actual production will perpetuate.

Many industries have achieved a level of execution discipline well beyond what the mining industry is experiencing today. Advances in technology have led a number of waves of increased execution performance. The industrial interne is driving the latest wave of execution excellence and setting the standard with which most must meet just to stay competitive.

Decades ago, manufacturing established process and systems to support Operational Stability, setting a foundation for agile decision-making and dramatic transformations. Today, companies from a wide variety of industries can design, simulate, and manage their businesses by leveraging seamless collaborative environments, connecting their operations, employees, suppliers, and even their customers. This technology exists today for mining companies, if they choose to embrace it.

Why Operational Stability is the Foundation for Change

Toyota is recognized as being one of the most efficient companies in the world. It is also one of the most studied since it pioneered key methods of reducing variation, with stability being the first and most critical step, because it underpins everything else that follows. Stability includes the general predictability and consistent availability in terms of manpower, machines, materials, and methods. Under each of these basic building blocks of manufacturing, Toyota tries to establish consistency and predictability[i].

In Figure 2 “Lean Manufacturing Principles”, we see the framework for business transformation followed by Toyota, which ultimately enabled it to become an agile organization, able to rapidly adapt to changing market conditions. Operational stability is the fundamental starting point required for everything that follows.

A former Toyota Chief Engineer identified 7 wastes of manufacturing, all of which apply to mining:

- Overproduction
- Transportation
- Unnecessary Inventory
- Inappropriate Processing
- Waiting
- Excess Motion
- Defects[ii]

Figure 2 – Lean Manufacturing Principles

The Benefits of Stability-enabling Technology: An Example from Manufacturing

When AGCO, the world's largest manufacturer of tractors and third largest supplier of agricultural machinery, decided to build the world's most modern tractor factory capable of producing 20,000 tractors a year, it knew it had an opportunity to significantly improve upon the performance of its existing factors.

AGCO's factories implemented a Manufacturing Execution System (MES) / Manufacturing Operations Management (MOM) platform which captures real-time production data to enable quick visibility to all production activity with drill-down options to the machine level. A production dashboard provides key performance indicators which are updated continuously for management review. The system automatically captures machine / production states (running, downtime, set-up). AGCO's system downloads work orders from their ERP system, which are then updated with production data to form a seamless continuous improvement loop.

With its MES/MOM, AGCO has:

- Documented a doubling of production volume without reducing efficiency, as measured by fixed cost KPIs (efficiency and productivity) or production time and variable cost per unit, resulting in a higher profit margin.
- In its first year improved Overall Equipment Effectiveness by 22% in core machining area.

- Projected that assembly line productivity will increase by 25-30% in the first three years.
- With increased transparency improved allocation and utilization of capital invested for machines.
- Standardized, reliable and objective process performance data such as production times, setup times, and non-productive times has allowed better management of valuable capital equipment and product quality.
- Monitoring, improving and accelerating the machine maintenance processes has reduced downtime leading to a lower operating costs.
- Consistent, automated data acquisition and KPI calculations which provide greater transparency on the shop-floor in support of waste elimination and performance improvements – including comparisons across sites.[iii]

Technology-driven Variation Reduction: A Manufacturing Example

To meet production and quality targets, and control costs, manufacturing has strived to optimize the performance of the business processes that see inputs of material and labor turned into outputs and finally finished products. By monitoring and analyzing processes, manufacturing has found ways reduce variation by making them more efficient and predictable.

A good example is Cummins, a global manufacturer of engines, filtration, and power generation products, who sought to improve production efficiency and to deliver zero-defect products to its customers. To realize this goal, it recognized that technology had a key role to play. Its solution was to employ a holistic, Manufacturing Execution System (MES) to gain real-time visibility into, and greater control over, the performance and efficiency its manufacturing operations.

With its MES, Cummins directs manufacturing processes that are synchronized with product conveyance systems and in-line, order specific quantity measurement, test-cell and camera systems. Its MES also enables Cummins to precisely align both kitting and sub-assembly operations with final assembly processes in a complex assembly environment where no two engineers in a row on the line are typically the same.

The synchronization of its processes has translated into greater efficiency, higher manufacturing productivity and competitive differentiation. It has realized throughput improvements up to 25%, due in part to significant quality improvement and waste reduction.[iv]

Lean Manufacturing Principles and Technology Applied in Mining to Achieve Operational Stability

In equating the old analogy of how to eat an elephant to that of mining, the answer is, of course, to take it one bite at a time and we could consider the same strategy with mining. The question that needs to be considered is how big of bites do we take? And how big of bites do we plan for?

The key causes of instability in mining include:

- Planning: Not all required tasks/activities are planned sufficiently.
- Scheduling: Not all planned tasks are robustly scheduled based upon specifications (sequence, time, duration, tons, grade, maintenance, safety, regulatory compliance, etc.).
- Execution: Not all scheduled tasks are completed or tracked to specifications (sequence, time, duration, tons, grade, maintenance, safety, regulatory compliance, etc.).

By focusing on tactical and operational control, mining companies can achieve increased stability and conformance to plan. While mining faces some factors such as geology and weather conditions that manufacturing does not, a great deal of variation can be reduced in mining activities. If planning and operational data is utilized effectively, it can provide rapid insight into how well activities are performing, enabling fast adjustments as operating conditions change. The analytics it enables will also drive continuous improvement.

Every mine will develop a long term plan in order to ensure its success not only for the current year, but for many years to come. It is an important step in planning a mine that needs to be continually re-evaluated based on the current conditions and a mine will generally re-create its long range plan once or twice a year. Preparing for each fiscal year, every mine will develop a budget which will usually include breaking the planned production down on a month by month basis. At the end of the month, they will compare how well they performed compared to their budgeted expectations.

Meeting that expected budget each month is left to the responsibility of the short range plan. The short range plan will be the highest resolution plan issued to mine operations and will likely only provide detail into the priorities, work areas, or assignments of key mining equipment on a shift basis and to a horizon of several weeks. All of the specific details like task creation, resource leveling, task assignments and sequence are left to the operations supervisors on an ad-hoc basis. If we refer back to our analogy, the short range plan could be seen as the individual chews required to complete each bite. The short range plan allows for the larger section being dealt with to be fully developed, hopefully in tandem with the forecasted plan.

A method for aiding in the smaller bites is to employ the use of what is referred to as “Short Interval Control”. This term refers to a controlled system of planning an operation’s activities in fixed time intervals throughout a shift (LeanProduction.com, 2013).[v] As we seen in Figure 3, “Short Interval Control” is performed several times per shift and combines historical analysis, identification of future risks, and proactive action planning. Additionally one of the key principles of Lean manufacturing is the reduction of waste (Wijaya, Kumar, R., and Kumar, U., 2009)[vi] and combining Short Interval Control with Lean ideas will assist in minimizing the waste when unexpected events occur.

A key feature of Short Interval Control is the use of real-time production data to guide instantaneous front-line decision making. Teams are trained to collect, analyze, and react to this data in order to drive significant performance improvement. The core principle behind Short Interval Control: the past cannot be hanged, but we can learn from it to improve the future[vii].

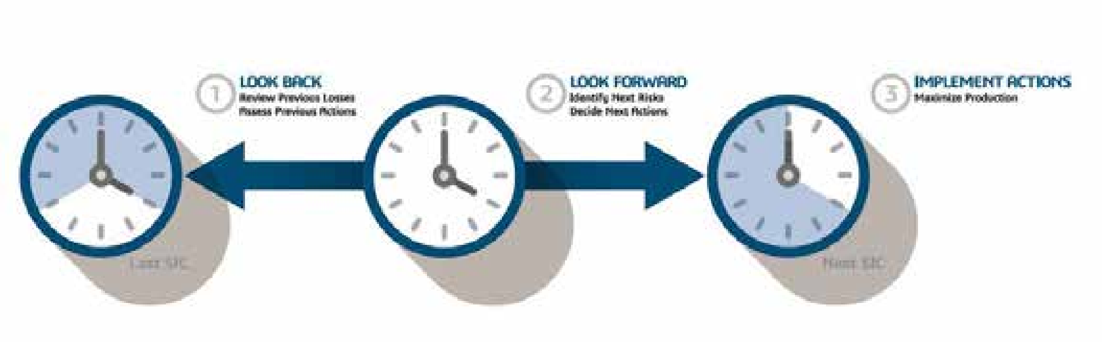

Figure 3 – Short Interval Control

Wastes can be in the form of idle time on equipment, excessive stockpile balances, extra waste stripping, and unnecessary use of consumable materials and all are symptoms of an unstable mining operation. Modeling and enforcing business processes across all tasks in the mine using the principles of Lean manufacturing will assist in developing more predictability in task completion, thus allowing for consistent realization of resource leveling and reduced waste. Software from the manufacturing industry exists to help with the management of the business processes and execution. Such software could help mines in becoming stable and predictable with the added benefits of controlling costs by reducing wastes, improving visibility into safety hazards, and reducing the time required to train miners across multiple functions.

What the adoption of Manufacturing Methodologies and Technologies Would Look Like in Mining

"But, you might ask, what does operating a collection of large mining pits in the Pilbara have in common with producing precision engine components or wheel bearings?

The answer is that that these approaches to process and production are about bigger and more general questions than a specific product or sector. At their heart they are about solving problems and the essential problem is the same for everyone. What is wasting our time, our labor, our workforce skills, our energy consumption, our resources and our money? How do we discover it, isolate it, analyze it and eradicate it?" Sam Walsh – Rio Tinto (Smith, 2012)[viii]

The manufacturing and construction industries have found significant benefits in adopting philosophies such as Lean, TOC, and Six-Sigma. These practices lend themselves well to technology and software has been developed in these industries in order to apply these practices. Technologies that will translate well to the mining industry include Manufacturing Execution Systems (MES), Supply Chain Planning and Optimization (SCP&O), and Business Process Modeling (BPM). To adapt these models to mining, however, the technology used will require interoperability between the systems and a tailored configuration to fully realize the benefits of implementation to a mine site.

While many mining companies have elements of MES/MOM, only a few are just starting to adopt the ISA-95 architecture found in manufacturing (seen in Figure 4, "Manufacturing Industry´s Standard Architecture: ISA-S95 Alignment"). Level three (3) deploys MES/MOM, or in the case of mining what might be called Mining Execution Systems and Mining Operation Management (MOM) platforms.

Figure 4 – Manufacturing Industry´s Standard Architecture: ISA-S95 Alignment

MES/MOM enables superior work management through increased visibility and control over performance. It does this through up-to-the-minute tracking and management of: mining and processing activities; equipment; maintenance; labor; support; and other inputs and outputs to provide:

- The ability to update activities and tasks between scheduling cycles.

- Real-time visibility into capacity, availability, and performance.
- Enhanced ability to manage activities and tasks and priorities to account for changes in production and unexpected events.
- Communication of new and updated work orders instantly wherever they are required.
- Assurance that activities and tasks are completed to specification (sequence, time, duration, tons, grade, maintenance, safety, regulatory compliance, etc.).
- Efficient handover of incomplete activities and tasks between shifts.

An additional level of stability is enabled when scheduling can be connected to the MES/MOM to achieve Short Interval Control. When continuous feedback (Figure 5, "Short Interval Control Driven by Integrated Work Management") loops become part of the scheduling process for production, blending, waste, maintenance, and support schedules, adjustments can be rapidly made to keep production on track. With wireless infrastructure, MES/MOM can gather data from any part of the mine, even underground, and dispatch work orders digitally to employees in real-time.

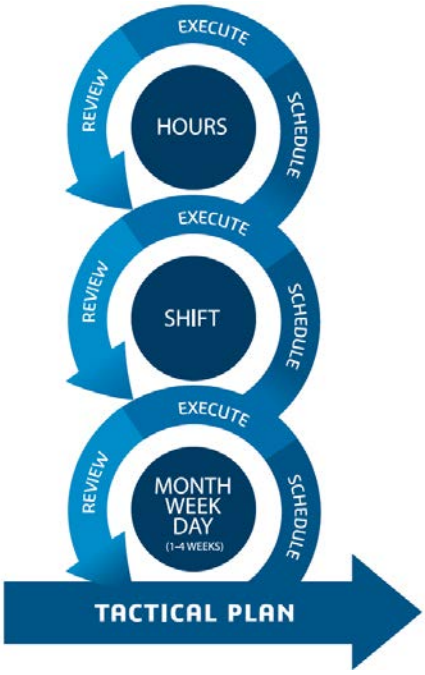

Figure 5 - Short Interval Control Driven by Integrated Work Management

Standardized work processes, when Business Process Modeling capabilities are available in the MES/MOM, are another key aspect that underpins stability. While some mining companies have undertaken great efforts to document procedures, it is difficult to ensure that they are followed if they remain on paper and filed away. Through MOM systems, work instructions, review and approvals become routine and consistent, since they are embedded in employees' job tasks, as shown in Figure 6. This ensures better execution every day, month and quarter.

Figure 6 - Work Instructions Generated in MES/MOM Delivered Digitally and Instantaneously

Lean Manufacturing concepts used in manufacturing are starting to be applied in mining. One example is has seen an early adopter double production and achieved a 44% improvement in unit cost, without adding additional equipment.

Leveraging Manufacturing Supply Chain Planning and Optimization for Production Scheduling in Mining

Typically, short range mine planning does not account for the individual task details required to advance a tunnel, produce ore from a stope or progress a shovel along a pit bench. Planners will often rely on average production rates to estimate production performance without considering the resource availability, resource leveling, and manpower allocations. This will often lead to the creation of schedules that do not fully consider all of the operational factors, such as maintenance schedules or full resource leveling, resulting in plans that are not achievable given the resource availability. On the other hand, scheduling to this level of detail several months ahead of mining often requires a significant investment of a mine planner’s time when using traditional mine planning tools, and will require constant upkeep as work progresses through the schedule. It is not uncommon in the manufacturing industry to use Supply Chain Planning and Optimization (SCP&O) software to plan their logistics, production and support resources down to the individual task level realizing the benefits of improved resource utilization.

Figure 7 – is an example user interface of SCP&O software applied to the short range planning challenges of a mine and provide similar benefits found in the industries noted above. SCP&O tools like provide the short range mine planner with visibility, in the form of Key Performance Indicators (KPI), which provide direct feedback with the implications of each scheduling decision that is made. As the scheduler makes changes to the plan he or she will immediately see if the changes benefit the KPIs of interest and if the change was a good decision. Adapting these tools into the mining practice would give short range mine planners the visibility to consider highly detailed planning scenarios which integrate the availability of individual resources and mine workers, status of active tasks, maintenance schedules, and support activities. This will allow mine planners to optimize the sequence of activities and resource allocations in a more efficient and responsive method.

In addition to the KPI feedback of scheduling decisions, the optimization tools that are available in SCP&O software can be configured to solve highly detailed scheduling puzzles, typically too time consuming to be considered by a human planner. These optimizers could include:

- Level resource allocations for many months ahead of mining;
- Define blending strategies that consider all the quality requirements and minimize re-handled ore;
- Allocate the ideal individuals and resources to tasks; and
- Integrate inside of a short term planning cycle to continually reevaluate the schedule based on actual progress of work being done.

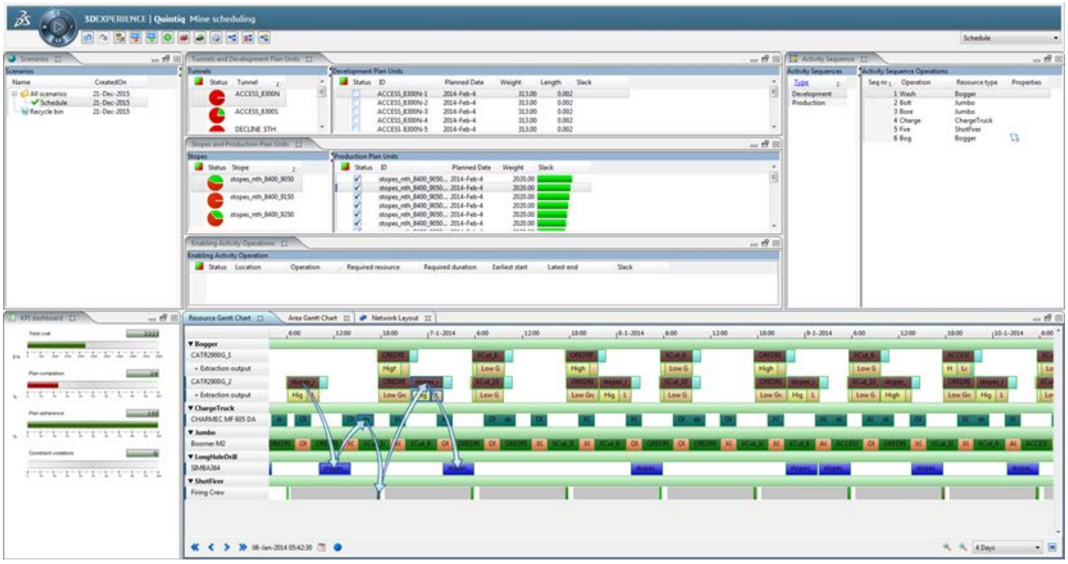

Figure 7 – Example SCP&O software Production Scheduling User Interface

SCP&O software will also allow mines to integrate their planning to include additional tasks not related to mine development or production. This includes tasks that must be scheduled in order to support productive work and include things such as installing electrical, utilities and mine ventilation infrastructure. These tasks need to be considered in the schedule to allocate resources, isolate production tasks from the support task's work area, and integrate dependencies on other tasks.

Another advantage that SCP&O tools provide to mine planners is the inter-operability of these systems between different planning departments or thresholds. By integrating the planning systems together to share a unified data model, mine planners will have visibility into the decisions made by maintenance planners, and can be forced to comply with each machine's scheduled downtime. Alternately, maintenance planners and mine planners will have the ability to collaborate on schedules, sequencing and optimizing planned maintenance tasks as it will fit best in the production schedule. Also, sharing a data model will allow long range mine planners will have an up-to-date forecast of what has been planned in the coming months from which they can develop the long range plan, and short range planners will be able to measure their adherence to a long range schedule.

The features of SCP&O systems which provide value to a mine planner include KPI feedback to planning decisions, optimizers, and integrated planning. Used together, these will allow mine planners to realize benefits of better utilization of the entire mining fleet, reduced time wasted on activities that do not directly benefit the planning horizon, and eliminate wasted operating expenses by not working on tasks that benefit the current horizon.

Adopting Manufacturing's Work Execution Management Systems in Mining

Adapting the MES technologies being utilized by the manufacturing industry and embracing the principles that drive continuous improvement to mining will begin the development of a new industry standard for the development and production tasks that form the repetitive processes found in mining. Not unlike the components of a vehicle being assembled on a production line, the mining process has its various components that need to be able to mesh seamlessly.

Currently, in many mines the processes of managing all of the work being performed is simply discussed during a shift handover, or toolbox meeting at the start of the day. This practice could lead to tasks being delayed, forgotten, or not progressing because of incomplete shift handovers. Irregular work or support type tasks may require the work be performed by personnel with particular qualifications, or the task may require specific instructions and an explicit sequence. Managing qualified personnel and paper work instructions can be time consuming and prone to errors.

In the interest of becoming consistent in the execution of tasks, the MES will allow mining management to develop business process models as standards for execution for repetitive tasks. Mining supervision can assign work to individual miners and specific resources that will interact with the system providing feedback on the progress of the ongoing work in real time. This will allow for the integration of actual tasks with the supply, support tasks, and even logistical information. This direct access to the status of all underway and planned work in the mine creates a stronger sense of control for managers, mine planners and supervisors in their response to unforeseen events such as a delay due to poor ground conditions, or capitalizing fortunate events like an early task completion.

Such access to information can also provide on the floor responsiveness to uncommon, dangerous or high-risk tasks. Miners that are assigned these tasks will have immediate access to information regarding the task's safety hazards, the most current approved engineering drawings and detailed work instructions outlining the process and standard of work that must be performed, as seen in the example of an equipment interface in Figure 8.

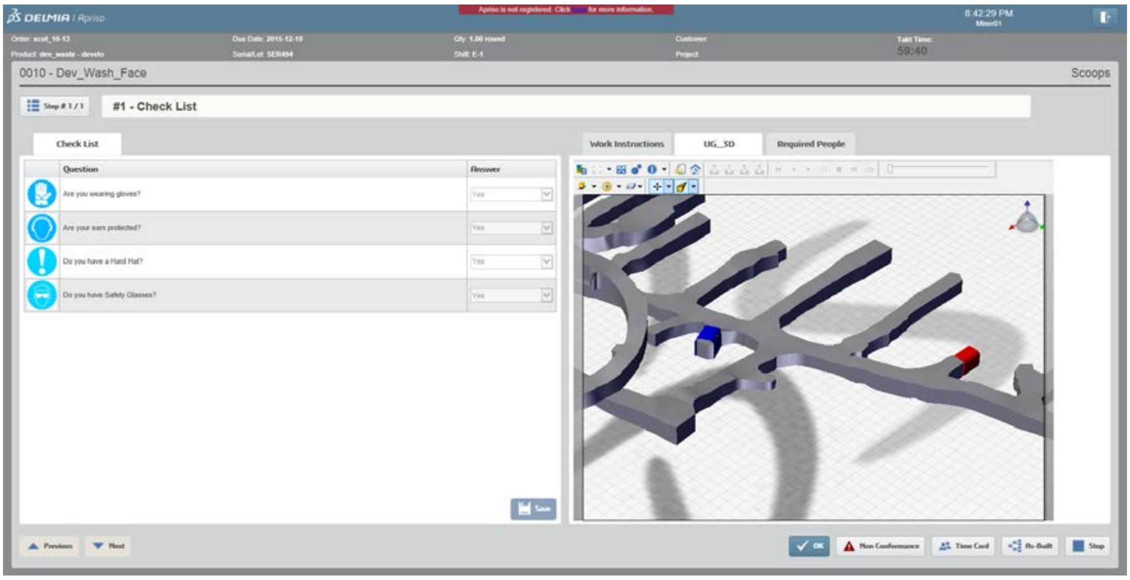

Figure 8 – 3D Work Instructions delivered directly to the equipment operator

Further, use of an MES would provide greater control in the business process improvement. If the KPI's and real-time completion of tasks are better able to match, increased stability and predictability become the base level for performance. For example, if a site implemented a business process in the MES and began to note greater stability and predictability, they could change a detail of the process in a test scenario to create a comparable. If the results demonstrated a change in efficiencies, they could choose to accept or reject the change, as the case may be, but the use of MES in running the test scenario would allow them to control the deployment of the new process. MES provides the synchronicity, visibility and control that will help a mine realize the benefits of reduced time and effort in managing tasks, people and resources. The software allows supervision to maintain momentum on work by managing all tasks to completion and will reduce the variability between production outputs and completed tasks. MES also facilitates mine planners to quickly and consistently react to changing conditions in the mine reducing the losses associated with their delays.

Combining Production Scheduling and Work Management to Achieve Short Interval Control in Mining

Combining the SCP&O and MES software provide the perfect platform for deploying a system dedicated to ensuring the mine is always performing the work that will give the best chance at achieving its production targets. Following a structured process that allows planners to identify and capitalize on opportunities as well as predict losses and respond proactively several times throughout the shift is known in the manufacturing industry as Short Interval Control (SIC) (LeanProduction.com, 2013). Dundee Precious Metals has adopted a Short Interval Control system at the Chelopech Mine in Bulgaria and even early in its adoption they were realizing benefits including increased visibility into the workers in the mine, dispatching, and coordinating activities (Howes and Forrest, 2012)[ix].

Applying the principles of SIC along with the data collected in an MES and the scheduling superiority of SCP&O software will enable mines to consistently meet their targets by increasing the resolution of their planning cycle. With a SIC system, mine supervisors will have visibility into the current condition of the mine including the location of equipment and workers, status of the work being performed, scheduled tasks, crusher rates, stockpile balances, and resource availability.

When unforeseen events do occur, the MES can transfer this information to the SCP&O software, allowing the scheduler to quickly re-allocate the sequence of tasks and the resource assignments. This real-time responsiveness will provide the greatest chance of achieving the shift's objectives in the face of unforeseen events. Once the new schedule is complete it is published back to the MES, resources and workers are informed of task assignments within minutes. Minimizing the time it takes to react to these unforeseen events will allow mine planners to reduce the waste due to unfortunate circumstances, or capitalize on gains in the schedule.

Dundee Precious Metals' Chelopech Mine is a prime example of the use of Short Interval Control where they were able to deploy a system that allows mine supervision to monitor production tasks in real-time and manage a short interval control system all from a central control room. Implementing the Short Interval Control system assisted Chelopech in their objective of increasing production and significantly reducing the mining cost per tonne. (Howes and Forrest 2012)

From Operational Stability to Business Agility and the Virtual Mine

Operational Stability is the first and most important step a mining company can take to begin transforming itself. It not delivers the benefits of greater productivity and conformance to plan; it ensures improvements gained are sustainable. Operational Stability is also the foundation for Business Agility and the Virtual Mine. The Virtual Mine will provide access to all data, from every mining operation. This "big data" will be analyzable to yield patterns, trends, new ideas and game-changing improvements that could not otherwise be discovered.

The Virtual Mine will form a platform for innovation by being the virtual environment to try out innovative, perhaps challenging ideas that would never tried out in real life due to risk or cost of failure. It will offer immersive, 3D simulation which allow all ideas, small and large, to be explored, so they can be proved and improve upon before being implemented in the real world.

With the Virtual Mine comes, full control over all aspects of your business because all the vital data is available, coming from processes that are consistently executed and fully understood. Everyone in the enterprise is connected together, removing the requirement of location on site, for example for mining professionals. Data is at the heart of decision-making as the right data is available when and where it is needed. Every required detail about efficiency is accessible.

The Virtual Mine will improve how mining companies understand, control, and increase the value of their businesses. With data from across all aspects of the operations, it will also be a collaborative engine, with end-to-end traceability and decision-support. The Virtual Mine will provide a single version of the truth by centralizing, sharing and managing all data such as: Resources, Reserves, mine designs, production schedules, analytics, historical data, etc. Through this, it will provide an immediate, global understanding of how well processes are functioning right now, this week, this month, etc. with the ability to rapidly identify problems and introduce improvements. It will also drive advanced simulation which will allow more complex scenarios to be researched

In the Virtual Mine it will be possible to simulate ideas in the virtual world to try out any idea to see what its impact will be in the real world before you make changes, including ideas that would be cost-prohibitive to try out in the real world. On a large scale, for example, one could walk through a virtual version of a mine site before construction, while at the same time review the economic impacts of changes to the configuration of equipment, the plant, or locations of stockpiles or haul roads. Out-of-left-field ideas, such as the utilization of blimps to in lieu of the building of a railway network to connect the mine to a port could also be explored. On a smaller scale, engineers may use the virtual world to test the assembly of a virtual version of a modular plant before it is fabricated and shipped to site.

As a virtual version of a mining business and its operations, the Virtual Mine will enable collaboration in new ways, bringing together the best and brightest minds from inside and outside of the company. A virtual team of experts, located anywhere in the world may work together on problems such as how to best respond to a sudden and unexpected change in commodity demand to answer the question of how the business could maximize the opportunity, without decreasing productivity or escalating costs. They would have all the data and tools that would allow them to test the impacts of rapidly scaling production at specific mine sites and then how different configurations of equipment or utilization of the plant, etc. could be deployed.

In another scenario, a mining company may use the Virtual Mine to examine ways, even in conjunction with its vendors, to examine ways to increase the productivity of equipment by 10%. A virtual team of experts from the mining company and manufacturing could connect and collaborate in the Virtual Mine, utilizing real world data to find ways to modify equipment design or best practices to get more out of existing machinery. As they test things, they would be able "see" the performance in the virtual representation of the mine.

CONCLUSION

Change is not always easy, especially in a complex industry such as mining. The first step is coming to grips with the fact that it is required. While mining is not manufacturing, it does have technologies that can be employed in our industry to enable it to achieve operational stability.

With operational stability comes improved productivity, lower costs and the stepping stone to transformative change for mining enterprises. It is the required starting place for achieving operational excellence and ultimately agility because it delivers the data required to understand how well the processes that make up a mining business are functioning and also enables them to function consistently by reducing the variation in them.

When operational stability is achieved, it provides the data that can be used to make transformative decisions, and also delivers the foundation to deliver it by enabling processes to be rapidly adjusted without negatively impacting productivity or costs. The data it provides can be further leveraged in what we call "The Virtual Mine", to enable business agility.

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WEAR MONITORING: INTELLIGENT INTERIOR COATINGS FOR SLURRY PIPE LINES AND OTHER HIGH WEAR MINING EQUIPMENT

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Wear monitoring: Intelligent interior coatings for slurry pipe lines and other high wear mining equipment

ABSTRACT

Downtime for maintenance required due to abrasion of contact surfaces of pipes, chutes, pumps, and other equipment in contact with ore is a significant cost factor in mining operations. Under today's difficult market conditions, savings achieved by reducing idle time caused by maintenance may be pivotal for the profitability of a mining operation.

Inner surface wear of pipes, pumps, hydrocyclones, mills, or other wear-exposed mining equipment components can be reduced by abrasion protection cladding, coating, and lining materials. Polymer coatings, particularly polyurethanes (PUR) pose a very high performance to cost ratio. In applications like oil sands tailings and hydrotransport, cement transport, as well as station piping in phosphate and copper slurry pump and choke stations, urethane interior pipe coatings and machinery wear plating have shown to increase design life by factors of up to 10, in some cases even more.

This paper describes novel intelligent high performance polyurethane elastomer abrasion protection products that not only exhibit life time extension. These products contain sensors monitoring material wear. Particularly in the case of interior steel pipe coating, this novel family of products represents a disruptive innovation in that data from the sensors inside the coating are transmitted to the outside of the pipe without the need for a hole through the pipe wall. Such intelligent coatings are in operation in oil sands mining since April 2015. Operators are able to reduce downtime by planning replacement shutdowns much more exactly than was possible without intelligent PUR wear protection products.

Experience with other high performance polyurethane wear parts containing wear monitoring sensors based on the same principle will be described here as well.
A summary of test data and case studies on field experience will be presented.

INTRODUCTION

Polymeric Coatings and Linings for Steel Pipe Protection

Carbon steel remains the most widely used material for pipelines in the oil and gas industry, as it has been for a more or less a century.

A number of methods have been developed over the decades to prevent external as well as internal corrosion damage to pipeline steel.

The industry segment of slurry pipelines (transporting multiphase streams that contain solids) is growing strongly, in oil and gas and in the mining industry. The global amount of slurry pipelines in operation is currently passing the 10'000 km mark[1].

Solid particles in slurry streams exert abrasion on interior pipeline walls. In most cases, abrasion is highest at and around the 6 o'clock position since solid particle concentration is highest in the lower part of the pipe due to gravity. Although this does not apply to every case, in the majority of cases where abrasion occurs, this holds true. Wear conditions, particle properties, and particle concentration have a strong influence wear (i.e., pipe wall material loss) in the lower regions of the pipe circumference. Hence, internal wear protection of steel pipes is an important issue in multiphase and particularly in slurry pipelines.

Figure 1 shows a steel pipe segment with extreme wear at and around the six o'clock position.

Figure 1: Abrasive wear on steel pipe

Although a majority of method developments for pipeline protection from corrosion are available; there are methods for abrasion protection that have been developed and are being applied widely today.

If internal erosion and internal corrosion occur concurrently, the combined effect is stronger than the sum of both individual effects. The so-called erosion-corrosion effect can wear down steel pipe very rapidly.

Polymeric coatings and liners are generally well suited both for corrosion protection and for wear protection of pipelines. Since they are not metallic, these materials cannot corrode. Polymeric internal coatings and internal liners in use today normally have a smoother surface than steel (peak-to-valley height profile approx. 30-35 μm in case of polymeric internal flow coatings vs. approx. 85 μm in case of carbon steel).

In the pipeline industry, the term "coating" is commonly used for external coatings of steel pipes whilst the term "liner" describes protective polymer layers inside steel pipes. This is not quite correct technically. The inside of pipes can be protected by polymeric "inner tubes" which are inserted into the pipe mechanically and kept in place by mechanical force. In real life, these mostly are HDPE liners which are pulled through a compression ring, inserted into the pipe, and then remain in place by mechanical force which is created by the polymer relaxing and thereby expanding back to its original size. Or by inserting a heated liner that is soft and flexible and will re-harden upon cooling to ambient temperature.

Polymeric materials that are chemically bound to the interior pipe wall, however, are coatings by nature, regardless of the generalization “liners” used in the industry.

The term “polymeric” encompasses a wide range of materials. Not every polymer coating is suited for every application. Whilst numerous types of polymers are suited as internal flow coating or as internal or external corrosion protection coating, only a very limited range of polymers can be used for internal wear protection.

Desired properties for polymeric internal coatings and linings of steel pipes are:

- High degree of corrosion protection, high barrier function (no penetration of product stream components that could reach the pipe wall);
- Strong adhesion steel (in case of coatings); stable mechanical force positioning the liner inside the pipe (ion case of linings);
- Flow enhancing surface providing low friction;
- Strongest possible abrasion resistance in case of multiphase / slurry applications;
- Strongest possible erosion-corrosion resistance in case of multiphase / slurry applications;
- Compatibility and resistance to product stream chemical composition and product stream temperature;
- Option of in-field repair.

High Performance Polyurethane Elastomer Coatings

High Performance Polyurethanes excellently fulfill all of the requirements for internal coatings listed above.

Besides high abrasion and tear resistance, high barrier function against hydrocarbons and water, as well as very high elasticity, polyurethanes are known for their “building kit” synthesis. By combining different compounds as two main components plus adding different other components (cross linkers, chain extenders, and others), a vast number of different materials can be synthesized. Each one of these different material grades has a different property profile. Figure 2 shows a schematic formula of polyurethane.

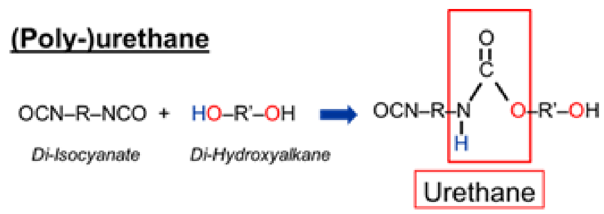

Figure 2: Polyurethane

Interior Pipe Coatings Made From High Performance Polyurethane

Decades ago, numerous attempts were made to apply interior pipe coating made from standard polyurethanes to high wear applications. The rather weak adhesion of standard polyurethane to steel, caused some dramatic cases of coating disbondment. With the high tear resistance of polyurethane elastomers, even localized small-area disbondment in some cases lead to removal of long stretches of coating and clogging up of the entire pipeline system.

Novel high performance polyurethanes exhibit such strong adhesion to steel that the issue of disbondment does not occur, even under extreme conditions. The so-called cold-wall effect is one of these extreme conditions. When above-ground slurry pipelines are exposed to freezing climate and there is an aqueous product stream (e.g., in oil sands mining) inside the pipe, an interior coating with high adhesion to the steel and with high barrier function will not allow water to reach the frozen pipe wall. If water would reach the pipe wall, it would freeze and thereby expand. This expansion would lead to mechanical disbondment of the coating. Initially, blisters would appear and the coating would come off eventually, clogging pipes and pumps. RoCoat™ High Performance Polyurethane Coatings have proven resistance to the cold wall effect over years of application in Canadian oil sands operations where outside temperatures range between -20 and -40 °C in winter[2].

Figure 3 shows a steel pipe internally coated with high performance polyurethane coating for oil sands hydrotransport application and figure 4 the installation of such pipes at a hydrotransport line in oil sands mining at Fort Mc Murray, Alberta, Canada.

Figure 3: Internally coated pipe for oil sands hydrotransport

Figure 4: Installation of internally coated pipes at oil sands hydrotransport line

CONDITION MONITORING OF SLURRY PIPELINES

Challenges with internally coated or lined steel pipelines

For metallic materials there are numerous methods of non-destructive testing and condition monitoring available. However, such testing and monitoring proves very difficult for nonmetallic coatings and linings of steel pipes. The most prominent reason for this is that methods like ultrasound, eddy current, electromagnetic acoustic transducer, and others will receive strong signals from the steel layer which will prevent any (much weaker) signal or attenuation from the internal polymer coating or lining of which wear is to be determined.

As long as internal coatings are thin (below 10-15 mm), there are some modern methods of intelligent in-line inspection that can provide at least approximations of coating wear as well as information on the status of the steel underneath. For thicker interior coatings and liners as they are normally used in slurry pipelines, however, in-line inspection does not work. Any method that tries to measure internal coating or lining wear from the outside must overcome the hurdle presented by the steel pipe wall.

In high wear applications like oil sands and mining slurries, this inability poses a significant threat. It is essential for operators to know how much wear an internal coating has undergone; particularly at or around the 6 o'clock position.
Although High Performance Polyurethane wear protection coatings extend the life time of steel pipes in such applications by factors of 5-10 and sometimes even longer, the point at which the wear limit is reached needs to be known. Until very recently, the only way to determine wear was to shut down the pipeline and physically inspect the inside.

We developed a method to determine internal coating or liner wear from the outside of the pipe. It involves sensors placed inside the polyurethane coating, thereby rendering the coating "intelligent". These sensors do not require a hole through the pipe wall for data transmission to the outside world. Development of this concept took a number of years and extensive testing. So-called "instrumentation spools" with such embedded sensors are now in operation in tailings lines of oil sands mines in the Athabasca basin of Alberta, Canada. Experience gathered since April 2015 shows that the system functions and is able to indicate coating wear. Conditions of this tailings line are:

- Oil sands tailings
- 32" steel pipeline with 1" thick interior RoCoat™ High Performance Polyurethane Elastomer coating
- Flow velocity 5 m/s
- Stream temperature 45-65 °C
- 6 measurement units placed around the pipe circumference (commercial product has 2 rings of 12 units each arranged around the circumference)
- Tailings containing rocks of up to 5" (12.5 cm) diameter

Figure 5 shows a drawing of the instrumentation spools' principle.

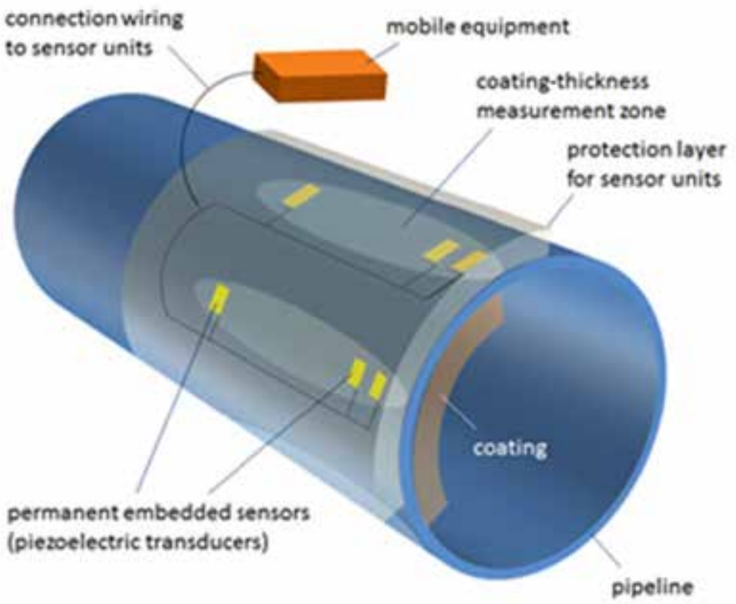

Figure 5: Principle of RoCoat Instrumentation Spools

The embedded sensors do not require a battery or other internal power source. They are placed against the inner pipe wall before the RoCoat polyurethane material is cast into the pipe. Each sensor is placed on top of a microchip. These microchips receive power from an ultrasonic transceiver placed on the outside of the steel pipe. The ultrasonic impulse penetrating the pipe wall activates the sensors and provides enough energy for sensor activation/measurement and for data transfer back to the external ultrasound transceiver. I.e., the chip collects sensor data and then transmits them back to the external transceiver by an ultrasonic impulse.

There are two versions of this system. One requires inspectors to connect hand-held readers to contacts located at the positions where the chips are placed on the inside of the pipe. The inspector's reading device then activates the ultrasonic transceiver and reads out the data.

In the second version, there is a permanently attached readout device, powered by batteries or grid power (depending on location), that transmits sensor readout to a central monitoring station via wireless data transmission.

The graph in figure 6 shows the differentiation of readout signals between polyurethane coating thicknesses in the range of 2 mm to 20 mm (dotted black line ? point of measurement). The actual readout at handheld or remote monitoring screens would, of course, be actual mm values.

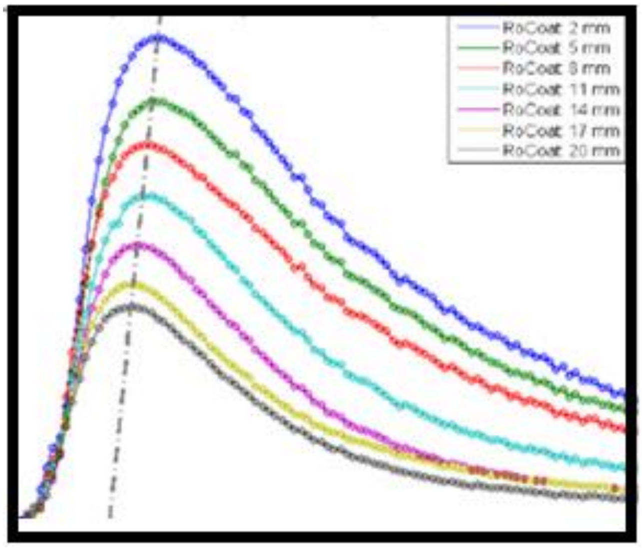

Figure 6: signal differentiation of internal wear monitoring as function of coating thickness

At this point in our presentation at WMC 2016, a demonstration of an instrumentation spool will be given.

With cost of a single shutdown often reaching the million dollar range, even just 2 or 3 months of postponement of a shutdown can mean significant savings for the operator. If the actual wear status of a pipe (or of its internal coating, respectively) is not known, shutdowns need to be performed earlier to reduce the risk of unexpected leaks due to worn pipe. With continuous monitoring by instrumentation spools, shutdown can be timed when they are really needed. Calculations by operators and engineering consultants estimate savings of to 25 % or more of maintenance cost by such a system.

The functionality of this system has been proven in the field. Operators are already capitalizing on the possibility to much more accurately schedule shutdowns and maintenance. Given the high wear resistance of High Performance Polyurethane Elastomer internal pipe coatings, it will be a few years until a statistical amount of data from more than one location will be available so that actual savings can be compared to the engineers' and operators' predictions.

CONDITION MONITORING ON MINING WEAR PARTS OTHER THAN PIPES

The "instrumentation spool" principle described above is applicable to other polyurethane wear parts that are cast onto a metal surface. For parts where the polyurethane product is directly accessible for electrical connectors, the sensor system seated on an ultrasonically activated chip can in some cases be replaced by the same sensors directly connected to a 2-phase wire that is used to power the sensors and to read out signals.

An example is a wear sensor that can be placed into ball mill lifters. In this case, the connecting wires are led through the fastening bolts of the lifters whilst wireless transmitters on the outside of the bolts transfer data to a central receiving station. This system, schematically shown in figure 7, was described at an earlier World Mining Congress[2].

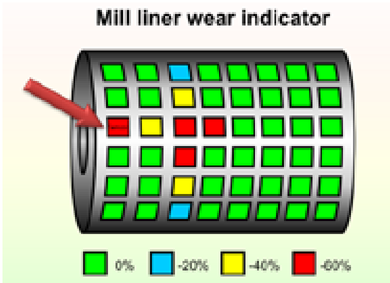

Figure 7: Lifter Wear Monitoring System

"INTELLIGENT" POLYURETHANE ELASTOMER SENSORS

Polyurethane elastomers can be employed to create intelligent materials.

Generally, High Performance Polyurethane Elastomers are dielectric; they do not exhibit electrical conductivity. Hence, they can be used as dielectric core of an electric capacitor.

Since some polyurethane elastomers do quickly and completely revert to their original shape after having undergone mechanical deformation, they are particularly suited as dielectric layer in dielectric elastomer sensors (DESs) if their dielectric and other mechanical properties can be adjusted to the requirements of such sensor systems. Not every polyurethane is applicable in these sensor systems; in fact, most polyurethanes are not. The building kit chemistry allows the synthesis of custom designed high performance polyurethane elastomers. After years of intense research, the authors' team was able to design polyurethane elastomers that ideally fit the specification profile for DESs.

The sensors thus created are thin films and hence can easily be integrated into wear parts, exterior spray coatings, or other products made from polyurethane. In these cases, the sensor literally becomes part of the urethane product. Alternatively, by placing / gluing the sensor onto a substrate (steel, concrete, others), strain (bending, vibration, expansion), impact, and other forces on that substrate can be measured.

Since the sensors are basically flexible films, a high degree of design freedom for their application exists.

Such DE sensors are very well suited for use as continuous monitoring devices.

The film electrodes of individual DE sensors are connected to wires that provide electrical power as well as transmit the signal to an electronic data analyzer.

The size of each single sensor can be designed as required by the application. A sensor can have an area of a few cm² or of a m² or more.

DE Sensors, Properties and Functions

A DES is basically a capacitor made of at least three layers of thin films that are placed on top of each other like a sandwich. The two outer films must be made from an electrically conductive material so that they can act as electrodes. The center film must be made from a dielectric material. The concept of dielectric elastomer sensors is shown in Figure 8.

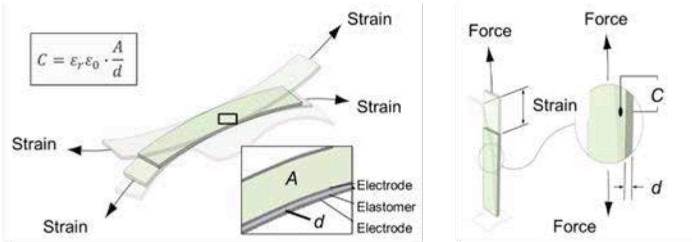

Figure 8: Principle of DE Sensors

If there is a voltage applied to the two electrode layers and the center film is elastomeric, any distortion of this central layer will result in an electrical current. This is caused by a reversed electrostrictive effect. The actual electrostrictive effect comes into play if a high voltage is applied to the electrode layers. In case of a high voltage applied to the electrode layers, the electrostrictive effect causes a pressure to be exerted on the dielectric elastomer layer. The elastomer is not compressible. Therefore, it will expand orthogonally to the direction of the pressure, i.e., the central layer will become elongated. The DE system becomes an actuator. For pipeline monitoring, however, DE sensors are relevant.

In DES trials published elsewhere[3], thin metal films were used as electrode layers. For strain or vibration monitoring, the dielectric elastomer layer will undergo elongation or distortion in any direction. Since high performance polyurethane elastomers used here have elongations at break that can reach 100 % or more, it is important that the electrodes can follow these movements and geometrical changes. Therefore, to make such a DES system feasible, the electrode layers need to be stretchable as well. This does not hold true for metal films.

To make the electrode layers as flexible and stretchable as the polyurethane layer inside the sandwich, new electrodes had to be developed. To obtain a material with an elongation profile similar to that of the high performance polyurethane elastomer DE layer, novel films made from conductive high performance polyurethane elastomers were developed.

High Performance Polyurethane Elastomer DE Sensors

Electric capacitance C of a dielectric elastomer corresponds to the ratio of added electric charge Q and resulting electric voltage V. For the area of overlapping electrodes A and the electrode distance z the following equation is valid:

$$C = \frac{Q}{U} = \varepsilon_0 \varepsilon_r \frac{A}{z} = \varepsilon_0 \varepsilon_r \frac{a \cdot b}{z}$$

With the electric constant being ε_0 and the dielectric number ε_r of the used dielectric material. One particular property of elastomers is their incompressibility. Thus the volume of dielectric elastomers stays constant during deformation. While measuring the electric capacitance C, information about mechanical deformation of the sensor and structure can be collected.

A difference between tensile strain and compression strain sensors has to be made. For a tensile strain DES the length b and the thickness z change during deformation while the width a stays constant. This behavior was simulated for an oblong dielectric elastomer sensor with a_0 = 1 cm, b_0 = 10 cm and z_0 = 80 µm for three different materials with ε_r = 3, 4.5 and 10. Results are shown in figure 9.

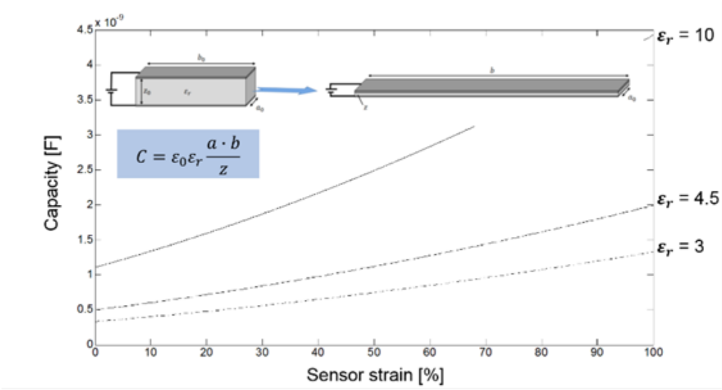

Figure 9: Simulation results for electric capacitance C versus tensile strain for three DESs with a_0 = 1 cm, b_0 = 10 cm and z_0 = 80 μm for ε_r = 3, 4.5 and 10

From the simulation results, an increase of capacitance C with increasing deformation is observed. Furthermore, a stronger change of capacitance C is observed with higher values of permittivity ε_r.

For dielectric elastomer compression strain sensors the electrode distance z_0 is lowered to z during compression strain. The incompressibility of the dielectric elastomer leads to an expansion of the sensor area A. Both effects lead to an increase in capacitance C. Figure 10 shows simulation results for three DE compression strain sensors with A_0 = 1 cm^2, z_0 = 80 μm and ε_r = 3, 4.5 and 10.

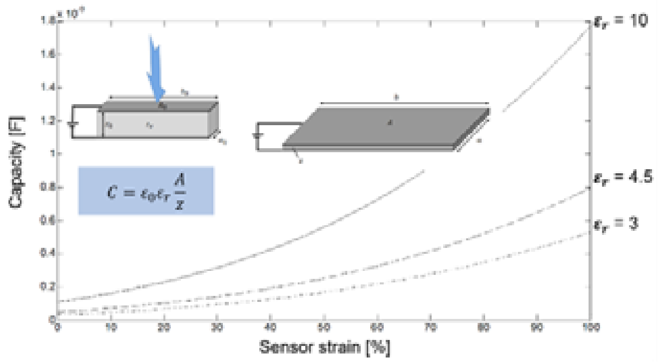

Figure 10: Simulation results for the electric capacitance C versus compression strain for three DES with a_0 = 1 cm, b_0 = 10 cm and z_0 = 80 μm for ε_r = 3, 4.5 and 10.

Potential of DE Sensors in Mining Applications

Whilst there are a number of applications where our High Performance Polyurethane DE sensors have already been tested, their potential for mining is just being explored.

An example is a pipe joint (see figure 11) where the sensors are applied in the form of an external “coating” (oversprayed with a polyurea coating layer). Bending tests of this pipe showed reliable and reproducible strain measurement with clear indication of bending angle and location of the sensor.

Figure 11 shows the actual pipe used in these experiments and its “intelligent coating” with embedded DE sensors and figure 12 shows the sensor readings from a bending experiment of this pipe.

Figure 11: Pipe with “intelligent coating”

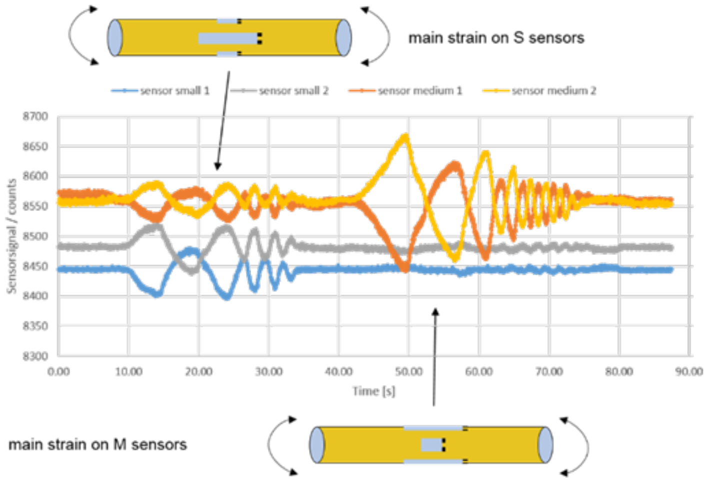

Figure 12: Bending test of pipe prototype and resulting DES signals

Generally, such sensor films can be applied to mining equipment to prevent damage and catastrophic failure due to strain, impact force, or wear on any equipment, be it made from metal, polymers, or other materials. Forces acting on e.g. shovel arms, conveyor rollers, other rotating equipment can be monitored.

Indirectly, DE sensor systems can also be used to measure wear of polyurethane parts, this research and development is still ongoing.

CONCLUSIONS

High Performance Polyurethanes and Polyureas have excellent corrosion, abrasion, and erosion-corrosion protection properties and are therefore used as interior pipe coatings in high wear applications and as exterior pipe coatings for corrosion protection.

Embedded sensors in internal polyurethane coatings can help operators of slurry pipelines save significant maintenance cost. Successful field trials have been ongoing for more than a year now.

Similar embedded sensor systems can be used with polyurethane wear parts other than internally coated pipes, again reducing maintenance cost and down time.

The building block chemistry of polyurethanes and polyureas allows design of Dielectric Elastomer Sensors that can be considered an intelligent elastomer. Such intelligent elastomers could be used to monitor critical equipment in mining.

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