Rock Characteristics and Ball Mill Energy Requirements at Goldfields Ghana Limited, Tarkwa Gold Mine (TGM)*

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Abstract

Mineral processing plants often experience changes in throughput; blending patterns, and rock properties. These changes can have great impact on milling operations. As mining progresses, new deposits are discovered, which may have different characteristics from the designed plant parameters, a situation requiring re-assessment of the plant operating parameters. Goldfields Ghana Limited, Tarkwa Gold Mine (TGM), processes ores which occur in conglomerate reefs. The competence of the ore was observed to increase with increasing mining depth. Other rock properties such as porosity, micro-cracks and gold dissolution were also observed to change with depth. It therefore became necessary to conduct a study to ascertain the effect of changes in rock characteristics on the performance of the existing ball mill. The study characterized the mineralogy and rock characteristics of the ores being mined from three pits and the results were compared with design parameters. The parameters examined had deviated from the design; Work Index (WI) for example was lower than design and required simulation and adjustment. Samples taken at the same depth from each of the pits showed that Akontasi Pit has the most competent ores, followed by Kottraverchi Pit and then Teberebie Pit. Furthermore, throughput was the most sensitive variable and easiest to manipulate to achieve the energy draw required. Simulations showed that a plant throughput of about 1665 t/hr, instead of the current value of 1500 t/hr would be most suitable as an energy draw solution.

Keywords: Work index, Power draw, Computer simulation

1 Introduction

Size reduction is very expensive and the energy requirements as well as cost per tonne of ore comminuted increases from blasting (0.43 kWh/t) through crushing (3.24 kWh/t) to grinding (10.0 kWh/t) as the highest (Higgins 1998; Elorantra, 1997) while efficiency decreases. Thus, milling is the most expensive and yet the least efficient. In a comminution circuit, overall power consumption and power draw during milling are very important parameters (Siddall *et al.*, 1996). Since milling is generally inefficient, it is necessary to maximize the use of energy that is available to the comminution circuit. A milling circuit is generally designed based on grindability and abrasive index tests carried out on samples during the plant design stages of the mine (Levin, 1992; Comeau, 1996) and the expected tonnage to be processed.

Plant designing is done in some cases, without considering the blending patterns of the processing plant and the different ore horizons to be encountered during mining. Thus at several periods in the operation of a mineral processing plant, changes in throughput, blending patterns and rock properties have a great impact on milling operations. Also, new deposits may be discovered after the plant has been set up and the new deposits may have different characteristics from what the plant was designed for. Such situations require re-assessment of the grindability and work indices of the different ores encountered. Re-assessment will aid in developing a blend system and predict appropriate tonnages for the available ore types in order to fit into the available plant design (Elorantra, 1995; Elorantra, 1997; Fuerstenau *et al.*, 1995; Napier-Munn *et al.*, 1999).

Several mining companies in Ghana process ores which occur in conglomerate reefs consisting essentially of both rounded and angular quartz pebbles of varying sizes in a matrix which is highly silicified (Kesse, 1985). Faulting, fracturing and primary lithological permeability also modify the depth to porosity relationship.

With prolonged operation of Goldfields Ghana Limited, Tarkwa Gold Mine (TGM); its mining pits have become deeper and ore competence has increased drastically. Currently, the mine experiences increased downtime due to mechanical and mill part breakdown, coarse grinding and increased scats generation and throughput under-achievement. These are indicators of an overstressed mill.

All these factors contribute to vast variation of current plant energy requirements and plant design energy calculations and requirements. Extra plant energy usage leads to higher maintenance costs, increased plant downtimes and consequently revenue loses to the mine (Kanchibotla, 2000; Forbah and Amankwah, 2010). Since the Work Index (WI) gives an idea of the energy usage during grinding, its determination helps to forecast

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energy requirements as ore exploitation goes on and new ores are encountered.

While ore breakage rate is dependent on rock strength, mill feed size depends on in-situ rock structure, rock strength, blast intensity and primary crusher specifications and parameters (Simkus and Dance, 1998). The first step in developing an accurate mill throughput model is to have in-depth knowledge of the geological parameters that define the rock strength and structure. Among other parameters that determine the competence of ore are Uniaxial Compressive Strength, Point Load Strength, Crushability Index, Abrasive Index and Work Index.

This work investigates the variation in the BWI profile with increasing pit depths and design calculations, so as to maximize production at a given grind size while minimizing the energy consumption per tonne of material milled. The data obtained will be used to estimate variations of work index with different ore types and depths and develop a proactive strategy to optimize the grinding circuit energy at the Mine.

2 Materials and Methods Used

2.1 Sampling and Sampling Preparation

Twenty four (24) samples weighing about 50 kg each and average particle size of 50 mm were taken from various Pits (Akontasi, Kottraverchi and Teberebie). Let A represent Akontasi, B represent Kottraverchi and C represent Teberebie. The sample names and depth matrix were recorded. Some of these samples were taken from the same pits at different depths while others were taken from the same pit but, from different reefs. This would help with comparison to establish the changes with ore depth, pit and reef variations.

The depths were measured in meters above sea level; a sample described as GREEF69RLC, was taken from the G reef of Teberebie (C) Pit at 69 meters above mean sea level. Mining is done in 3 m flitch of a 6 m bench. Duplicate samples were taken from the same pit, reef, and depth for quality assurance.

2.2 Mineralogical Analysis

Mineralogical analysis was conducted to determine the main ore and rock forming minerals present in the various samples. Samples were studied megascopically and also from thin and polished sections using Leitz optical microscope under both transmitted and reflected light.

2.3 Porosity, Relative Density and Ore Competence Analysis

The porosity tests on the rock samples were done by applying the Saturation and Buoyancy Technique as prescribed in the relevant sections of the International Society for Rock Mechanics (ISRM) Suggested Methods (Anon., 1981).

2.4 Abrasive Index Determination

The abrasive index was determined using the Julius Kruttschnitt abrasive index method (Napier-Munn *et al*., 1999). Three (3) kg of -55 + 38 mm ore particles were milled for 10 min at 53 rpm. The mill had both diameter and length, 0.3 m and equipped with 10 mm lifters. The products were screened and the index determined from the size analysis curves.

2.5 Work Index Determination

The as-received samples were prepared to all passing 3.35 mm. The -106 µm particles were also removed and analysis was conducted on the -3.35 mm + 106 µm material. The tests were performed in a standard Bond ball mill with weight of material equivalent to 700 cm^3. After every grinding cycle, the mass of the minus 106 µm fraction was replaced with fresh feed to keep the mass of the mill feed constant. This cycle was repeated until the net mass of minus 106 µm material produced per mill revolution attained equilibrium with a circulating load of 250%. The Work Index (WI) of each sample was determined using Equation 1 (Napier-Munn *et al.*, 1999):

$$\mathrm{WI} = \frac{44.5}{(\mathrm{P_1})^{0.23} \times (\mathrm{Gbp})^{0.82} \times 10\left(\frac{1}{\sqrt{P_2}} - \frac{1}{\sqrt{F_2}}\right)} \qquad (1)$$

where WI is the work index, P_1 is the screen test size in microns, Gbp is the net grams undersize per revolution, P_2 is the 80% product passing size in microns and F_2 is the 80% feed passing size in microns.

2.6 Plant Data Collation, Analysis and Computer Simulations

Data from the month end reports of the Mine for six (6) months before the test work, were collected and collated. The following plant data and parameters were collected on monthly basis and investigated to ascertain the possibility of establishing certain trends:

(i) ROM pad fragmentation;
(ii) Crusher availability;
(iii) Throughput;

(iv) Plant power draw (kWh/t);
(v) Steel ball consumption (kg/t);
(vi) Scats quantities;
(vii) Ore blends;
(viii) Grind achievement; and,
(ix) Cyclone performance.

These parameters helped in establishing whether there were any abnormalities that could be traced to the variations in ore characteristics being investigated. The plant is a dynamic set up and therefore sufficient data was necessary to convincingly determine the effect of a particular parameter on the process.

3 Results and Discussion

The current study aimed at determining the changes in mineralogy, grindability, rock characteristics such as porosity, abrasive index, and; Work Index of the various ore types and pit horizons at TGM.

3.1 Mineralogical of Analysis

Based on the initial megascopic analysis, three groupings were identified; hard, medium and soft ores. The groupings were not related to the depth at which the samples were taken. Several samples of the three ore groupings were characterised.

The main rock types identified were quartzite and conglomerates. Quantitative estimates indicated that the ores contained 75-80% quartz, 7-10% feldspars (highly altered plagioclase) and 5% mica which were present as biotite and muscovite. The quartz was fractured in some of the sections, possibly as a result of metamorphic pressures or due to blasting. The plagioclase feldspars and micas were altered to chlorites and sericites in the matrix of the rock. The matrix also included fine-grained quartz, hematite, and minor carbonates; altered minerals predominate in the matrix. Test with dilute hydrochloric acid was positive in some of the samples indicating the presence of carbonates in hand specimen identification, but none was identified under the microscope.

Reflected light microscopy identified various amounts of pyrite as the main sulphide mineral. These are principally sub-hedral to amorphous grains of pyrite, found aligned along minute fractures or partially disseminated in the rock. Several grains showed etched-out (corroded) surfaces, indicative of reactive processes. Minor hematite and ilmenite were also observed in some of the samples. In general, the samples had similar minerals but, the deep seated samples were fresh while the near surface samples were weathered. The photomicrographs of the samples analysed are presented in Figs. 1 to 5.

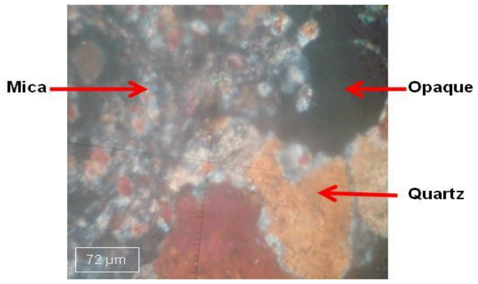

Fig. 1 Photomicrograph of Thin Section of Hard Rock Sample showing an Opaque Mineral, 'Weathered' Mica (Chlorites), and Quartz.

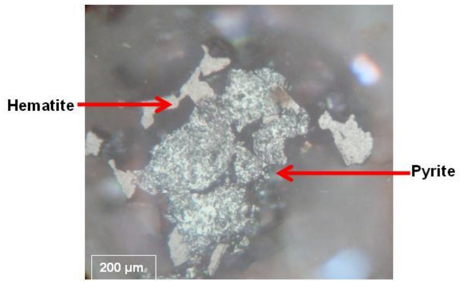

Fig. 2 Photomicrograph of Hard Rock Sample in Oil, showing Corroded Pyrite and Hematite.

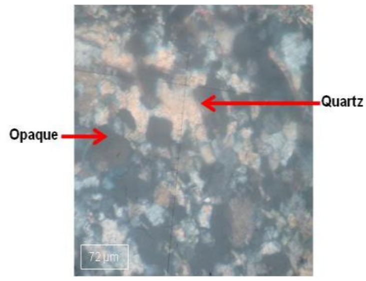

Fig. 3 Photomicrographs of the Medium Seated Sample showing Aggregations of Sub-Hedral Quartz, in a Finer-Grained Matrix of Quartz, Opaque Minerals, Chlorites and Quartz

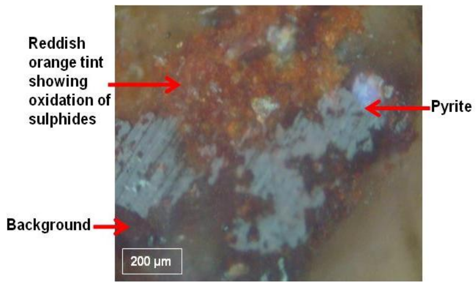

Fig. 4 Photomicrographs of the Medium Seated Sample showing Slightly Deformed Pyrite replaced by Hematite (grey with a bluish tint).

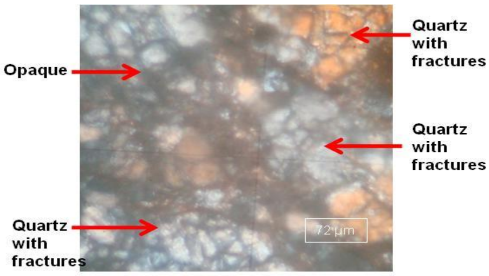

Fig. 5 Thin Section Photomicrograph of the Weathered Rock Sample showing Highly Fractured Quartz

The mineral associations in the ore have influence on their comminution characteristics. Analysis of the work index results (presented in section 3.3) against the mineralogy show that the hard rocks containing about 80% quartz, without weathered components had a high work index with low grindability. The soft ore type taken from higher areas of the pit were more porous, and due to weathering had a relatively low work index and higher grindability. Conglomerates by their nature contain quartz pebbles cemented together by fine material. In a milling circuit, the cementing material may be milled easily while the pebbles being harder may generate scats and grit.

3.2 Porosity, Relative Density and Ore Competence Analysis

Porosity data analysed in relation with the ore competence (Table 1) shows that weathered ores had the highest average porosity of 6.32 and that of the hard ores were 1.66. The ore from the medium horizons has a porosity of 3.23. Porosity has a functional relationship with micro-cracks and fractures; hence the weathered/soft material had higher porosity.

Table 1 Ore Competence Properties

Ore hardness	Ave. porosity	Rel. density	Abrasive index	WI
Soft	6.32	2.45-2.47	1.30	<11
Medium	3.23	2.58-2.61	0.45	11-12
Hard	1.66	2.62-2.68	0.39	>12

From the mineralogical analysis, the main mineral was quartz with relative percentage of 80% and a theoretical density of 2.67. Thus, the hard rocks with relative density between 2.62 and 2.68 show a good correspondence with that of the main mineral.

Though the weathered (softer) ore also contained quartz, the presence of fractures and micro-cracks increased the porosity and hence the bulk density leading to a decrease in the relative density. The abrasive indices show that both the medium and hard ores had high abrasive resistance according to the classification given by Napier-Munn *et al.* (1999). The weathered material had low resistance to abrasion.

3.3 Work Index

The design parameters for the milling circuit under consideration were:

Work index – 12.65 kWhr/t;

Tonnage – 1500 t;

Motor power – 15.4 MW;

F_{80} - 3800 microns; and

P_{80} 106 microns.

The maximum, average and minimum WI values were 12.52, 11.39 and 8.59 kWh/t respectively. All the work indices obtained were lower than the design value (12.65 kWh/t). Fig. 6 shows a graph of WI values in descending depth order plotted against the sample names.

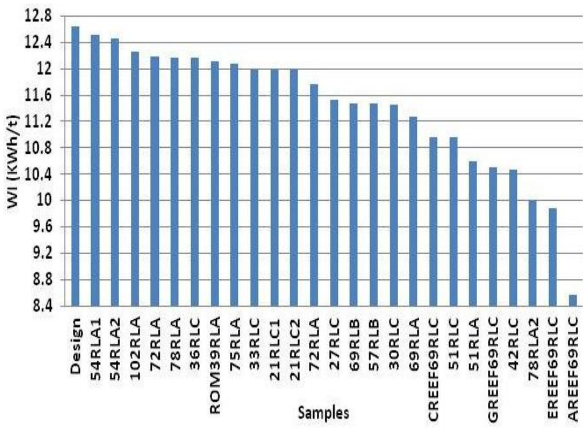

Fig. 6 Work Index of Samples against Mill Design WI

3.3.1 Ore Depth Versus Competence

Ore competence and invariably WI increases with depth. All three pits were plotted according to depths. Data from Teberebie Pit is plotted in Fig. 7. Akontasi Pit and Kottraverchi Pit had similar observations to Teberebie Pit. 21RLC was the least in WI. 36RLC had the highest WI (12.17 kWh/t). A geological fault or fold could elevate a particular section by a few meters and therefore sampling at the two ends at the same depth would produce different results. Other factors worth considering were natural (due to heavy faulting) and artificial (closeness of sample to blasting charge, amount of fragmentation charge used) micro-fractures of the sample.

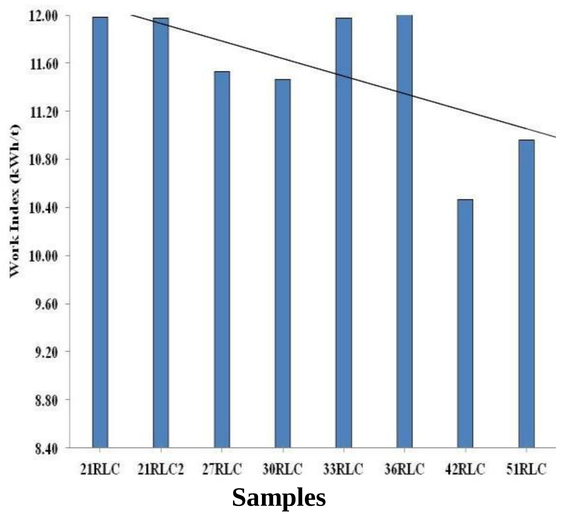

Fig. 7 Work Index Comparison from Teberebie Pit

The ores from Akontasi Pit showed comparatively higher work indices for the same depths. The WI obtained were comparatively higher than that of Teberebie Pit with values between 12.52 kWh/t and 10.00 kWh/t corresponding to samples 54RLA and 78RLA respectively. Ores from the same depth of the pit showed high differences in WI.

For example, 78RLA1 and 78RLA2 were all taken at 78 RL but showed 12.17 kWh/t and 10.00 kWh/t respectively with 102RLA showing again a rise in WI (12.26 kWh/t) at a depth of 102 RL. 102RLA was taken at the highest elevation but it rather had the highest WI and energy value. In all, 54RLA had the highest work index (12.52 kWh/t) at a depth of 54 RL. Again Akontasi Pit samples were consistent with the depth/competence theory. Akontasi Pit is a ridge and hence the likelihood of faults and folds. The ridges could offer explanation for the high variation of competence with depth.

A value of 12.52 kWh/t was the closest WI value to the design mill WI of 12.65 kWh/t. Since all the WI were lower than that of the design, it could be deduced theoretically that the ores should not offer challenges for the processing plant in terms of power draw, wear rate and the other design parameters. However, this is not the case. Plant operations confirm that Kottraverchi Pit ores were the most competent and hence anytime those ores run through the plant, there was increased energy draw, scats production and wear rates. Since, these observations are not WI related, other design variables may be considered.

3.3.2 Same depth, pit variations across reefs

Interviews with mine geologists revealed selective mining and the mineable material, classified into reefs labelled A, B, C, D, E, F, G and H. The alternating reefs are waste and ores respectively; and in this case A, C, E, G are the ores while B, D, F, and H are the waste material. None of the waste was sampled for WI analysis but the ore reefs sampled show that they were comparatively soft and in the order G< E< A. Reef C was the hardest, they were all sampled at the same depth (69 RL). This is shown in Fig. 8 and could give an indication of mineralogical variations. The reefs were soft while the ores from the quartzite and grits above the reefs showed high work indices.

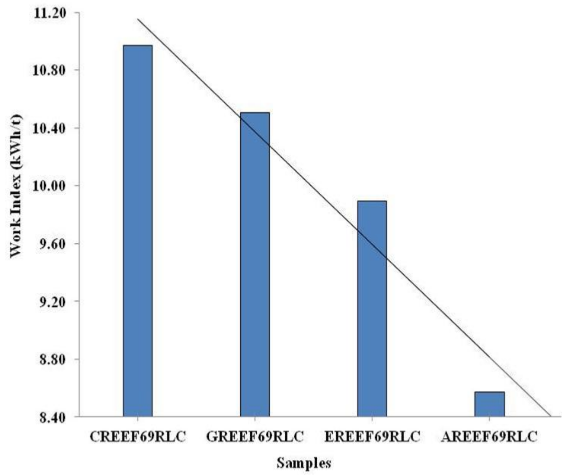

Fig. 8 Graph of WI of Various Reefs of Teberebie Pit

3.3.3 Same Depth Variations across Pits

Samples were taken from different pits at the same depths for comparison and the values obtained were plotted in Fig. 9.

All the samples were taken at 69 RL. From the graph at the same pit depth the hardest material was from Kottraverchi Pit, followed by Akontasi Pit and then Teberebie Pit. Teberebie Pit's sample was a reef sample and it is worth noting that the reef samples were generally incompetent due to its mineralogical characteristics.

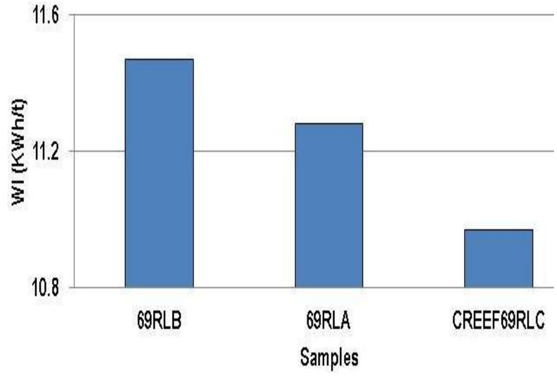

Fig. 9 Same Depth Plots of Sample WI from Different Pits

3.4 Plant Data Collation, Analysis and Computer Simulation

3.4.1 Blending

Blending is done mostly when very competent ores are encountered and these materials cause problems with power draw and wear rates. It is done to produce a suitable WI of the composite material after blending and prevent the associated problems. In this case all the work indices calculated were lower than the design mill WI and therefore it is expected that the mill should be able to handle the material well. The material was classified into hard, soft and medium to aid blending. The classifications were: soft had work indices less than 11 kWh/t, medium hardness was in the range 11 kWh/t to 12 kWh/t and hard ores had indices above 12 kWh/t. These classifications were done based on the observation that all the various groups showed similar mineralogical characteristics and were within a range of WI values.

Practically, the surface ores were assumed to be weathered and thus soft. Thus as mining progresses, the ore is expected to be more competent. However, the study shows that depth of the ore is not necessarily an indicator of hardness. Thus predicting a blending pattern would be quite challenging. Blending based on visual assessment of hardness and more in particular along grade control lines was quite suitable.

3.4.2 Power Draw Simulation

The current operating parameters for the mill are:

(i) Ball mill motor rating (14 MW);

(ii) Feed size (3.8 mm = 3800 µm);

(iii) Product size (106 µm);

(iv) Work Index (12.65 kWh/t); and

(v) Plant throughput (1500 t/hr).

Using these design values the modular simulator developed using Bond's equation shows that the installed motor power of 14 MW is not adequate for grinding as the calculated is 15.4 MW. If any of these parameters is changed; it will result in a change in the energy used. Mill energy draw has to be strictly monitored on the plant because it informs on the accuracy of comminution and equipment burden. After analysis of the plant data an interesting observation came to bear; plant power draw was consistently below expected and this was due to various variations in operating parameters. Subsequent simulations were done to vary other parameters to help achieve the right energy utilization.

The maximum, average and minimum WI values of 12.52 kWh/t, 11.39 kWh/t, and 8.59 kWh/t respectively were simulated using the energy equation developed by Bond (Wills and Atkinson, 1993). The values obtained were 15194, 13823 and 10425 kW respectively maintaining all other design parameters. The average WI would draw 13823 kW which is still 1529 kW short of the available. Other process variables were thus simulated to ascertain their potential to increase power draw. For these simulations, a positive deviation means that the simulated value was higher than the design and a negative deviation means the simulated value was lower than the design.

Design parameter for feed size was 3.8 mm. By using an average work index of 11.39 kWh/t, several iterations were run for the feed particle size but it became clear that the energy draw was not very sensitive to increases in feed particle size; variations of multiples of 100 did not change the energy draw substantially. Therefore; feed particle size was not useful in correcting or augmenting the energy draw.

The current operating product size was 106 µm but because of low average WI (11.39) the current mill power draw reports a negative variation of 1529 kW and hence simulations were used here to ascertain the corrective potential of product size. Here, product size changes in the range 53 to 212 µm were used according to the Tyler Series of screen selection. It was realized that increments (150 µm) in product particle size produced a negative 4174 kW variation which was not desirable. Reducing the product particle size (75 µm) also produced a positive variation of 1605 kW. Also, 53 µm was simulated and that resulted in a positive deviation of 7552 kW; this was too high and hence the best product particle size would be 75 µm. This means that theoretically; there would be a lower energy variation from the set point if the product particle size can be reduced to 75 µm which is lower than the current operating size of 106 µm and would have advantages for gold liberation. This can only be achieved with increase in mill residence time, wear rates, and processing cost.

Finally plant throughput was also simulated; the mill was designed to handle 1500 t/hr at a work index of 12.65 kWh/t. With the low average WI of 11.39 kWh/t there was a 1529 kW deficiency that has to be compensated for. Currently, the plant is running at 1365 t/hr and simulating this shows that there is an even higher deficiency in energy draw of 2773 kW. To compensate for the energy deficiency it is necessary to increase the plant throughput. Simulations revealed that a plant throughput of between 1665 and 1670 t/hr would produce the lowest deficiency of negative 8.62 kW and positive 37.46 kW respectively.

Since the maximum WI is already lower than the design, blending to obtain a suitable WI is not an issue and other changes may be considered. It has been found out that the energy required is not very sensitive to feed size and therefore the major variations could be product particle size and plant throughput. It will be comparatively easier to increase throughput than reduce product size. Throughput increment could also positively affect production. From the simulations, 1665 t/hr was the best augmentation.

4 Conclusions

This study sought to understand the effect of ore characteristics from three TGM mining pits on feed rate, power draw and ball mill performance. The rock types identified were conglomerate and quartzite and the main minerals were quartz, feldspars, iron oxides and sulphides. Mineralogical analysis classified the ores into hard, medium and soft. There were little differences in the mineral constituents of the hard and medium ores except for the grain sizes which were averagely smaller for medium than for hard. Soft ores had a lot of micro-cracks and the pyrites present was undergoing oxidation to hematite or magnetite. The micro-cracks confirm high porosity and gold dissolution in soft ores. From the WI results, Akontasi was the most competent followed by Kottraverchi and finally Teberebie. Mineralogical composition, micro-cracks, fragmentation parameters and ore competence could be the possible reasons for the variation in WI and energy.

The ores from Akontasi Pit showed higher work indices. The values of WI obtained were comparatively higher than that of Teberebie Pit. Ores from the same depth of different pits showed highest WI for Akontasi then Kottraverchi and finally Teberebie Pit. The highest WI (12.52 kWh/t) obtained was below the design mill work index of 12.65 kWh/t. Material sampled from reefs showed lower WI than those from the quartzite and grits at the same depths. This gives an indication of the mineralogical differences coming into play.

Current operating conditions produce insufficient mill power draw. Since the mine is not practicing methods {Mine-to-Mill concept (Kanchibotla, 2000)} that have the potential to modify work index during blasting, other factors had to be simulated to ascertain their potential to increase power draw. Simulations showed that plant throughput should be increased to about 1665 t/hr to produce the optimum energy draw augmentation. This has advantages for production.

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References

Anon. (1981), “Rock Characterization Testing and Monitoring”; *International Society of Rock Mechanics (ISRM) Suggested Methods* (Brown, E. T. ED)”, Pergamon Press, Oxford, UK, pp. 84-85.

Comeau, W. (1996), “Explosive Energy Partitioning and Fragment Size Measurement – Importance of Correct Evaluation of Fines in Blasted Rock”, *Proc. of the Fragblast-5 Workshop on Measurement of Blast Fragmentation*, Montreal,Canada,pp. 237-240.

Elorantra, J. W. (1995), “The Effect of Fragmentation on Downstream Processing Costs”, *Proceedings of Explo’95 Conference*, Brisbane, Qld, Australia, pp. 25-28.

Elorantra, J. W. (1997), “The Efficiency of Blasting versus Crushing and Grinding”, *Proceedings of the 23rd Annual Conference on Explosives and Blasting Technique*, International Society of Explosives Engineers, pp. 157-163.

Forbah Jnr, J. Y. and Amankwah, R. K. (2010), “Effects of Rock Fragmentation on Communition – A Case Study at Damang Gold Mine”, *Presented at the 1st UMaT International Mining and Mineral Conference*, Tarkwa, Ghana.

Fuerstenau, M. C., Chi, G., and Bradt, R. C. (1995), “Optimization of Energy Utilization and Production Costs in Mining and Ore Preparation”, *XIX International Mineral Processing Congress*, San Francisco, California, pp. 161-164.

Higgins, M. (1998), “JKSimBlast – Blast Simulation and Management”, *Presented at Blasting Analysis International Eighth High-Tech Seminar*, Nashville, Tennessee, pp. 1-9.

Kanchibotla, S.S. (2000), “Mine to Mill Blasting to Maximise the Profitability of Mineral Industry Operations”, *Proc. 27th ISEE Conf.*, Anaheim, USA.

Kesse, G. O. (1985), “The Mineral and Rock Resources of Ghana” A. A. Balkema, Rotterdam, Holland.

Levin, J. (1992), “Indicators of Grindability and Grinding Efficiency”, *J. S. Afr. Inst. Min. Metall.*,Vol. 92, No. 10, pp. 283-289.

Napier-Munn, T. J., Morrell, S., Morrison, R. D., Kojovic, T. (1999), "Mineral Communition Circuits – Their Operation and Optimisation", *Julius Kruttschnitt Mineral Research Centre (JKMRC), University of Queensland*, Isles Road, Indooroopilly, Queensland 4068, Australia, pp. 413.

Siddall, B., Henderson, G., and Putland, B. (1996), "Factors Influencing Sizing of SAG Mills from Drill Core Samples", *Proc. Conf. International AG and SAG Grinding Technology*, UBC, Vancouver, Canada, Vol. 2, pp. 463.

Simkus, R., and Dance A. (1998), "Tracking Hardness and Size: Measuring and Monitoring ROM Ore Properties at Highland Valley Copper", *Proc. Mine to Mill Conf.*, Brisbane, Australia.

Wills, B. A., and Atkinson, K. (1993), "Some Observations on the Fracture and Liberation of Mineral Assemblies", *Minerals Eng*., pp. 697.

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