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The business value of best practice process mineralogy

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ABSTRACT

Modern Process Mineralogy has been making significant advances in methodology and data interpretation since it was assembled in the mid-1980s as a multi-disciplined team approach to obtaining mineralogical information from drill core and plant samples so as to infer the metallurgical processing requirements of that ore. This hybrid discipline consists of teams that include geologists, mineralogists, samplers, mineral processors and often others, working together. The degree of cross-training, communication and trust dictates the potential capacity of the team and it is possible to develop technical capabilities that surpass those of conventional teams. A pivotal tool for technically efficient and plant-oriented process mineralogy is, of course, the use of modern, automated laboratory technology. In these cases, process mineralogy, though associated with some capital investment, is a valuable risk reduction tool and an operations optimization tool for any mining company, not only in terms of finances but also in terms of human and intellectual capital. However, if the teams are dysfunctional and information is not interpreted correctly due to limited experience in the team or less than best practice, or it is not implemented or used, much of the value can be lost. Process Mineralogy can then be regarded as 'time consuming and expensive'. In this paper, the business value of best practice Process Mineralogy is outlined and discussed. Case studies that include 'green fields' new design applications and 'brown fields' interventions to mature operations have been selected to demonstrate the tremendous financial value that can be achieved are presented, along with those where costly disasters could have been averted. The list is not intended to be exhaustive or complete, and the reader is referred to the extensive literature available. Examples are selected for this publication specifically to illustrate the delicate balance between generating additional business value through potentially expensive mineralogical analyses and the lost opportunities of underperforming flowsheets, unanticipated losses due to high feed variance, inadequate liberation or deleterious minerals, over-reagentised circuits, or extra costs of unnecessary or underutilised equipment.

1. Introduction

1.1. Best practice process mineralogy

'*Process mineralogy*' can be defined as the practical study of minerals associated with the processing of ores, concentrates and smelter products for the development and optimization of metallurgical flowsheets, including the waste and environmental management considerations or as (Henley, 1983; Jones, 1987; Petruk, 2000) put it more simply 'the application of mineralogy in making processes more effective' (Becker et al., 2016). This hybrid discipline consists of teams that include geologists, mineralogists, samplers, mineral processors and often others, working together. The degree of cross-training, communication and trust dictates the potential capacity of the team and where an appropriate work dynamic is fostered, in which relationships flourish as much as does the ethic of technical excellence, it is possible to develop technical capabilities that surpass those of conventional teams.

Current best practice of Process Mineralogy is the cumulative product of several teams across the world working at developing this platform by way of new equipment, associated software, methods and associated quality controls over several decades (Lotter, 2011; Bradshaw, 2014). Although modern laboratory technology in concert with powerful software offers fast and large-scale generation of data, our industry has observed a considerable deficiency in training of succession mineralogists. The reasons for this situation are manifold and need to be addressed in discussions on strategic business planning.

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In as much as the equipment has seen a great deal of advancement, the value potential of the data arising therefrom is only deliverable through a well-trained and experienced team.

It has been shown by several of these teams that a key part of the successful use of the toolbox is high-quality training, both within-discipline and intra-disciplinary. The latter takes longer, and works best through the medium of projects being executed, with group discussions mutually interpreting the data to hand. Several generations of team members balance the experience of the team well, with the older members mentoring and guiding the younger ones, at the same time as learning new skills from the latter. It is highly preferable that most of the team members have several years of operations experience before being assigned to this multi-disciplined team. The intra-disciplinary training generally takes two years to attain a core level of multi-discipline expertise, but the learning never stops. For example, the habit of reading new publications on the subject, attending conferences and having discussions with the presenting authors, and networking with other practitioners, all add considerably to the learning and skill development.

This mentoring dynamic leads the efficient interpretation of the large volumes of data that arise from the modern practice into the specific process implications. These large data sets potentially threaten the project unless they are analysed, interpreted and summarised before being presented to the end-user. Provided this process is in operation, the reports and recommendations presented to clients in operations are summarised, readable and practical for the end-user at the operation. The key skill to develop in these teams is the ability to assess a project and to define the correct and appropriate selection of tools and equipment to complete the job effectively and efficiently. Cross-checks using common sense instead of a default setting of "the computer is always right" are critical.

The reputation of the Process Mineralogy team thus builds in the mining company or commercial laboratory as a result of the interactive, synergistic and focussed approach in project work, delivering financial value. This enables the executive to continue supporting the team across the metals business cycle.

Gaudin's first liberation model of 1939 presented a penetrating analysis of the problem. His work was followed for decades by geometrical probability models, for example Bodziony (1965) who showed that the techniques of integral geometry could accommodate the problems associated with the indeterminate nature of the geometrical mineralogical structure. Mathematical liberation models were written in the 1970s and 1980s as a lead into the definition of the grinding requirements of an ore for flotation (King, 1979, 1989, for example).

The connection between mineralogy and metallurgical performance in a plant was recognised long ago (Gaudin, 1939; Petruk, 1976; Petruk and Hughson, 1977; Cabri, 1981; Petruk and Schnarr, 1981; Peyerl, 1983; Baum et al., 1989) for example) as was the need to provide diagnostic sampling techniques of a plant (Restarick, 1976) and to improve the statistical reliability of mineralogical and process measurements (Henley, 1983; Lotter, 1995, 2005).

The development of Quantitative Evaluation of Minerals by Scanning Electron Microscopy (QEM ∗ SEM) (and the second generation QEMSCAN) (Grant et al., 1976; Barbery et al., 1979; Sutherland, 1993; Gottlieb et al., 2000), and the later development of the Mineral Liberation Analyser (MLA) (Gu, 2003; Fandrich et al., 2007) as well as of the Tescan Integrated Mineral Analyser (TIMA) (Gottlieb and Thorpe, 2016) formed the breakthrough platforms into what is now known as Modern Process Mineralogy. At Falconbridge Limited, for example, this vision was taken into a project to develop the opportunity and deliver value into operations using this new integrated approach, in which an internal rate of return of 92% p.a. was shown for the investment in the laboratory equipment, sampling, and cost of plant modifications (Lotter et al., 2002). In this case, the Process Mineralogy platform was designed using geology, sampling, mineralogy and mineral processing. The later addition of applied statistics to the interpretation of flotation tests and plant scale trials further enhanced this development.

The re-tooling of mineralogical laboratories with automated instrumentation such as X-ray Diffraction (XRD) Rietveld, Fourier Transform Near Infrared (FT-NIR), Automated Mineral Analyzers and other equipment can reduce these metallurgical risks and provide high-throughput and fast-turnaround mineralogical data (Baum, 2009, 2014a, 2014b; Baum and Ausburn, 2014; Baum et al., 2014).

Geometallurgical units (Lotter et al., 2003; Fragomeni et al., 2005) can be defined as an ore type or group of ore types that possess a unique set of textural and compositional properties from which it can be predicted they will have similar metallurgical performance. Sampling of an orebody based on geometallurgical units will define metallurgical variability and allow process engineers to design more robust flowsheet options. This variability can be muted when samples from different geometallurgical units are blended and tested as one sample. Composites are created by ensuring grade and grade distributions from a specific area defining the geometallurgical unit within a resource are maintained. The method used to divide an orebody into geometallurgical units is based on a review of geological data including host rock, alteration, grain sizes, texture, structural geology, grade, sulphide mineralogy and metal ratios with focus on characteristics which are known to affect metallurgical performance (Lotter et al., 2003; McKay et al., 2007). The foregoing list is, however, not complete and also uses hardness testing and the grade/recovery curve as characterising parameters (Fragomeni et al., 2005, for example). Statistical analysis is often used to help define preliminary units. In addition, it is recommended that a variability program based on smaller samples from throughout a geometallurgical unit is completed prior to finalising the divisions between geometallurgical units. This approach will quantify the range in performance that can be expected from within a unit, and provides a cross check that the geometallurgical unit definition is robust Additionally the sampling requirements are less demanding when the orebody is sampled at the individual geomet unit level instead of as a run-of-mine mixture, when expressed as minimum sample mass (Lotter, 2010). Early predictions of likely grinding requirements of an ore using the sulphide grain size data obtained from a series of polished thin sections measured by QEMSCAN were proposed by Fragomeni et al. (2005). Earlier, equivalent work at Mount Isa Mines, Queensland, identified ranges of textures and associated grain sizes, leading to the concept of staged grinding and flotation (Bojcevski et al., 1998). Recently, an initiative to model geometallurgical units in terms of texture, predicted grind size and liberation behaviour from drill core using scanning electron microscopy was reported by Bonnici et al. (2009). Recently, this practice was advanced to a position whereby geometallurgical units may be populated with estimated recovery values of paymetals (Evans, 2010).

The synergy between sampling, mineralogy and mineral processing in modern process mineralogy is shown in Fig. 1. Starting from representative sample material (Gy, 1979), the mineralogical characterization of the sample material develops powerful information as to the type, size and quantity of minerals present. From this information, metallurgical processing implications are developed and communicated to the mineral processing team, who work on flowsheet development strategies. This cuts down on the mineral processing resource and schedule considerably compared to the older conventional mineral processing approach.

The foundation of good chemical, mineralogical and metallurgical data is a statistically sound, robust sampling approach. Carrasco et al. (2004) and Lotter and Laplante (2007a, 2007b) have documented these issues. As illustrated by Carrasco et al. (2004), inadequate sampling in a copper operation had resulted in hidden losses of a considerable magnitude over a 20-year period, i.e. probably more than US $ 2 billion. Laboratory automation – from sample preparation through chemical and mineralogical labs – is a pivotal addition to good sampling as it minimizes sample preparation errors and provides the better data platform for continuous process adjustments (Best et al., 2007).

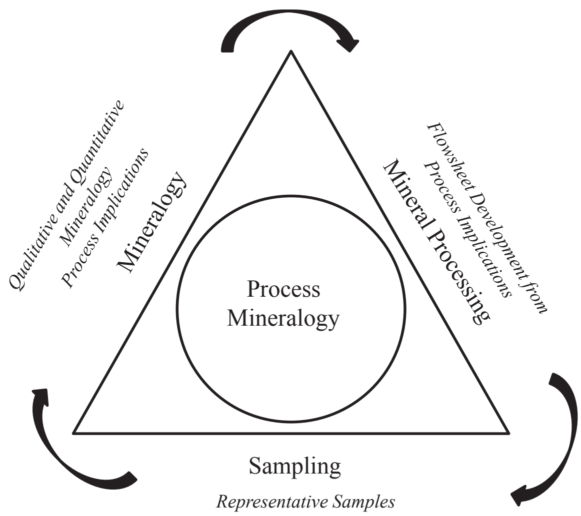

Fig. 1. Synergistic Interaction Between Sampling, Mineralogy and Mineral Processing in Process Mineralogy (after Lotter et al., 2002).

Use of this arrangement at a best practice level does cost more than the conventional approach, however when correctly performed, and when the mineral processing recommendations are used, significant value is delivered for the project in terms of cash flow, as shown schematically in Fig. 2 for a Greenfield project, and in Fig. 3 for a Brownfield project or retrofit to a mature operation.

Fig. 2 shows that the early costs at the beginning of the project are greater for the best practice process mineralogy case than for the conventional mineral processing case, but that the revenues due to improved process performance after commissioning are much greater. If the improved process performance is not obtained, then no value from the increased expenditure is delivered and the added mineralogical analyses are a wasteful expense.

Fig. 3 shows at the time of the retrofit, additional value can be created with simultaneous mineralogical characterization and process diagnosis so that the circuit performance can be optimised.

The purpose of this paper is to demonstrate the business value of best practice Process Mineralogy in selected case studies. These demonstrate that, although the cost of this best practice is higher than older conventional mineral processing, the value delivery in return for that investment is very significant. Numerous examples of equivalent development and applications have since been published, for example Lotter et al. (2003), Baum et al. (2004), Fragomeni et al. (2005), Charland et al. (2006), Dai et al. (2008), McKay et al. (2007), Triffett et al. (2008), MacDonald et al. (2011); Rule Schouwstra (2011), and Gu et al. (2014). The installation of the first fully integrated mine-site Process Mineralogy Laboratory for production at Cerro Verde in 2005 (Fennel et al., 2005) and the subsequent start-up of the first automated X-ray Diffraction Near Infrared (XRD-NIR) Mineralogy Laboratory (AXN Lab at Freeport in Arizona operating in concert with a large Central Analytical Center) represent major milestones for daily, quantitative mine- and plant-related mineralogy support (Baum, 2009).

Although specific value examples will be provided hereinafter, one should not underestimate the extreme value addition/cost savings of continuous process mineralogy in a *"Fire-Fighting-Emergency-Room-Mode"*. For a large copper mining company, this can amount to US $ 11–20 million of benefits per year. It is obvious, that these value generations alone cover more than operating cost and capital expenditures for modern mineralogy laboratories.

2. Case studies

2.1. Greenfield projects

The startup of a new concentrator, a much anticipated event, is a critical stage in the project. The achievement of designed throughput rate as tonnes per day milled as well as the designed final concentrate grade and recovery have a major influence on the return on investment (Mackey and Nesset, 2003).

The monetary values lost through plant ramp-up delays and so-called "de-bottlenecking" are large and can range from < US $ 100 to > $ 500 million for single operations. However, a substantial increase of business value is achieved if the ramp up can be accelerated. According to Meadows (2014) and Meadows and Baum (2016), a combination of better ore characterization, tailored flow sheet design, good sampling and robust metallurgical testing (without short cuts) are the keys to reducing slow ramp-up. In a 14.6 mt/y copper plant, this could equate to +/- US $ 163 million more revenue and in increased NPV of roughly $ 118 million – which is equivalent to the total

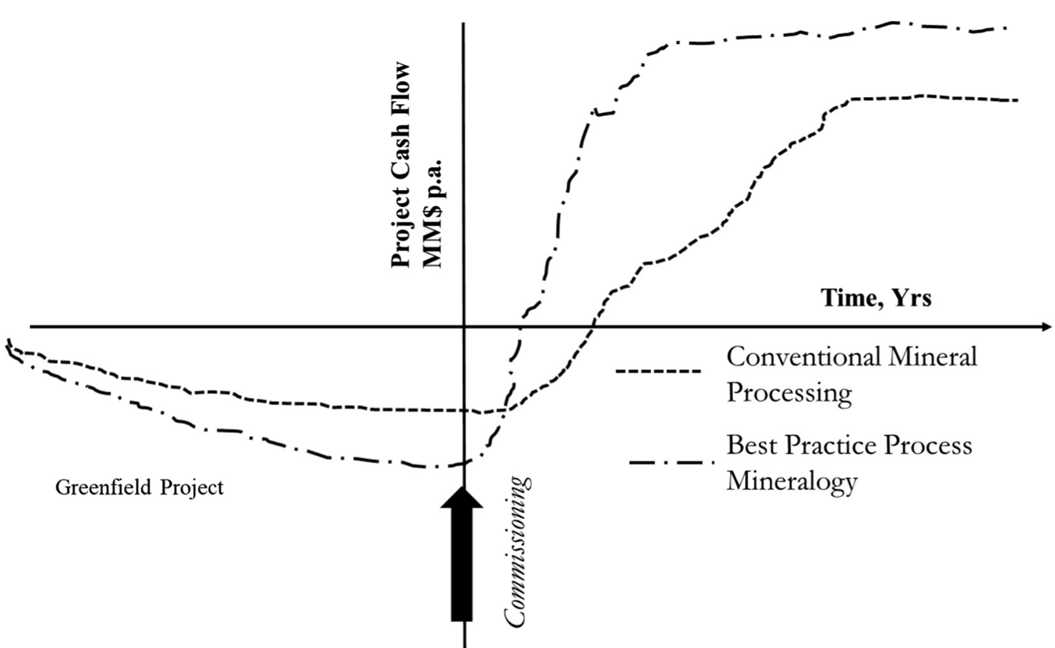

Fig. 2. Comparison of Project Cash Flows Before and After Project Commissioning using Conventional Mineral Processing and Best Practice Process Mineralogy in a Greenfield Project.

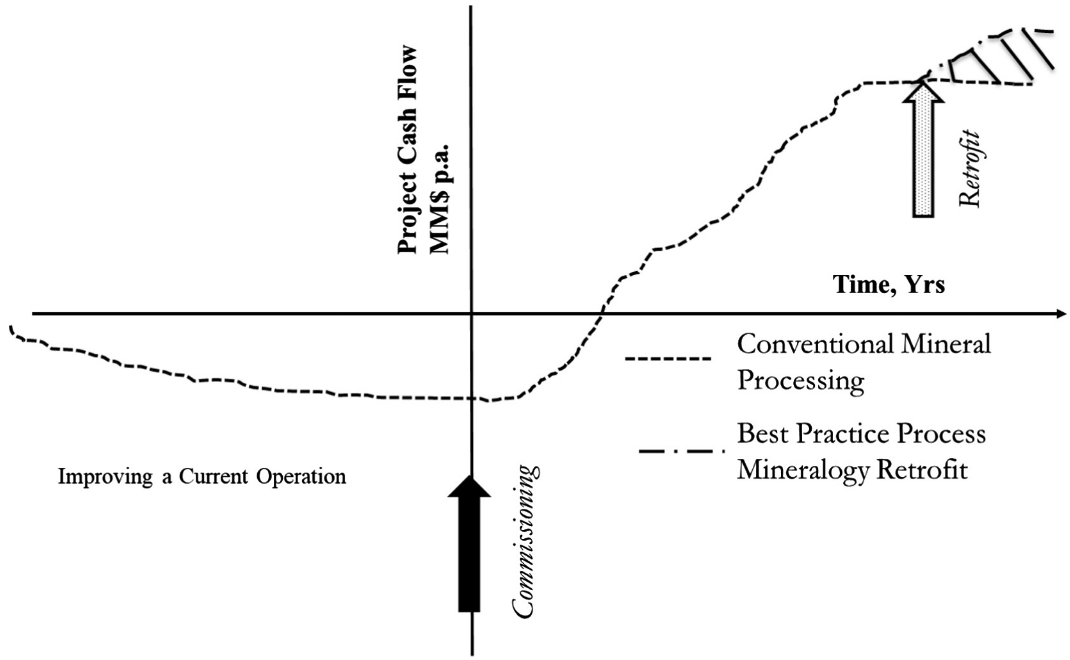

Fig. 3. Effect on Cash Flow of a Mature Plant Operation with Retrofit by Best Practice Process MineralogyCharacterisation.

equipment cost.

2.1.1. Comminution circuit design

Comminution circuits represent a substantial portion of capital investment necessary to commission a new mining project, with some budgets allocating 35–50% of the plant capital expenditure to comminution and its required ancillary services (Lane et al., 2002). Furthermore, grinding is also a major driver of operating performance due to its influence on mineral liberation and resulting metallurgical recovery as well as high consumption of consumables, energy and in some cases, water. There is an increasing realisation that mining and mineral processing should be considered as linked activities, rather than as separate and unrelated activities. As an example of the new trend, the increasing dominance of autogenous and semi-autogenous grinding circuits in new operations is helping to focus attention on the linkage between fragmentation in mining and grinding circuit performance. The general mining rule is that coarse fragmentation is preferred within the constraint of limiting the amount of boulder size muck. Comminution circuits using fine crushing and rod and/or ball milling are largely insensitive to the size of muck produced in mining. However, Autogenous (AG)/S emi-Autogenous (SAG) grinding circuits are quite sensitive to the mix of fine and coarse material in the mill feed (McKee et al., 1995). The purpose of the circuit is to treat the crushed ore at a desired treatment rate (called the feed rate), producing a finished product at the same rate as the feed rate with a maximum of that finished product reporting to the ideal product size. It then stands to reason that best practice mineralogy should consider grinding circuit design. For the purpose of this paper grinding circuit design will be defined as follows:

The selection of size reduction equipment (grinding mills etc.) or processes (drill and blast) which can profitably achieve:

a target throughput rate of ore when producing a given product specification,
with a known confidence interval on its ability to meet that target.

A number of authors have proposed different methodologies to select, scope, size and design appropriate comminution equipment, for example Bond (1961), Barratt and Doll (2008), Barratt (1989), Morrell (2008), Mular (2002), Powell and Morrison (2007), Rowland (1985), and Starkey et al. (2015).

Many of these methodologies are proprietary or not necessarily compatible with the desired final circuit flowsheet. This often leads to particular unit steps of comminution with a reputation for complexity (such as controlled blasting, AG/SAG milling, HPGR and ultrafine grinding) being managed by independent experts at the design stage, despite the potential to achieve a superior grinding circuit design through a more collaborative approach.

While overall agreement has not yet been achieved in the industry, there are some larger themes which should be agreed upon. First is the need for the characterization of the ore by unit at a preliminary design stage. This unit division generally involves the compositing of samples which represent either known metallurgical units with the geological distribution of the ore body or known chronological division of proposed feed from the mine. The characterization of metallurgical units is the common approach when the deposit is well understood from a geological perspective, while chronological units are often useful when the mining method has been predefined and sequencing is roughly understood as is sometimes the case with a simple open pit mine.

In each method of comminution characterization, further testing is required to understand the variability of the measured hardness within each ore zone. This not only provides a distribution around the measured composite average but also provides a resolution of data when it comes time for the creation of geometallurgical models of grinding design parameters and/or throughput predictions. It is this portion of the testing exercise which satisfies the second criteria described above, that is to produce a design with a known confidence interval or probability of successful implementation.

For complex ore bodies (which are becoming more and more common), a division by metallurgical unit is the superior choice due to the flexibility it imparts to later analysis; particularly when the grinding throughput variability within each metallurgical unit is also well understood. The selection of samples on the basis of metallurgical units is largely independent of changes to the recovery flowsheet, most notably to changes in the throughput rate and mine plan which often occur well after the testwork has been completed. This allows the designers a great deal of flexibility to adapt to changes in the project without compromising their source data set. In contrast, when ore bodies are sampled on a chronological basis a change to the mine plan becomes a significant challenge to the interpretation of the raw testwork data and may require additional sampling at the new boundaries of the

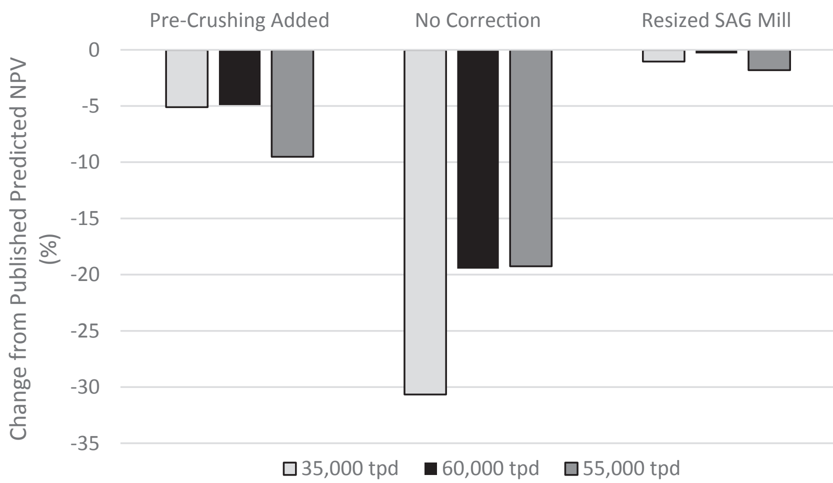

Fig. 4. Change in Expected NPV for Corrections to Undersized SAG Circuit.

chronological units in order to define the variability within the new subsets.

This flexibility has clear and demonstrable value as calculated by Reeves et al. (2015) shown in Fig. 4. This paper showed in a review of three copper/gold mine case studies that the selection of one unit step alone, the AG/SAG mill, has a significant impact on project valuation. It calculated that the selection of an AG/SAG mill that is one size (2 ft. diameter) smaller than necessary to meet the design criteria created a process bottleneck resulting in an average NPV loss of 5% relative to the value expected by investors as presented in NI43-101 studies even after correction by the addition of a pre-crushing plant. Had the design revised prior to construction that loss is approximately 1% of NPV.

While there were other factors which influenced the selection of the undersized grinding equipment in those three case studies, it is clear that there is value to the flexible interpretation of grinding testwork provided by metallurgical units which would allow for a correction of the mill sizing exercise even at a relatively late stage of the project development.

The primary risk that exists in the definition of metallurgical units is the potential for recursive revision of those units. In measuring the process characteristics of a given unit, the boundaries of that ore zone may need to change. This risk is accentuated for grinding measurements which are usually not included in the testwork program until relatively late in the project development after these units have been defined. As such, some consideration should be paid to the measurement of variability within each ore zone and at boundary intersections. For this reason (and others described elsewhere), the number of units should be kept to a minimum to reduce the number of boundary interfaces between units.

2.2. The Montcalm project

The Montcalm base metal project, located near Timmins, North Ontario, was successfully commissioned in 2004. In this case, the flowsheet had already been designed by conventional means at the time Falconbridge purchased the resource, and due to the limited seven-year life of mine, had to be fast-tracked to commission into an anticipated upswing in the nickel market. Thus the testwork required an accurate prediction of metallurgical performance by the frozen flowsheet from samples of drill-core (Fragomeni et al., 2009).

The Montcalm flowsheet consists of rod mill and ball mill grinding to a target of p80 of 39 µm. Run-of-mine mill feed is prepared by on-site crushing to −19 mm to minimize crushing plant contamination with the Kidd Creek Cu/Zn ore. Rougher flotation feed is produced by rod and ball milling and is subjected to bulk Cu/Ni roughing, followed by two stages of bulk cleaning. Bulk concentrate is subjected to Cu/Ni separation using conventional column cells. Xanthate is the collector and MIBC is used as frother. Depramin C, a CMC, is added to depress ferromagnesian silicates in the flotation process (Lotter and Fragomeni, 2010).

The orebody is hosted in a norite and gabbro intrusive complex with minor peridotite lenses and mafic and granodiorite dykes. Shear zones and faults are locally encountered and host chloritic alteration products, including talc. Sampling by drill-core ahead of commissioning the project identified three end members. The Montcalm ore reserves occur as three distinct lenses, referred to as the East, West and Deep Zones.

The mineralogical assemblage is locally variable, with changing ratios of the main sulphides: pyrrhotite, pyrite, pentlandite and chalcopyrite. The silicate gangue is primarily composed of plagioclase and amphibole, exhibiting variable degrees of sericitisation or chloritisation. Calcite, magnetite, zoisite (a Ca-Al silicate), quartz, biotite and talc occur as minor accessory gangue minerals.

Following this structure, the core was sampled separately and taken through the crushing and blending methodology described earlier. Care was taken to ensure that the full variability of the orebody was captured in the drill-core sampling for the flotation testwork (Charland et al., 2006).

The Montcalm end member ore characterization conducted prior to flotation testing included:

1. performing a whole rock thin section investigation of each ore end member, and
2. size-by-size liberation evaluation of each end member from laboratory-scale grinds that simulated the production grind.

Geological review of the Montcalm deposit resulted in the definition of three ore end members or geomet units, distinguished by sulphide texture and grade (Kormos and Whittaker, 2002). These are disseminated ores, net-textured ores and massive sulphides. A limited number of drill core samples were measured by QEMSCAN. These initial analyses define modal mineralogy of the end members and thus provide the processing team with an understanding of which mineral species are expected during production of the ore body.

This approach also can give initial warnings of critical textures (e.g., grain sizes, possible liberation issues) and problematic mineralogy (such as, in the case of Montcalm, the presence of significant amounts of pyrite). There were a few significant observations reported in the thin section study. The first was the recognition that pyrite is present in significant proportions but also that its presence was quite variable throughout the ore body (Kormos and Whittaker, 2002). During initial laboratory scale flotation tests, some tests showed that the ratio of pyrite recovery to concentrate was high, resulting in an unacceptably low concentrate grade. The mineralogy study focused on defining pyrite content in the ore body so variation in feed to the plant could be better understood. It also gave the metallurgical team a preliminary indication that the pH modifier proposed for the plant (soda ash) may not be adequate for pyrite rejection.

Despite the limited number of thin sections measured, a second important observation was made relating to grain sizes of the sulphides. Pentlandite occurs as coarse grains and also as very small flames locked within pyrrhotite – or a bimodal distribution. Chalcopyrite is on average much finer than the pentlandite and is associated most often with silicate gangue. It was also noted that the average grain size for pentlandite was significantly larger than the proposed target grind of 39 µm. It was suggested that a coarser primary grind and implementation of a regrind later in the circuit would be a better alternative for treating this ore body.

High-confidence flotation testing of a life-of-mine ore composite taken from drill-core produced an estimate of bulk concentrate recovery and grade. This estimate proved to be a recovery of 82.9% Ni at a bulk concentrate grade of 9.0% Ni (Arrué et al., 2007). On commissioning, the plant demonstrated a nickel recovery of 84.0% with a concentrate grade of 9.93% Ni, in good agreement with the laboratory scale estimate. It is also worth noting that this result was delivered within three months of startup as a Type 1 or better in the McNulty commissioning models, as shown in Fig. 5.

3. Brownfield projects

A considerable application potential for robust Process Mineralogy remains untapped as plant surveys too often are used sporadically and/or for circuit troubleshooting only (Baum et al., 2013b). As multiple plant surveys in copper operations have shown (Meadows et al., 2013, 2014,), the net project cash flow can be increased by 50–200% if a combination of robust ore characterization and plant process mineralogy is performed.

According to Meadows et al. (2013), considering the case of a typical copper concentrator with 100,000 metric tonnes per day, assuming a 0.5% Cu head grade and a US $ 3.3/lb Cu price, every 0.1% recovery difference corresponds to US $ 1.21 million per year. In practice, it is not uncommon that the plant recovery can be as much as 5% lower than the designed value due to poor process design and insufficient testing *(in both cases detrimental mineralogy features were missed)*. This corresponds to a loss of +/-US $ 60 million per year.

One of the early successes of plant process mineralogy was the El Indio gold-silver-copper operation in Chile. At this operation, a 3-year continuous Process Mineralogy program (1984–1987) in concert with metallurgical optimization achieved significant recovery improvements (Baum et al., 1989). It resulted in a 10% gold recovery increase and minor silver and copper recovery improvements as well as a better arsenic trioxide product from the roaster (Fig. 6).

As illustrated by Kendrick et al. (2003), a detailed concentrator survey with robust sampling of all ore types and plant streams, subsequent quantitative mineralogy and rougher kinetics float testing achieved significant metallurgical and economic improvements (Figs. 7a and 7b).

3.1. The Amandelbult project

In the South African platinum industry, the robust process mineralogy platform developed by Rustenburg Platinum Mines and Anglo American since the 1980s led to a thorough understanding of the platinum group element (PGE) mineral hosts in their Merensky, UG2 and Platreef ore types, and led to a list of processing implications that were practically turned into sustainable performance improvements as retrofits to the standard plant layouts (Kinloch, 1982; Peyerl, 1983). Whereas the first advances in metallurgical performance were achieved in the mid-1980s with main stream regrinding of rougher tailings and the addition of a scavenger flotation circuit (Lotter, 1995), the next generation of improvements was brought to hand by the installation of niche regrind Isamills treating the rougher and scavenger concentrates (Rule and Schouwstra, 2011). In this case, the UG2 plant at the Amandelbult mine, Limpopo Province, was identified for a flowsheet retrofit based on detailed mineralogical study of its key flowsheet streams. These studies consistently showed that, apart from the expected ultrafine losses of PGM to the tailings, a significant amount of incompletely liberated PGM was seen in the silicate mineral phases. The plant was retrofitted with Isamills to regrind the classified mainstream rougher tailings silicates, and to regrind the medium grade cleaner circuit feed, as shown in Fig. 8 (Rule and Schouwstra, 2011). The UG2 flowsheet employed in both Amandelbult plants is the typical Anglo Platinum split regrind, MF2 UG2 circuit but now incorporating stirred milling. The primary circuit performs the function of liberating the silicate minerals from the chromite spinel particles in the chromitite matrix of the ROM UG2. The PGMs largely are contained within the silicate matrix between the chromite spinel grains in the ore. The liberated material or partially liberated material – typically PGMs or PGM–base metal sulphides – is recovered in the primary rougher flotation. PGMs are small sized – typically with an average grain size of

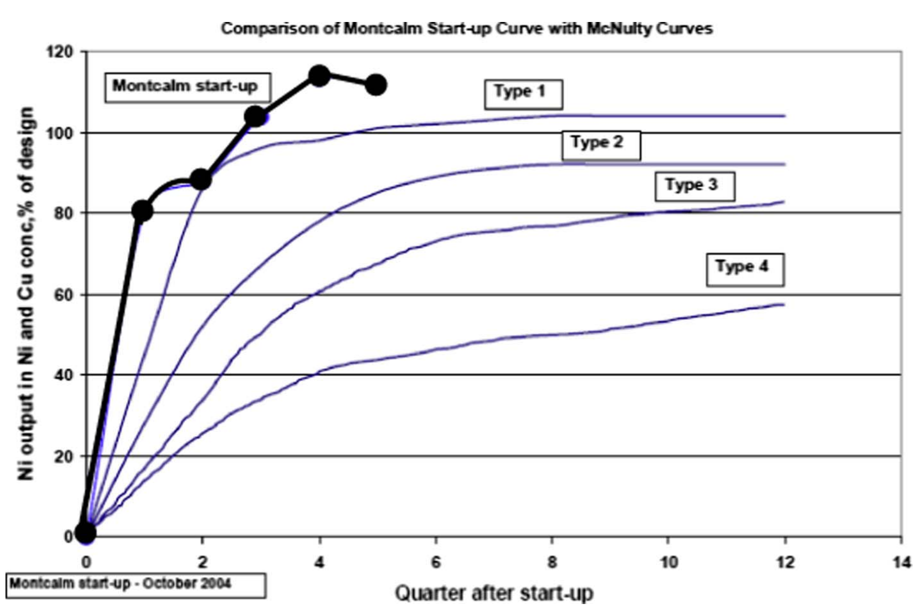

Fig. 5. Plot of the Montcalm Startup Curve Against the Four McNulty Startup Types (after Fragomeni et al., 2009).

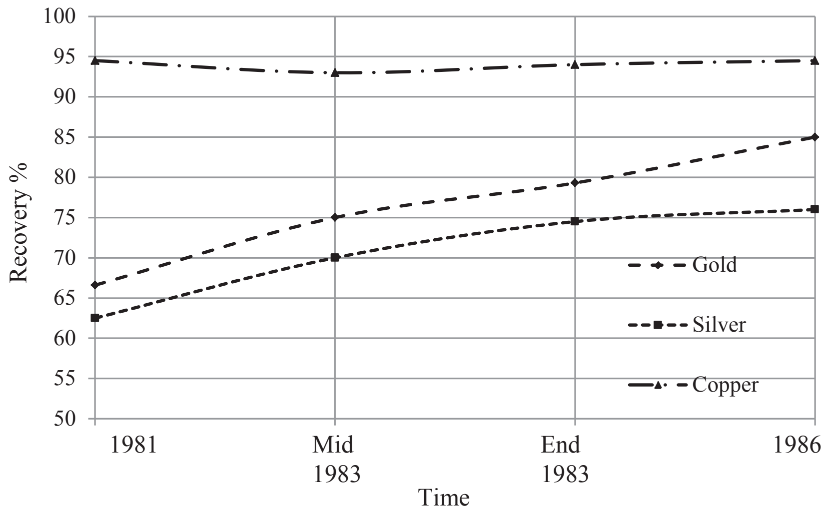

Fig. 6. Plant improvement at the El Indio Operation after 3 Years of Process Mineralogy (after Baum et al., 1989).

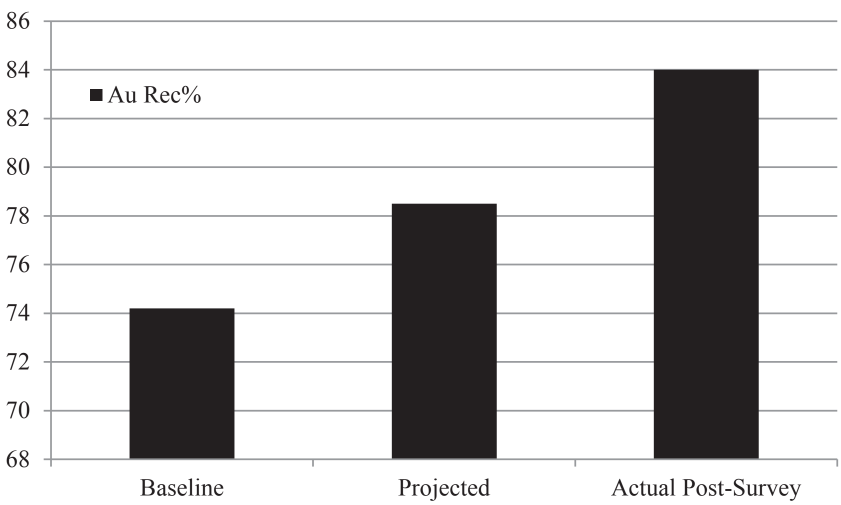

Fig. 7a. Gold recovery improvements after Mineralogy/Metallurgy Survey (modified after Kendrick et al., 2003).

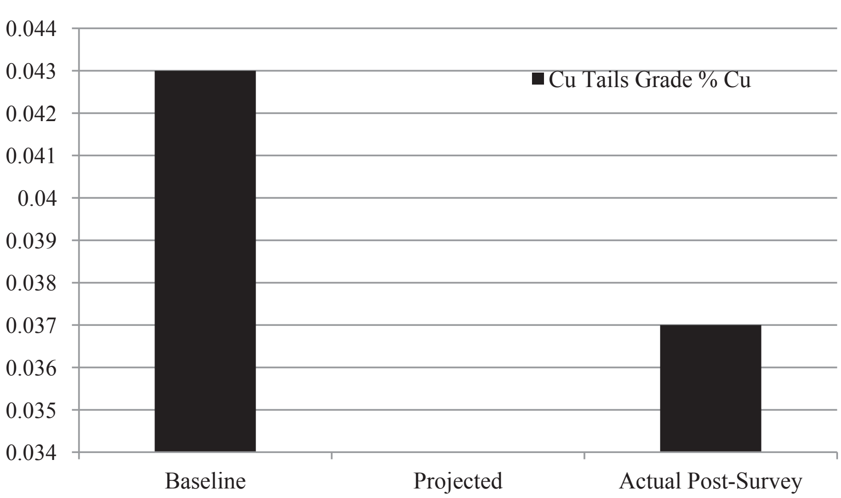

Fig. 7b. Reduction of copper losses in tails after Mineralogy/Metallurgy Survey (modified after Kendrick et al., 2003).

less than 10 µm. The primary flotation step recovers the majority of the liberated and partially liberated PGMs or PGM–base metal sulphide composites; the following split regrind stage raises total PGM extraction to almost 90%. The mainstream is split using hydrocyclones, taking advantage of density difference between silicates and chromite spinel, producing an underflow stream with coarser particles enriched in chromite spinel; the cyclone overflow is enriched in silicates, some importantly with PGMs. This stream is then sent for fine grinding through the Main Stream Inert Grinding (MIG) circuit after pre-treatment by closed circuit ball milling. The coarser chromite enriched stream is treated in an open circuit ball mill. The products are treated in separate flotation circuits. Amandelbult is the second largest production site for Anglo Platinum, annually producing roughly 450,000–650,000 oz of platinum and 820–1150 oz of total PGMs (production data from the last 5 years). The complex consists of three individual plants: the original Merensky plant, with a capacity of 3.75

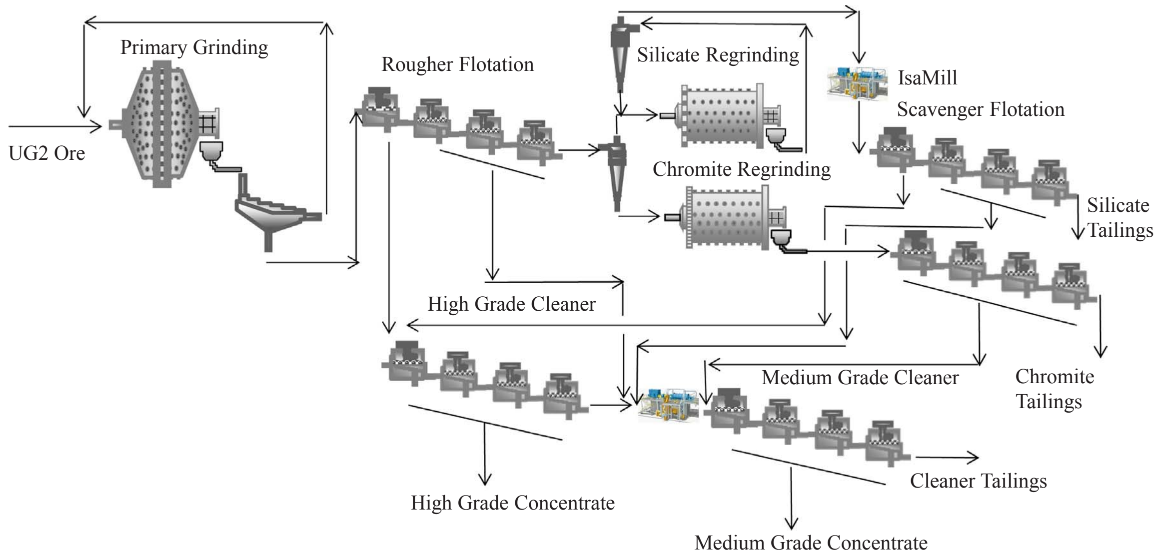

Fig. 8. Flowsheet of the Modified Amandelbult UG2 Flowsheet, Showing Location of Two IsaMills.

million tonnes per year; UG2 #1, commissioned in 2000, with a capacity of 2.5 million tonnes per year and UG2 #2. The second UG2 plant was re-commissioned at an expanded capacity of 2.5 million tonnes per year in 2009. As is the general industry trend, the importance of UG2 has grown remarkably in the last decade and currently makes up more than two-thirds of the ore tonnes processed.

The data shown in Table 1 indicate the typical Amandelbult UG2 operating data over a period before and just after the installation of the stirred mills in late 2009 (Figure 7). The tails values were historically between 0.8 and 1.0 gpt PGM, 4E, that is (Pt, Pd, Rh + Au). During 2010, the trend has been downwards and values of as low as 0.5 to 0.6 gpt have been achieved regularly in the second half of the year, illustrating the increasing impact of the optimised stirred milling flowsheets. Inspection of the table of operations data shows that the state of liberation of the base metal sulphides improved from 56–60% to 69–72% as a result of the Isamills.

This reduction in grade of PGE in final tailings added significant business value to the Amandelbult operations, adding very roughly 30,000 oz p.a. to the platinum production from the same ore tonnage treated as before.

3.2. The Lac des Isles project

The engagement of the Process Mineralogy toolbox with existing concentrators requiring performance improvements was well-demonstrated by Martin et al. (2003), in the case of the Lac des Iles expansion project in Ontario, Canada. The operation was expanded from a 2400 tonnes per day (tpd) operation to a much larger 15,000 tpd business. This required a new concentrator, which was designed from a pre-feasibility study. One major difference between the two flowsheets was the 80% passing size (d80) size of the float feed, presumably recognising the need for a finer grind to liberate the discrete PGM. The change in d80 size was from 150 to 75 µm. Additionally the flotation residence time was increased from 19 to 55 min. The collector suite used was a mixture of Potassium Amyl Xanthate (PAX) and di-isobutyl dithiophosphate. A new heavy (750 g per tonne (g/t) milled) dose of talc depressant as Carboxy-Methyl Cellulose (CMC) was used in the rougher float. This was another change in the practice. Methyl Isobutyl Carbinol (MIBC) frother completed the reagent suite. Primary concentrates were reground in vertimills to a d80 size of 20 µm before cleaning in two separate cleaner circuits. Shortly after commissioning in October 2001, it became apparent that, whereas the concentrate grade was almost in agreement with the design value of 170 g/t Pd, the recovery of Pd was short of design. Actual Pd recoveries amounted to 67.5%, as compared to the design requirement of 82%.

Several plant surveys ensued, supported by mineralogy as well as size-by-size paymetal analysis of streams, each delivering clues to flowsheet improvement. The survey methodology was not described; however the mineralogy was performed by QEMSCAN. These were implemented across a schedule and progressively advanced the grade and recovery of the saleable concentrate. The heavy talc depressant dose in the rougher float was lightened so as to allow some talc to float. This stabilised the froth. Additional cleaner capacity was found by re-commissioning the cleaner circuit from the older, smaller concentrator. A key discovery was the bimodal size distribution of the palladium mineral host grain sizes. The two modes of this distribution are approximately 20 and 5 µm.

It was found that mostly, it was the coarser size distribution of discrete Platinum Group Minerals (PGM), as kotulskite and palladoarsenide, that were being recovered. The appropriate regrinding of all primary concentrates in the vertimills had a major effect on both concentrate grade and recovery. The regrind product size was a d80 of 20 µm. The two cleaner circuits were simplified to a single circuit. All of these changes improved the saleable concentrate to a grade of 240 g/t Pd at a recovery of 74%, delivering a recovery gain of 6.5% Pd.

Table 1
Amandelbult UG2 Performance Data.

Item	Feb 08	Dec 08	Feb 09	Dec 09	Feb 10
Grade g/t 4E	0.9	1.0	1.0	0.9	0.7
Tailings d50 µm	50	63	47	38	57
Alt. silicate mass%	2.4	2.5	3.3	3.3	2.1
BMS Lib%	56	60	59	72	69
PGM Lib%	24	27	33	33	24

3.3. The Raglan project

The Raglan Ni-Cu-PGE deposit is located in northern Quebec on the Ungava Peninsula. The deposit is hosted by an alternating succession of thick komatiitic peridotite flows and sills of Archean age (Lesher, 1999) that are part of the Cape Smith Belt. The deposit has also been pervasively serpentinized and was then metamorphosed to regional greenschist facies (St. Onge and Lucas, 1986; Barnes and Barnes, 1990). Mineralization occurs in a series of lenses that grade stratigraphically from massive sulphides at the base, upward into net-textured and, finally, disseminated sulphides. Post-serpentinization metamorphism has created complex replacement textures where sulphides have replaced silicates to define reverse net-textured sulphides (Dillon-Lietch et al., 1986). The result is a metallurgically challenging texture from which to separate ore sulphides. At Raglan, three main end-members or geometallurgical units have been defined, each of which have different mineralogy and mineral processing characteristics (Fragomeni et al., 2005). These are: massive sulphides, net-textured sulphides, and disseminated sulphides.

The Raglan operation was commissioned into production in January 1998. The initial measured treatment capacity was approximately 108 tonnes per operating hour or 850,000 tpa, treating ore at a grade of 2.98% Ni. The commissioned grind at rougher float feed level was equivalent to a d80 size of 68 µm from a SAG/ball mill circuit with in-circuit crushing. Potassium amyl xanthate (PAX) was the standard xanthate used since operations were commissioned. The standard PAX dosage rate in the flotation circuit was 300 g/t of ore milled. A bulk concentrate for shipment to Sudbury was produced at a grade of 16% Ni and at a recovery of approximately 86.8%. These results closely matched their design equivalents, which were 100 tph milled with 87% nickel recovery at a 16% nickel grade in concentrate. From commissioning, several projects were identified and successfully implemented so as to increase capacity to 1 million tpa, and to improve metallurgical performance (Fragomeni et al., 2005; Langlois and Holmes, 2001; Lotter et al., 2002). A practice of surveying this circuit to benchmark the progress in the operation was implemented (Lotter et al., 2016).

Given the close agreement between designed and commissioned performance, at first glance there would not seem to be any motive to pursue an operations improvement program. But the mill had been designed using conventional mineral processing, and not by modern process mineralogy. Therefore there was no measure of performance entitlement against which the commissioned results could be benchmarked. A first survey of the commissioned flowsheet was performed in June 1998 so as to capture a representative sample suite of the key flowsheet streams for a closed mass and value balance, and thereafter for detailed mineralogical characterization by size class (Lotter et al., 2002). The commissioned flowsheet is shown in Fig. 9.

From the closed mass and value balance, it was clear that the initial rougher flotation recovery, at 92.8% Ni, was higher than the final recovery at saleable concentrate, which was 86.5% Ni. This implied that a total of 6.3% Ni recovery was being lost by the cleaner circuit in scavenger tailings.

The following liberation conventions are used: liberated particles are particles of any size which consist of more than 90% by area of the mineral of interest; middling particles are particles of any size which contain between 30% and 90% by area of the mineral of interest, and locked particles are those which contain less than 30% by area of the mineral of interest.

The mineralogical measurement and interpretation of the survey data soon showed that a dominant signature of textures in the flotation circuit was influencing the metallurgical performance. This was associated with a pattern of poor liberation in size classes coarser than 25 µm. Only 78.8% of the pentlandite in the rougher float feed was liberated, with 9.6% as middling particles and 11.6% as locked particles. The rougher recovery of pentlandite was 96.7%, including a class of middling particles – described as bladed and disseminated textures – which as an individual class displayed a rougher recovery of 47.5%. The grain sizes of pentlandite in rougher float feed are shown by size class in Table 2.

Examination of the rougher tailings showed that the NiFe sulphides present were distributed as 8.0% liberated, 20.0% middling, and 72.0% locked particles.

A large circulating load was found in the scavenger concentrate that recycles to the column cleaner circuit. The mass and value balance reported a solids circulating load of 394% (basis: rougher concentrate mass arisings = 100%) in the scavenger concentrate. This circulating load was associated with a dominance of middling particles with fine-grained texture reporting to the discarded scavenger tailings. The state of liberation of NiFe sulphides in the scavenger circuit is shown in Table 3.

Another notable feature in the data was that the two column tailings, i.e. both the primary column tailings and the recleaner column tailings, were joined as one stream to feed the scavenger flotation bank. Closer analysis showed that the liberation levels of pentlandite were different in these two streams, and that the recleaner tailing contained significant amounts of liberated pentlandite at fine sizes. These features are summarised in Table 4.

More detailed analysis of the primary column tailings showed that,

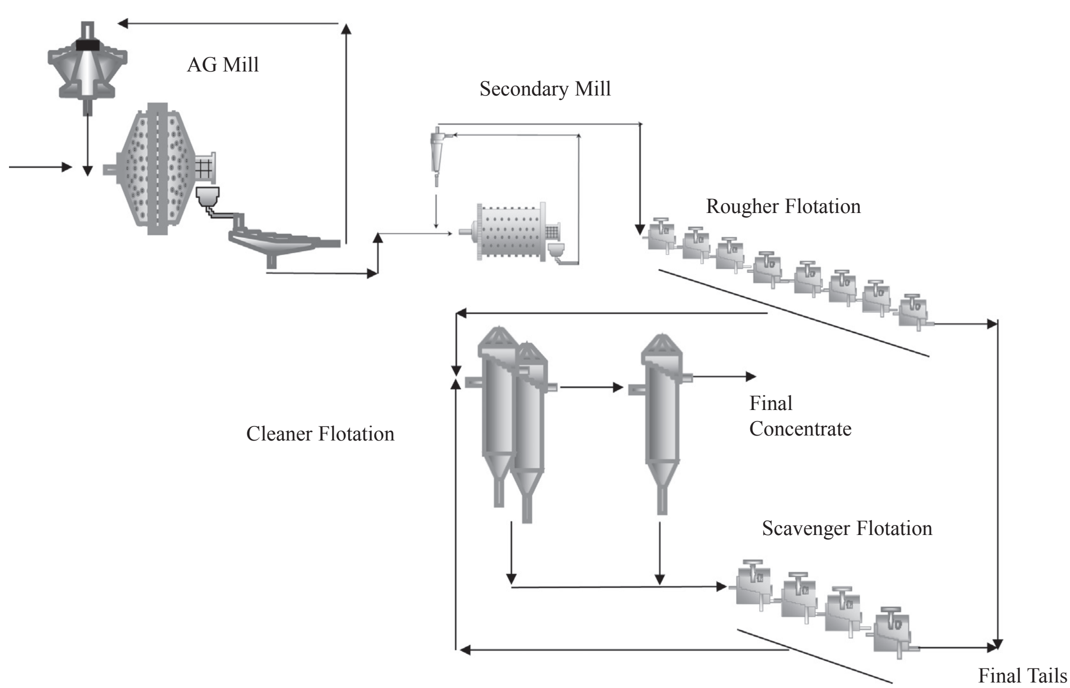

Fig. 9. Commissioned Flowsheet for Raglan, June 1998.

Table 2
Grain Size Means for NiFe Sulphides in Rougher Float Feed: Raglan 1998 Survey.

Particle Size Class	+106 µm	−106 + 53 µm	−53 + 25 µm	−25 + 15 µm	−15 + 7 µm	−7 + 3 µm
NiFe Sulphide Grain Size µm	14	28	21	13	8	4

Table 3
State of Liberation of NiFe Sulphides in the Scavenger Circuit: Raglan 1998 Survey.

State of Liberation	Degree of NiFe Sulphide Liberation%	
	Scavenger Concentrate	Scavenger Tailings
Liberated	30	26
Middling	35	29
Locked	35	45

Table 4
State of Liberation of NiFe Sulphides in the Column Cleaner Circuit: Raglan 1998 Survey.

State of Liberation	Degree of NiFe Sulphide Liberation%	
	Column Tailings	Recleaner Column Tailings
Liberated	18	72
Middling	37	14
Locked	45	14

for the locked and middling classes of NiFe sulphides, most were to be found in the coarser size fractions between 25 and 106 µm. These locked and middling particles carried NiFe sulphides at grain sizes between 11 and 21 µm – ideal feed material for a regrind mill. This explained the high circulating load in the scavenger concentrate: because of the incomplete liberation, the middling particles would float in the scavenger bank, be presented to the columns, rejected to the column tailings, and so on. Ultimately, a portion of these locked and middling particles would report to scavenger tailings.

Gangue mineralogy in the Raglan float feed is dominated by Mg silicates including serpentine, pyroxene, chlorite and trace levels of talc. Other supporting testwork, done in 1998 with high-confidence flotation testing, showed that the introduction of a carboxymethylcellulose depressant, Depramin C, at a dose of 400 g/t milled would assist in controlling the flotation of the Mg bearing minerals and would limit their interference with the sulphide flotation (Lotter and Fragomeni, 2010).

Several recommendations for improving flow sheet performance were made as a result of this survey and the associated flotation testwork. These were: re-routing of the recleaner column tailings to the head of the primary cleaner columns; installing a bypass concentrate at the first rougher flotation cell, with adjustment of level control, to release a fast-floating increment of liberated sulphides to final concentrate; regrinding of the primary column tailing prior to presenting this stream to the scavenger flotation bank; and the introduction of a gangue depressant in the rougher float, to control the Mg silicate flotation.

The implementation of these changes took place between 1998 and 2000 with the simpler changes being made first. The final modified flowsheet is shown in Fig. 10. The cumulative performance gains from these changes amounted to an increase in final concentrate grade from 16 to 18% Ni, together with recovery gains to final concentrate of 2.1% Ni, 1.5% Cu, 1.9% Pd, and 4.1% Pt (Lotter et al., 2011). These differences were measured from plant operating data. The modified flowsheet is shown in Fig. 10, and delivered a 92% p.a. rate of return for its costs (Lotter et al., 2016). In this reference, the book discusses 15 other case studies using Process Mineralogy.

3.4. Copper operations

Copper mining represents (amongst base metals mining) the largest volumes of ore and rock processing. Needless to say, ore variance and process mineralogical challenges have a corresponding large impact on the plants and their metallurgy.

There was a threefold increase of operating and capital cost for gold and copper between 2004 and 2014 (Marsden, 2016). Slower than expected ramp-up has added to the economic detriments. The result of poor ore characterization are unexpected outcomes, underperformance in the plant, cost overruns, project delays, and, frequently, recurring metallurgical problems. Marsden (2016) pointed out: "*You might be surprised at how many tonnes go through a mill and process that lose money*".

The re-tooling of mineralogical labs with automated instrumentation such as XRD Rietveld, FT-NIR, Automated Mineral Analyzers and other equipment can reduce these metallurgical risks and provide high-throughput and fast-turnaround mineralogical data (Zahn et al., 2007; Baum 2009, 2014a, 2014b; Baum and Ausburn, 2014; Baum et al., 2014; Ausburn and Baum, 2015) The foundation of good chemical, mineralogical and metallurgical data is a statistically sound, robust sampling approach. Laboratory automation – from sample preparation through chemical and mineralogical laboratories – is a pivotal addition to good sampling as it minimizes sample preparation errors and provides the better data platform for continuous process adjustments (Best et al., 2007).

3.4.1. Porphyry copper operations

Exploratory and routine support of porphyry copper operations has successfully demonstrated considerable value delivery to the business by way of recovery gains and in identifying and treating problematic minerals such as clays.

- The plant mineralogy survey at Candelaria led to a 10% increase of gold recovery, lime dosage reduction by 72%, and a reduction of copper losses in tailings by 16% relative. It is apparent from these outcomes, that about US $ 5.5 million additional revenue was gained the first year after the survey through enhanced gold recovery alone.
- The use of daily blast hole XRD mineralogy is of considerable economic importance to (a) alerting the mill of detrimental ore characteristics (e.g. pyrite depression) and/or (b) if Ore Control needs to blend the feeds to help remove spikes in deliveries. Better mineralogy of feeds in the case of an associated pressure oxidation plant can avoid several days of vessel cleanout which could save > US $ 600,000 of deferred production.
- The losses of unplanned Cu-Mo-Au concentrator shutdowns can range from < US $ 1 to > 2 million per day.
- High variance in ore/rock alteration can continuously "drain the budget". A 0.5% combined Cu-Mo-Au loss (related to high ore type variance), in a +/- 100Kt/d concentrator, can amount to US $ 18–30 million per year.
- A long-term pyrite dilution in the copper concentrate, again, in a +/- 100Kt/d plant can result in US $ 50–70 million losses per year.

The experience described above (and the benefits from 6 other concentrator surveys) confirm the conclusions made by Lotter and Laplante (2007a): "*…surveying of operating concentrators – with the view towards flowsheet improvement opportunity – has long been a valuable field*

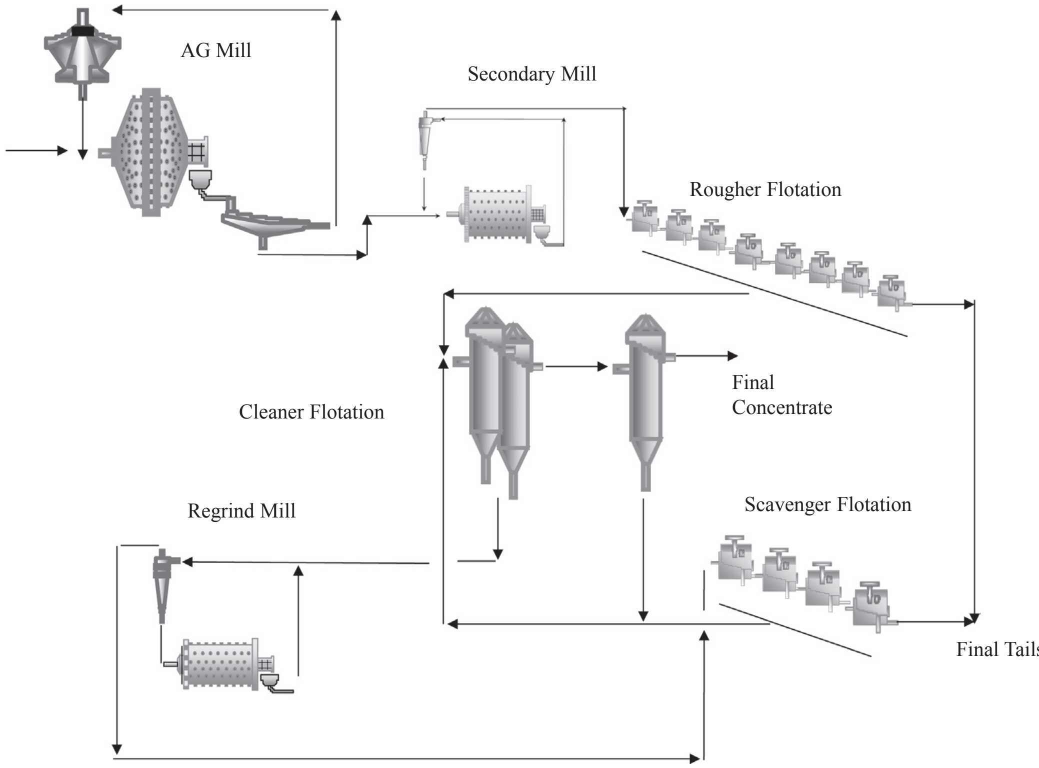

Fig. 10. Rearranged Flowsheet at Raglan using Four Changes Identified in the 1998 Survey.

of endeavor, but has seen little publication…

3.4.2. Heap leach operations

Hydrometallurgy of copper is a complex interaction of feed materials, process parameters, site practices, reagents and the variance thereof. As Baum et al. (2013a) have shown, operations which utilized a strong ore characterization program prior to start-up and process mineralogy in their day-to-day plant practices exhibited high extraction and good overall metallurgical performance.

The daily use of quantitative XRD + NIR Clay Mineralogy of ore feeds for heaps or concentrators (Allen et al., 2007; Brandt et al., 2011), including new spectral models via NIR/FT-NIR, enabled select operations in Arizona, Chile and Peru to develop a linear equation forecasting the milling rate for better crusher operation and subsequent heap leaching. In addition, the blast indices were improved, optimised routing and/or better placement of high-clay material reduced geotechnical heap failures and avoided permeability problems, excessive ponding and/or channelling.

- High clay contents can rapidly plug ore shoots (unless blended) or, worse, the high clay feed will reduce/destroy the permeability of lift areas on the heap pad. A plugged shoot event alone may equate to US $ 80,000/event of deferred production.
- Large geotechnical heap failures can cost up to or over US $ 15 million per case.
- In one mine, the daily use of XRD/NIR mineralogy on blast holes contributed to US $ 510,000/year in reduced acid consumption (amongst other benefits).
- A Heap Leach Survey via automated mineral analyzers performed on select heap modules at Cerro Verde (Fennel et al., 2005) identified several major features for leach improvements. The direct gain from this was estimated at a several million US $ increase/year through better handling of high-clay ores. The fact that the leach cycle can be profiled using automated mineralogy and produce results equivalent to expensive and lengthy column leach test constitutes an additional value potential in the range of US $ 0.6 - $ 2 million/year.

As Gu et al. (2014) pointed out, the economic values are derived from the concerted effort of process mineralogy and metallurgy, specifically when continuous plant improvements are made. If one were asked to express the business value of best practice mineralogy under one heading, it would be “risk reduction”. Consequently, we need to eliminate the false economic thinking established from long-term misleading conclusions that mineralogical analyses are expensive.

4. Concluding remarks

A powerful modern toolbox of sampling, geometallurgical unit definition, qualitative and quantitative mineralogy, and laboratory testing, now exist and are available so that it is possible to deliver significant business value to projects and operations if used correctly at a best practice level. Examples of benefits have included green fields as well as brown fields operations’ brownfields retrofit to ‘mature’ plants.

There are a number of reasons why this value is not always realised. If poor or non-representative sampling occurs, then specimens and not samples are elected for analysis and metallurgical tests can lead to inappropriate analysis and/or inadequate interpretation. This includes the selection of the wrong equipment and/or analysis technique, or the incorrect application of a dataset.

Without an understanding of the implications of the measurements, effective communication within the processing team, (i.e. geologists, mining engineers, mineralogists, process engineers & mineralogists, chemists, environmentalists etc.), and the effective implementation of changes, the information is useless and the resources are wasted. Open engagement between disciplines and with process mineralogy specialists will help manage these challenges and ensure the success of such projects is realised.

The biggest risk is short-term ‘cost reduction’ thinking rather than the longer term ‘value’ focus. Most importantly if the value is not recognised throughout the organisation and operational priorities are on short-term cost reduction, then appropriate resources will not be allocated by the various stakeholders including; mining companies, equipment suppliers, research and development companies as well as education and training providers. Without the appropriate skills and expertise to operate the expensive equipment and interpret and analyse the data obtained, the investment is wasted and operational risk remains high.

In the future, the need for effective process mineralogy is expected to increase. New, more complex operations, requiring integrated and sophisticated use of current and future knowledge will need to be developed to overcome technical, environmental or societal considerations; for example when the excessive use of energy and water cannot be tolerated or permitted. Innovative and novel technologies, and the skills to utilise them to process lower grade deposits can be developed. The mineralogical knowledge will be essential to the provision of minerals and metals for a sustainable world.

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