Pre-Concentration – More than Bulk Ore Sorting

L Pyle[1], *W Valery*[2], *P Holtham*[3] *and K Duffy*[4]

1. Minerals Processing Engineer, Hatch, Brisbane, Qld, 4000. Email: lindon.pyle@hatch.com
2. Global Director–Consulting and Technology, Hatch, Brisbane, Qld, 4000. Email: walter.valery@hatch.com
3. Senior Consultant, Hatch, Brisbane, Qld, 4000. Email: peter.holtham@hatch.com
4. Senior Process Engineer, Hatch, Brisbane, Qld, 4000. Email: kristy.duffy@hatch.com

Keywords: pre-concentration, sensor-based ore sorting, dense medium separation, efficient processing, gangue rejection

ABSTRACT

The mining industry is becoming increasingly focussed on pre-concentration due to its potential economic and environmental benefits, particularly for the more complex, low grade, high throughput operations now being developed. Advances in technology have seen bulk ore sorting (BOS) trialled at several operations. However, BOS should not be seen as a 'silver bullet', it can offer significant benefits, but only under the right conditions. Deposits with interspersed mineralogy or disseminated, fine-grained minerals can be difficult for BOS. Large batches of barren or low-grade material are required for BOS to be effective.

Hatch has recently completed several studies for clients who believed that bulk sorting was the most suitable pre-concentration technique for their operation, based only on an assumed sorter recovery or on laboratory trials of the chosen sensor. However, minimal consideration had been given to the heterogeneity of the deposit, the achievable separation batch size or sensor repeatability.

Evaluation of BOS needs to consider both heterogeneity and sensor suitability. The valuable or gangue mineral must be present in such a way that it can be separated effectively in the large bulk sorting batch sizes, without competing minerals or changing ore characteristics affecting the metal upgrade or recovery. Also, the sensor chosen must be able to detect either the valuable mineral, a proxy for the valuable mineral, or a gangue mineral for rejection with sufficient accuracy and precision (repeatability).

This paper presents three recent case studies, emphasising markers which indicated that bulk sorting was unsuitable for the project whilst highlighting the pre-concentration technologies which were applicable (including particle sorting, and coarse particle gravity separation). The modelling methodology, equipment sizing and forward work plans developed to verify these alternative solutions are also presented, demonstrating the key factors contributing to a thorough scoping assessment.

INTRODUCTION

The value of pre-concentration is being increasingly recognised for both brownfield expansions (targeting increased metal production within existing throughput constraints), and greenfield projects seeking opportunities to minimise CAPEX, OPEX and reduce the production of fine wet tailings, as part of a suite of technologies that will enable more eco and resource efficient processing (Duffy, et al., 2015) (Adair, et al., 2020). This interest is evidenced by the launch of the inaugural AusIMM Preconcentration Digital Conference in 2020.

Several technologies exist that can be used for pre-concentration, as shown in FIG 1. Selective mining applies an appropriate grade control block size to minimise dilution and can be coupled with selective blasting which involves the application of less blasting energy in waste sections to produce coarser fragmentation of waste, and higher blasting energy in ore sections to produce finer fragmentation of ore.

Pre-screening exploits the natural deportment of valuable mineralisation into finer size fractions following breakage, to reject a coarse, barren oversize.

FIG 1 – Summary of technologies available for pre-concentration

Sensor based sorting uses sensing technologies such as prompt gamma neutron activated analysis (PGNAA/PFTNA), magnetic resonance (MR), x-ray transmission (XRT) and x-ray fluorescence (XRF) to measure element or mineral concentrations. Bulk ore sorting involves the measurement and separation of batches of ore (either on a conveyor or in a shovel bucket), while particle sorting involves measurement and separation of individual particles. Bulk and particle sorting can also be used in combination, with bulk sorting followed by scavenging of bulk rejects using particle sorting.

Gravity concentration uses density differences between valuable and gangue minerals at a coarse size to produce a concentrate either by jigging or dense medium separation (DMS).

Coarse particle flotation involves recovery of partially liberated valuable mineral particles following coarse primary grinding (with a P80 of around 600 μm). The idea is to liberate the gangue and reject barren material at a coarse size, rather than having to grind the entire feed to the fine size required for complete liberation of the valuable mineral.

There is a growing interest in bulk ore sorting (BOS) due to the potential for significant energy and cost savings as well as environmental benefits, with site trials being undertaken at operations worldwide, such as the deployment of PGNAA and XRF sensors at the Anglo American Mogalakwena PGE mine (Scott, et al., 2020) and MR trials for copper sorting at the Cozamin mine (Beal & Singh, 2020). Ore and waste material can be separated at high throughput, close to the mine (where in situ heterogeneity is at a maximum), with minimal operating costs and without requiring significant changes to downstream circuits. However, successful bulk ore sorting requires the appropriate sensor selection to detect a valuable (or gangue) element/mineral with sufficient accuracy; and sufficient heterogeneity in-situ and after mining to enable effective separation (Duffy, et al., 2015).

USING GEOSTATISTICS TO EVALUATE DEPOSIT AMENABILITY TO BULK ORE SORTING

Effective bulk ore sorting is dependent upon sensor selection and the presence of heterogeneity both in-situ, and after mining, that enables large volumes of material to be designated as ore or waste. Several approaches to quantify the in-situ heterogeneity of orebodies have been presented in literature, including the use of fractals or examination of grade variation along composite drill hole lengths. Although "down-the-hole" analysis provides an initial indicator of grade variation and can be used to support further detailed analysis, linear variability from drill core lengths is not representative of the variability of grade in three dimensions. Different degrees of variability can be observed in different directions (anisotropy) depending on how the deposit is formed. Both methods fail to consider the impact of mining, load and haul, ROM stockpiling and crushing activities on how the in-situ heterogeneity translates to an on-belt grade variation for bulk ore sorting.

Mineral deposit grades are spatially correlated, which has a direct impact on variances within a block or parcel of ore. Classical statistics consider random and independent variables, so cannot adequately assess grade variability in a deposit, whereas geostatistics considers the spatial relationship between variables. In mineral resource estimation, the initial data is derived from diamond drill core or reverse circulation cuttings, which represents a small sample volume relative to the volume to be mined. The volume considered is also known as the support size in geostatistical terms, and as the support becomes larger and samples are composited to a larger scale the distribution of grade will become less variable (Harding & Deutsch, 2019).

In geostatistics, the covariance or variogram describes the degree of spatial dependence of a variable. The variogram consists of three key components, the nugget effect, range (distance in which variables are dependent) and the sill, as shown in FIG 2. To encompass the spatial relationship and variability in different directions, 3D variograms are modelled using an ellipsoid (also shown in FIG 2). Thus, providing a much more representative estimate of variability for the deposit considering anisotropy.

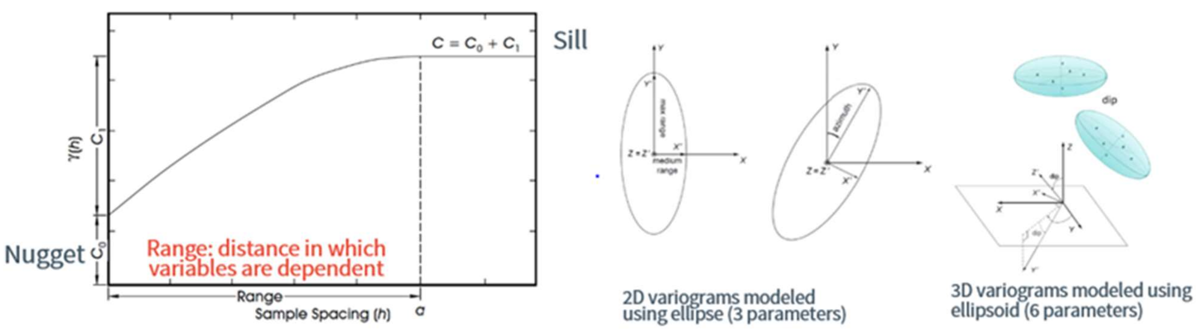

FIG 2 - Estimating 3D variability with geostatistics

Geostatistics can be used to evaluate the grade variation at different scales (different batch or block sizes) by calculating the dispersion variance. Dispersion variance is used to quantify the dispersion of the variable (in this case grade) as a function of the geometry and size of the domain. The dispersion variance, D^2, increases for smaller block sizes as the variability is averaged (or smoothed) when moving to larger blocks (a change of support), as shown in FIG 3.

The dispersion variance of smaller blocks or parcels *v* (i.e. a small support), inside a larger region/support, *V*, is the difference between the average variogram values (Gammabar) calculated on *V* and *v* (Brooker, 1991), shown numerically in Equation 1.

$$D^2(v,V) = \bar{\gamma}(V,V) - \bar{\gamma}(v,v) \quad (1)$$

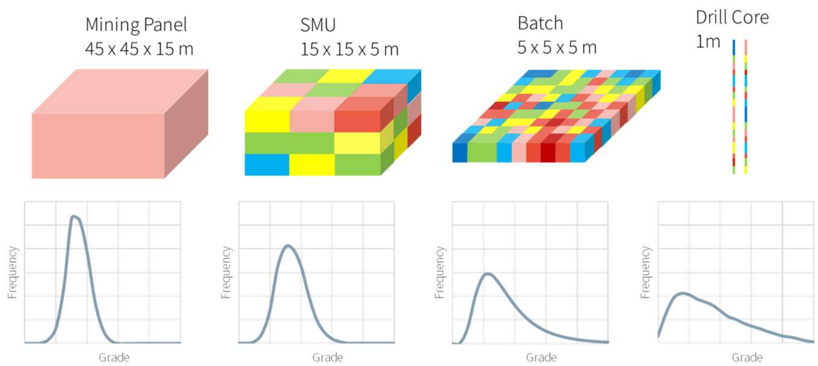

FIG 3 - Distribution of grade variability at different block/support sizes

Gammabar ($\bar{\gamma}$) in the equation above is calculated as the average of all possible vector pairs (head & tail) of grades over the range of the variogram. These pairs are set using a lag distance, where both ends of the vector sit within the mining block V. This sum is conducted for all pairs in all three axis directions, which is the critical step that allows this method to consider variation in three dimensions. The Gammabar calculation is repeated for the smaller batch size, v, within the larger V, to obtain two different values as inputs to Equation 1 (Isaaks & Srivastava, 1989).

To evaluate the potential for bulk ore sorting, it is important that the selection of the batch and block sizes v and V consider the mining method. The small v represents the smallest scale at which heterogeneity is maintained for sorting. It should be selected to compensate for the loss of in-situ heterogeneity through the mining process to represent grade variation as seen on the conveyor belt, for example 5 x 5 x5 m batches. The large V is determined by the mining block size – for existing operations this can be the grade control block model (e.g 10 x 10 x 10 m), or it may be even larger for greenfield projects e.g. the selective mining unit or mining panel size. The greater variation of the smaller block grades (compared to the larger mining blocks) determines the amount of material that could be rejected by the bulk sorter that would otherwise form part of the plant feed due to grade averaging within the larger mining blocks. The block size selection and dispersion variance calculation methodology has been published by Valery et. al (2016) and Reple et. al. (2018) as a case study evaluating the Phu Kham operation.

In summary, to quantify the variation in grade and thus potential benefits of batch ore sorting on each ore domain, the authors follow a five-step process:

1. Obtain the semi variogram function for the valuable element and ore domain/lithologies in the deposit. Each variogram should include 3 sets of data (sill and ranges in each direction).
2. Calculate Gammabar (average variogram value) between all head and tail pairs (grades) within the large mining block V, summing vectors in each of the three semi-variogram directions. Repeat this calculation for the small volume v.
3. Calculate dispersion variance, D^2 as Gammabar (V) minus Gammabar (v)
4. With each large mining block size V having a known average grade (via methods such as kriging, the grade control model or using the domain average grade), the distribution of grades inside the large block V when it is sorted in the small batch v can be constructed.
5. Apply the grade distribution to all small v batch portions inside V, the grade and tonnage values using the batch volume and ore specific gravity can be calculated and tonnage-grade curves plotted, which shows the mass of material rejected as an increasing cut-off grade is applied.

CASE STUDIES

The three case studies below summarise recent projects conducted by the authors to assess pre-concentration technologies. In each case, bulk ore sorting was evaluated but was determined to be suboptimal for the application and alternative strategies were recommended. These cases are presented to highlight different factors that affected bulk ore sorting suitability. Case 1 describes a greenfield, underground porphyry copper ore, Case 2 an open pit copper-cobalt operation processing marginal stockpiles, while Case 3 presents an analysis of bulk ore sorting for a greenfield open pit gold project.

Case Study 1 – Underground Porphyry Copper Ore

This study involved evaluation of potential pre-concentration routes for a greenfield porphyry copper project, with the copper occurring predominantly as disseminated covellite and pyrite-covellite veinlets. The proposed mining method was block caving, feeding a conventional SABC circuit followed by flotation to produce separate copper and pyrite concentrates. Initially bulk ore sorting appeared to offer significant potential as the low selectivity mining method would inevitably introduce large amounts of barren and low-grade material into the plant feed. Significant CAPEX savings were anticipated from rejecting a portion of the feed, thereby reducing the size of the concentrator. Coarse gangue rejection was expected to help alleviate challenges faced by the project in obtaining approval for wet tailings disposal.

Bulk ore sorting facilitates more accurate implementation of the selected cut-off grade, and the optimum cut-off grade will be dependent on the economics, mining methods, and downstream processing. In the case of block caving, there is very little scope for grade control in the mining operation; everything within the cave is mined which includes internal dilution and dilution from the cave walls and roof. Bulk sorting may be the only way for a caving operation to have some control for the implementation of cut-off grade.

However, the use of block caving poses significant challenges for successful bulk ore sorting. The disseminated nature of the orebody will limit the ability of BOS to reject totally barren batches of material. In addition, combining crushed ore underground from many draw points onto a single surface transfer conveyor will significantly reduce the in-situ heterogeneity. Therefore, Hatch evaluated BOS alongside alternative pre-concentration technologies including screening, DMS, particle sorting and coarse particle flotation to determine which options were viable for further study.

To overcome the lack of deposit-specific data, an estimated primary crusher product PSD was generated based on data from a comparable underground block caving Cu operation. The Cu distribution in this stream was then calculated using the technique described by Bazin, et al. (1994). This approach has been shown to be valid for many other projects (Runge, Tabosa and Holtham (2014). More recently, Ehrig et al. (2020) showed the Bazin relationship holding for a given ore over two separate size ranges, 3 – 200 mm and 13 – 150 µm. It is acknowledged that this technique cannot be used to estimate assay distributions to coarse fractions from flotation feed (cyclone overflow) samples due to the bias introduced through cycloning and density effect. Ideally, in an operating plant, coarser samples of SAG mill feed or secondary crusher product would be available for calculation of the ROM assay, however in greenfield projects crushed drill core may be the coarsest size by assay data source available.

Assay by size data for <1 mm crushed drill core samples from the project orebody was used as input to this Bazin analysis in order to estimate the Cu distribution in the primary crusher product PSD, as shown in FIG 4.

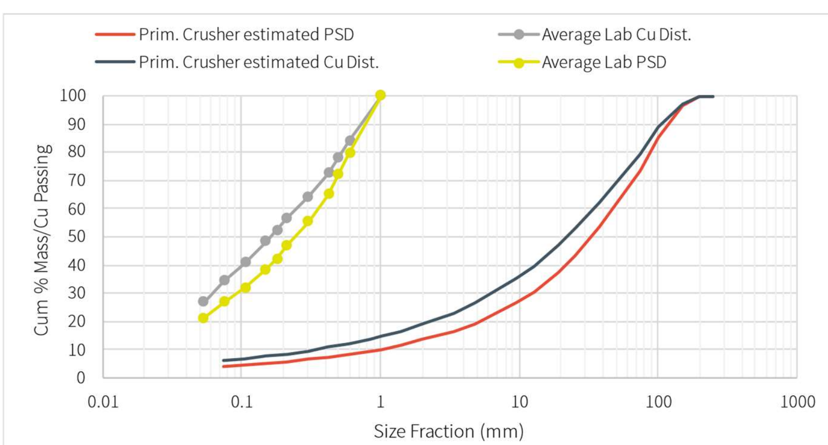

FIG 4 - Overall primary crusher product mass and estimated copper distribution with laboratory data

The data in FIG 4 was initially used to assess the potential for pre-concentration by pre-screening. By varying screen aperture size, mass and metal recovery curves to undersize were produced. Results from three domains in the orebody are shown in FIG 5 compared with published examples from the literature. In contrast to the published studies, which showed recoveries up to 83 % Cu in 60 % of the mass, this case study showed only 65 – 70 % Cu recovery depending on the domain being processed. The results are much closer to the parity line compared with the published data indicating a low potential for upgrade by screening alone.

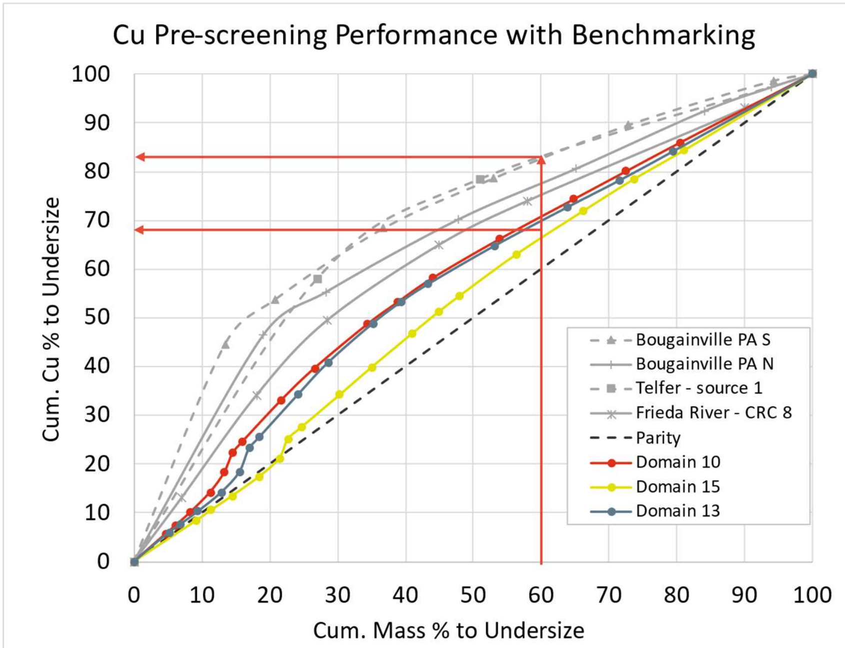

FIG 5 - Pre-screening benchmark cases and case study cumulative mass per copper curves

The amenability of the deposit to BOS was analysed using the geostatistical approach outlined earlier based on variograms from resource definition drill holes. Whilst the batch size that can be measured by a sensor on a conveyor may be only 10 – 40 tonnes (depending on measurement time), a block of this size cannot be mined without mixing with adjacent blocks, and in the case of a caving operation, also mixing with material extracted from other draw points. Therefore, the size of the small block used in the dispersion variance calculations should be larger than the batch size that will be measured by the sorter to account for this dilution. However, it is difficult to know what an appropriate size for v should be, as there is no actual data to validate models and assess the degree of mixing. A small block size, v, of 3 x 3 x 3m has been used in this analysis, representing approximately 80 tonnes to account for mixing before the bulk sorter. The large block size, V, was taken as 45 x 45 x 15 m based on the block size of the geological model. The grade variation of these small blocks within the larger mining blocks of 45 x 45 x 15m was then calculated. By applying a progressive cut-off grade to the smaller blocks, cumulative mass and metal recovery curves for bulk ore sorting were generated. At the proposed sorter cut-off grade of 0.5 % Cu, 93 % of the copper could be recovered in 85 % of the mass, achieving a plant feed grade upgrade ratio of 1.1.

As the scope for BOS in this project seemed limited, alternative methods such as particle sorting were examined that allow for separation on an individual particle basis and reduce the impact of mixing during the mining process.

Compared to other alternatives, particle sorting requires significant additional material handling and crushing to produce the narrow feed size range required for sorting. Both coarse (45 – 135 mm) and fine (15 – 45 mm) sorting circuits were proposed to meet design requirements of a 1:3 ratio of smallest: largest particle in the sorting feed stream. In this case, the +135 mm and -15mm streams report direct to the plant, however this should be evaluated on a case-by-case basis depending on metal deportment by size. Sorter throughput is highest when processing coarser particles, thus careful selection of sorting fractions in consultation with vendors is necessary to balance sorter performance and accuracy against capital cost. Penetrative XRT sorting technology was recommended for this application due to the presence of copper within sulphide minerals that can be identified based on their density differential from gangue.

No particle sorting test work was available to inform the study, thus data from published sorter performance treating copper sulphide ores was used to provide a range of expected performances, (FIG 6). Particle sorting performance is strongly dependent on ore type, mineralisation, liberation and of course sensor type, so it is recommended that initial sighter test work is performed on the main ore domains to obtain performance values. Initial test work is often carried out by suppliers for little or no cost and allows a significant improvement in the study outcomes to be delivered, even at a concept level.

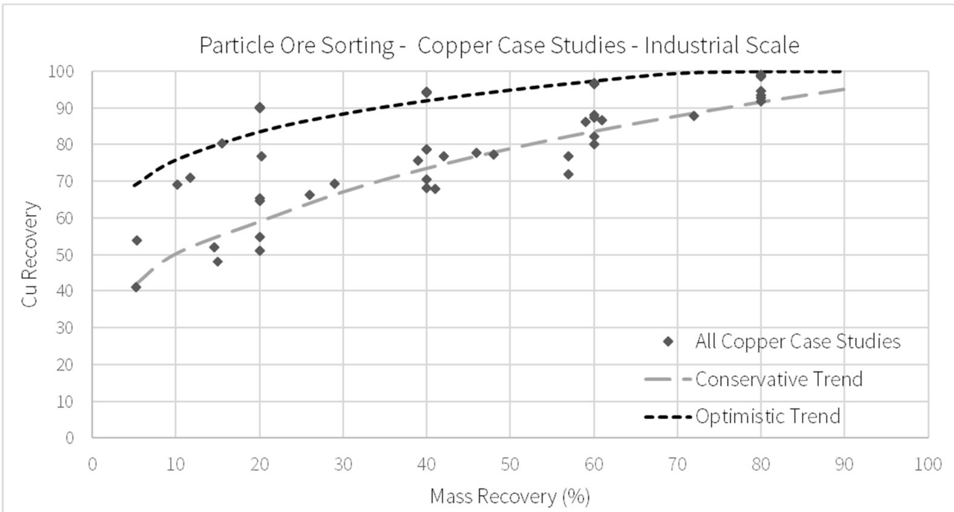

FIG 6 - Case studies of industrial scale Cu particle ore sorting with fitted trendlines

Gravity concentration via jigging and DMS were also evaluated as part of this project. With the simplicity, lower operating and capital cost of jigging, a trade-off against the improved upgrade and shaper separation performance of DMS.

Examination of the dominant copper and gangue minerals in the deposit identified a potential pre-concentration opportunity. There are distinct density differences between the primary quartz and alunite gangue (S.G ~2.7) and the denser sulphides (covellite & pyrite S.G ~4.7). However, separation of perfectly liberated minerals, is not achievable. To estimate more realistic mass and Cu recovery values, drill core data with mineralogical composition at 4m intervals was used to calculate the specific gravity of each length (FIG 7). By applying a progressive density cut point to the drill core, the resulting Cu recovery and mass rejection was predicted. With an accurate gravity separation method such as DMS, a sharp separation at an S.G of 2.7 could recover 94 % Cu whilst rejecting 31 % of the mass (FIG 7). This application would clearly have to be verified through more extensive test work, however it revealed potential that would otherwise have been overlooked if bulk ore sorting was the focus to the exclusion of all other pre-concentration methods.

Furthermore, whilst mixing would occur during mining by block caving, pre-concentration methods that separate individual particles (like DMS and particle sorting) are not affected by the mixing during mining which affects the performance of BOS due to the separation of batches of material. Therefore, a higher upgrade can be achieved (if suitable ore characteristics exist), but with the trade-off of higher cost and lower throughput than BOS.

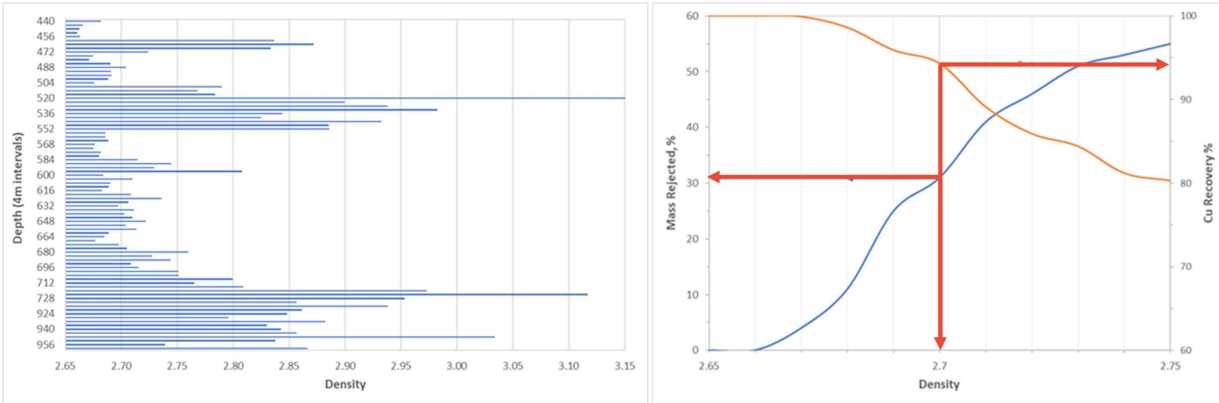

FIG 7 - Density variation down the drill hole (left) and resulting Cu recovery and mass rejection (right)

Finally, the use of coarse particle flotation cells for pre-concentration further downstream than the above alternatives was investigated. If liberation and heterogeneity limit the applicability of other pre concentration options, coarse particle flotation may still be viable. It may also be used in combination with other pre-concentration options for further energy and cost savings. If flotation could be performed at a P_{80} of 300 – 500 µm rather than 100 µm, the potential energy savings in comminution would be around 30 – 50 %. Coarser grind sizes also have the potential to significantly reduce operating costs for power and grinding media. They also result in coarser tailings streams creating certain operational advantages and cost reductions at sites that incorporate sand embarkments, tailings filtration or paste backfill.

The flotation of coarser particles within conventional mechanical cells is typically limited by turbulence that causes bubble-particle detachment in the pulp phase or drop back from the froth phase of heavier particles. This is further exacerbated by the poor liberation of coarse particles in the 150–200 µm range which reduces the strength and amount of bubble-particle contact.

The limitations of conventional flotation machines can be overcome through the utilisation of a fluidised-bed flotation machine specifically engineered for the selective recovery of feeds containing very coarse particles. In this study, the Eriez HydroFloat separator was assessed, as it is the only coarse particle flotation technology commercially available and recently applied on an industrial scale to base metals in a tailings scavenging duty (Vollert, et al., 2019). By using a quiescent, aerated fluidised bed, the turbulence commonly found in a mixed-tank contacting environment is greatly minimized. As a result, delicate bubble-coarse particle aggregates are more likely to report to the concentrate without disruption. The absence of a continuous froth phase minimises drop back that can occur at the pulp/froth interface. Furthermore, the HydroFloat operates most effectively with a feed tailored to a narrow size range, typically this is a top to bottom size ratio of 5:1.

Two applications of coarse particle flotation (CPF) technology are widely known in industry – the coarse gangue rejection (CGR) flowsheet and the tails scavenging flowsheet. Tails scavenging duty applies a CPF circuit to the final flotation tails stream to target metal losses in coarser size fractions e.g >150µm. In this duty the primary aim is improved recovery, which justifies the installation of the CPF circuit, with a secondary benefit being the potential to increase circuit throughput without the requirement to install additional grinding power to maintain the optimal flotation grindsize. In a coarse gangue rejection flowsheet, CPF technology is fed from grinding cyclone underflow to produce a coarse low grade concentrate and a coarse barren tail. A comparison of these two circuits based on testwork and plant data at the Cozamin Cu-Pb-Zn-Ag operation published by Regino et. al. (2020) found that the CGR flowsheet allowed 30% of final tails to be removed in this coarse barren fraction, reducing the size of the tailings storage facility and pumping requirements. Furthermore, the capacity of the flotation circuit could be reduced by 40% compared to a conventional plant flowsheet. By coarsening the grinding circuit product size to produce suitable feed PSD, grinding power requirements are reduced, compared to grinding the entire plant feed to a conventional flotation feed size, with an expected power reduction of 29-50%, and in a greenfield application this would also reduce plant footprint and civil/structural costs. There are currently no CGR circuits operating in base metals, as the tighter integration of HydroFloat with the plant flowsheet poses a significant risk to profitability and operability dependant on CPF technology. However, in the context of pre-concentration, with a focus on gangue rejection at coarser size fractions prior to conventional processing, this flowsheet offers the most significant benefits.

Installation of CPF within the primary grinding circuit was proposed in this case study to reject coarse gangue ahead of the main flotation circuit. The comminution circuit feed would be ground coarser than the current target grind size P80 of 180 µm, with grinding circuit product (cyclone overflow) classified using screens and cyclones, or a combination of these technologies, to prepare the feed to both fine (conventional) and coarse particle flotation stages. The fines of the classification stage would feed the proposed rougher flotation bank and the coarser stream would feed a fluidized bed coarse particle flotation cell. The coarse flotation concentrate would require further grinding and cleaning to produce a final concentrate and would, therefore be sent to the proposed rougher scavenger concentrate regrind ball mill. Coarse flotation tail would be rejected and combined with rougher-scavenger tail to form the final tail.

Pilot test work has shown that a fluidised bed cell could recover up to 70 % of composite sulphide particles between 300 – 600 µm with a sulphide liberation greater than 15 %, and that this would increase to >90% recovery of sulphides in finer fractions (Vollert, et al., 2019). Previous studies by Hatch, for a copper ore, also show that HydroFloat cell recovery could be maintained above 90 % for particle sizes up to 425 µm for copper sulphides with less than 10 % liberated present in the HydroFloat concentrate (Valery, et al., 2020).

In the absence of laboratory HydroFloat testwork to model the circuit performance, liberation data was analysed as an indicator of performance. QEMSCAN mineral liberation data showing the distribution and liberation by size of sulphide (covellite, enargite and pyrite) particles in testwork samples was used to provide an indicator of potential performance. The analysis suggested that approximately 96 % of sulphides below 450 µm had more than 15 % surface liberation and thus were amenable to recovery by coarse particle flotation (FIG 8 left).

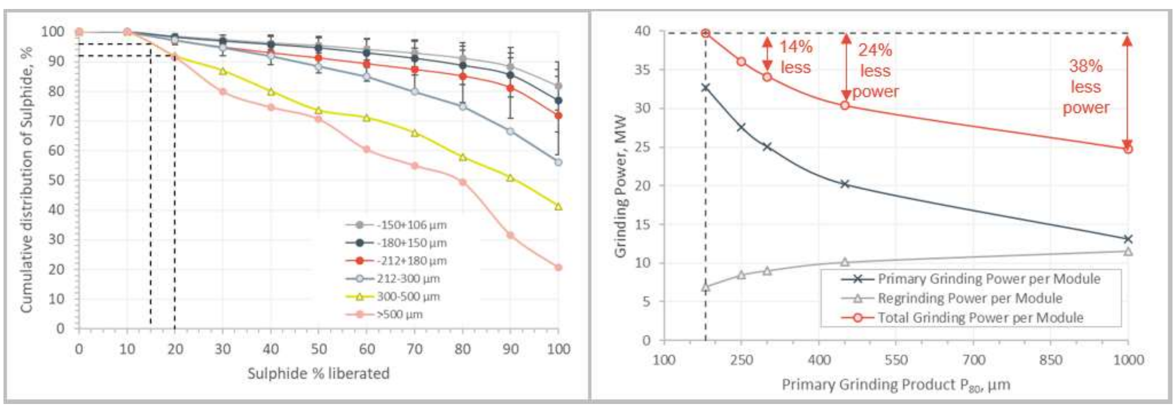

FIG 8 - Cumulative distribution of sulphides (covellite, enargite and pyrite) by degree of liberation (left), and simulated grinding power as a function of primary grinding product P80 (right).

Preliminary simulations and power-based calculations were used to estimate the impact of coarse flotation on the overall grinding energy for different primary grind sizes. Power based methodology can be used to calculate the total SAG – ball mill circuit comminution energy requirements as well as the increase in regrind power consumption that occurs as a result of regrinding the coarse HydroFloat concentrate prior to cleaning. This allows the total grinding power to be calculated for a fixed throughput and specific energy input requirements based on ore characteristics, feed size and product sizes. FIG 8 (right) shows the result of this calculation, with a significant reduction in primary grinding power when moving from the original target grind size P_{80} of 180 µm to 425 µm established as suitable for coarse ore flotation. Despite the increase in regrind energy consumption, the total power consumption decreases by 24 % which is a significant saving. In addition, CAPEX savings from smaller mill sizes as well as a reduction in footprint and civil/structural costs would be significant.

A summary of the performance of the pre-concentration routes assessed during the study is presented in FIG 9. Coarse particle flotation is not shown, as it appears later in the flowsheet than the other pre-concentration options. The potential benefits are not expected to be as significant and do not translate easily onto a mass-metal curve. However, if dissemination and liberation limits the applicability of other pre-concentration options, the potential to reduce energy consumption and increase throughput with coarse particle flotation is significant.

This low-grade porphyry deposit is a good target for pre-concentration due to its large size and thus large volumes of barren dilution material which could potentially be rejected prior to downstream processing. However, the ore may be finely disseminated and experience significant mixing during mining, limiting the upgrade potential of BOS which separates large batches of material. By considering all pre-concentration options, alternative opportunities such as gravity concentration via DMS were defined that would not have been apparent if BOS was the exclusive focus.

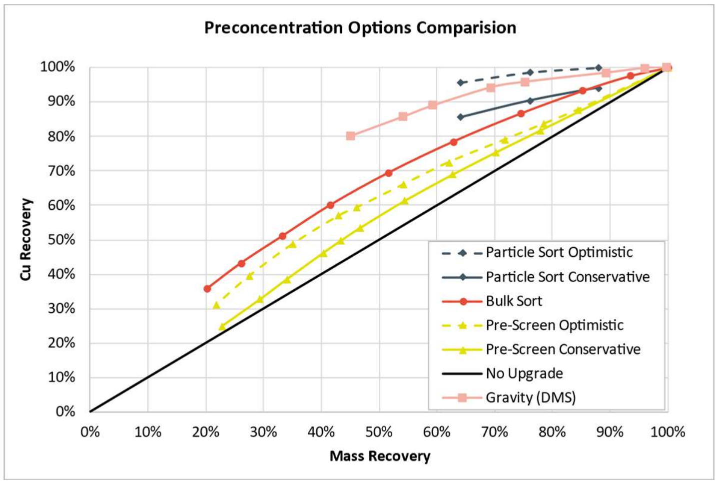

FIG 9 - Comparison of mass and metal recovery curves for preconcentration routes

Case Study 2 – Bulk Ore Sorting of Low-Grade Copper-Cobalt Stockpiles

In this project, the client had a significant volume of stockpiles to be re-processed, built from marginal material accumulated throughout the life of mine (LOM). The stockpiles consisted of oxide and sulphide minerals including malachite, carrolite and chalcopyrite, with gangue primarily composed of dolomite, magnesite and quartz. The initial project scope was to assess the suitability of bulk ore sorting to treat the four largest copper containing stockpiles and also as a proof-of-concept for application to the client's other operations.

There were several issues that significantly limited the potential for BOS in this scenario. An initial study undertaken by the company considered three scenarios for the Cu and Co grades within these stockpiles. However, an assumed sorter recovery of 90 % was used when predicting the copper ore accepted for downstream processing. This value was assumed with no consideration for the ability of the bulk ore sorting system to clearly distinguish ore and waste batches within the piles. The stockpiles were built from marginal material ranging from 0.6 – 1.5 % Cu (with all lower grade material sent to waste during mining) thus the grade variation available to achieve this assumed recovery was significantly limited.

The second area of concern was the competing requirements for the proposed BOS system. Cu and Co both contribute significantly to the overall profitability of the operation; however assay data from stockpile samples (and historical mine data) suggested there was no clear correlation between Cu and Co grades, thus rejecting material based on Cu grade alone has the potential to reject material high in Co. This also has a significant effect on the sensor types that would be suitable for BOS, for example although magnetic resonance sensors are extremely well suited to detecting chalcopyrite, they are unable to detect oxide minerals or cobalt. Alternate sensors, such as PGNAA which determine the full elemental composition of a material, would be required to assess both Cu and Co grade. Further discussions with the client subsequently indicated that the consumption and cost of sulphuric acid used in the downstream oxide leaching circuit was a significant factor in plant profitability, to the extent that even high Cu/Co grade batches would be unprofitable to process if they also contained significant amounts of acid consuming gangue (GAC) – mainly dolomite.

Assay results from 70 very small (20 kg) samples across the stockpiles were assessed to examine the variation in Cu/Co grade and acid consumption and illustrate the impact of multiple cut-off decision drivers for this operation. As shown in FIG 10, sorting based only on Cu grade would discard much of the Co and likewise a sort considering only Co grade would discard much of the Cu (red shaded areas are the reject fractions for the assay samples). Furthermore, a sort based on acid consumption would discard much of both the Cu and Co.

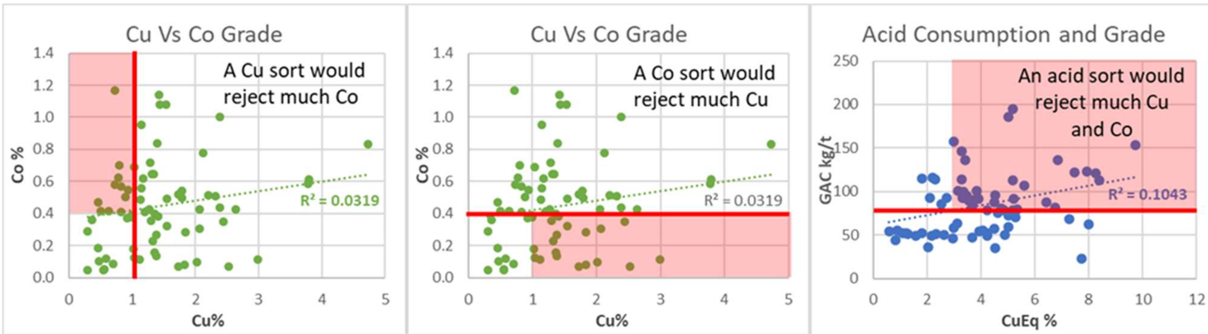

FIG 10 - Impact of competing factors on bulk sorting performance

Two methods were considered to address this issue, firstly sorting using a combined set of rules that would require Cu and Co grades to be above the cut-off threshold (expressed as a Cu equivalent based on the price difference between Cu and Co):

$$\text{Cu Eq} = \text{Cu\%} + \text{Co\% x 6 (For example, for 1\% Cu and 0.5\% Co, Cu Eq} = 4\%)$$

Rules such as the following could then be applied:

If Cu Eq < 4% = Discard

If Cu Eq > 6% and Acid cons > 60 kg/t = Discard

If Cu Eq > 6% and Acid cons < 60 kg/t = Accept

Secondly, calculation of a batch value, using historical regressions from plant data to relate the measured grade of Ca or CaO to plant acid consumption and subtracting this cost from the contained metal value:

$$Batch\ Value = Contained\ Value\ (Cu\ grade\ x\ price + Co\ grade\ x\ price) - Acid\ Cost\ (acid\ consumption\ x\ price)$$

To demonstrate the potential performance of a rules-based sorting algorithm, a series of sorting scenarios with different cut off requirements were applied to the 70 stockpile samples with assay data used to calculate an accept or reject decision. Various rules were applied to assess the improvement in performance by scaling the accepted acid consumption based on the contained metal (contained value), i.e. higher contained value batches justify higher acid processing costs. These scenarios use the Cu equivalent approach as discussed earlier. The best case achieved minor upgrades of ~1.2 and 1.1 for Co and Cu, respectively and ~10% reduction in acid consumption, recovering 78 and 72 % of the Cu and Co in 66 % of the mass. Other scenarios achieved a higher degree of acid consuming material rejection; however, the Cu/Co losses were unacceptable.

The analysis provided insight into the issues faced when multiple competing performance drivers impact the sorting decision. However, the assay samples represent 20 kg batches only and a further reduction in performance would be expected when considering full-scale sorting batches due to mixing and homogenisation.

The third factor which affected the suitability of bulk ore sorting for this project is the considerable mixing and homogenization that occurs during stockpile formation and reclaiming, which will reduce the grade variability. In order to model the change in variability, the grade control block model for the mine (which in this case had a fine resolution: 5 x 5 x 2.5 m) was used as a baseline, along with topography surveys and truck dispatch data, to build block models for copper of the four stockpiles. FIG 11 shows the resulting tonnage and grade curves. Solid lines represent the in situ material (from grade control), and dashed lines show the material properties once stockpiled. If a sorting cut-off grade of 1 % Cu was applied to the in situ material, approximately 15 million tonnes would be above cut-off grade, and this mass has a grade of 1.3 % Cu, as shown by the red arrows. However, if the same 1 % sorter cut-off grade was applied to the stockpiled material, only 10 million tonnes would be above this cut off, and the material would have a lower grade of only 1.08% Cu.

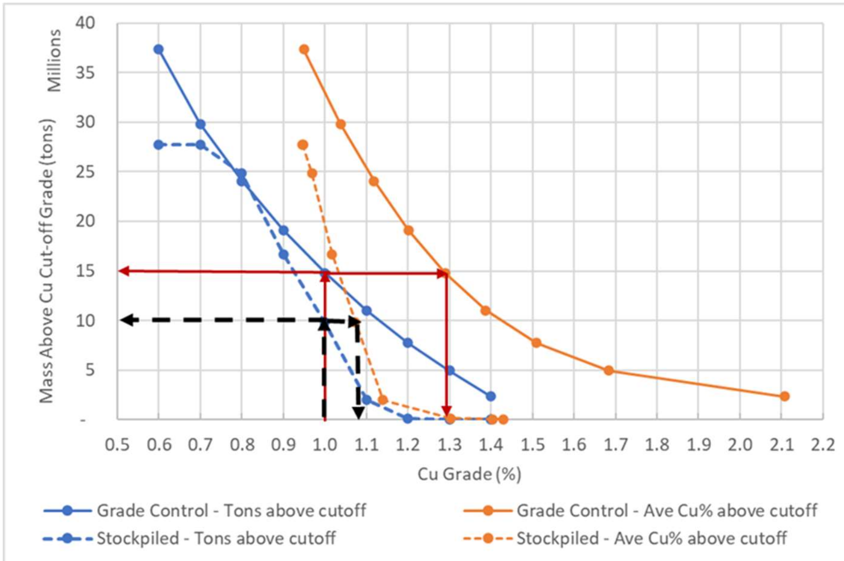

FIG 11 - Comparison of Cu grade and tonnage curves for in situ and stockpiled material

The stockpile block models were only populated with data for copper, not cobalt, thus in order to estimate the impact of stockpiling on cobalt variability and sorter performance an alternative approach was required. The grade control blocks with copper grade between 0.6 and 1.5 % Cu were selected as the blocks sent to the marginal stockpiles. The in-situ cobalt grade distribution in these selected blocks was modelled, then the standard deviation of the modelled grade distribution was reduced by a factor to account for the homogenisation of building and reclaiming the stockpiles. This homogeneity factor is related to the number of layers of the stockpile, assumed to be 10 layers in this case.

The homogenisation factor is an empirical measure of the reduction in variability within stockpiles and can be used to quantify the degree of homogenisation from different stockpile designs (De Wet, 1994) (Schofield, 1980).

$$H_f = \frac{S_{in}}{S_{out}} = k \times \sqrt{N}$$

H_f = homogenization factor
S_{in} = standard deviation of input (mined material - stockpile feed)
S_{out} = standard deviation of output (reclaimed stockpile product)
k = parameter in range of 0.5 – 0.7
N = number of stockpile layers

In past projects conducted by the authors grade standard deviation has been reduced between 2 to 4 times through the stockpiling and reclaim process, depending on the number of stockpile layers, significantly limiting the opportunity for bulk ore sorting.

Monte Carlo simulations were performed to randomly extract the blocks from the stockpile and create the cobalt grade distribution accounting for the homogeneity effect when reclaiming the material, as there was no spatial information regarding the blocks in the stockpile. The tonnages and cobalt grades were calculated for each cobalt grade interval (block) so that the stockpile potential upgrade and recovery for a cobalt bulk sort could be evaluated. The cobalt bulk sort final result in FIG 12 is more promising than the copper bulk sort, which is a function of the in situ grade distribution and mineralisation types. It is clear that while some improvement could be achieved for Co with a mass recovery of 60 % recovering ~75 % Co, when both elements are considered in a combined reject or accept decision the ability to reject barren mass is limited.

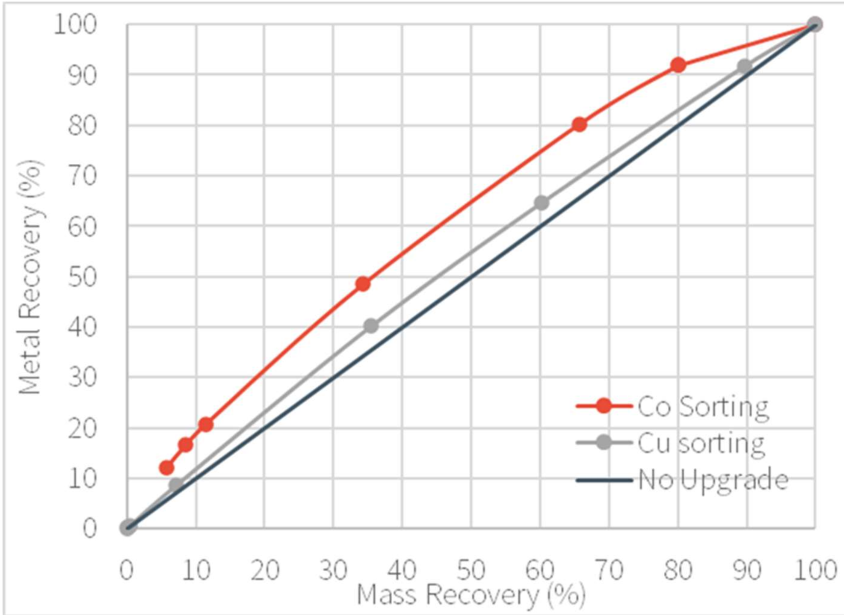

FIG 12 - Overall Cu and Co BOS performance

Alternative methods of pre-concentration were also proposed for this project, including screening, particle sorting and DMS. These act on individual particles rather than batches, eliminating the issues arising from homogenisation and competing sorting factors. However, particle size analysis indicated that a significant mass of ROM material was -10mm which typically bypasses some pre-concentration processes. With no assay data available for this fine fraction, the optimal destination (direct to plant feed or reject stockpiles) could not be selected. The destination of this stream is a key factor in determining the size of the pre-concentration plant and CAPEX requirements to supply the existing process plant at its design capacity. This is illustrated by the concept level CAPEX developed for this project in FIG 13. This estimate was developed based on the requirement to produce 250 t/h of pre-concentration product to feed the downstream plant, with major equipment scaled from past project estimates using the six-tenths rule, and lang factors applied to calculate direct & indirect costs. Options where the fine stream is rejected to waste stockpiles required a significantly higher feed rate and larger equipment sizes, which would make the capital cost prohibitive. Due to their complexity, options such as particle sorting which require additional infrastructure and crushing stages prior to separation have higher CAPEX (~US$ 27 million) and OPEX costs than technologies like BOS (~US$ 14 million in this study). More detailed techno-economic analysis of pre-concentration options conducted by (Pyle, et al., 2020) identified that the overall net present value (NPV) of these projects is strongly affected by the feed grade upgrade (and mass rejection) that can be achieved, thus, a successful application of pre-concentration must be suited to both the orebody mineralogy/liberation and provide financial benefits in terms of NPV and payback periods.

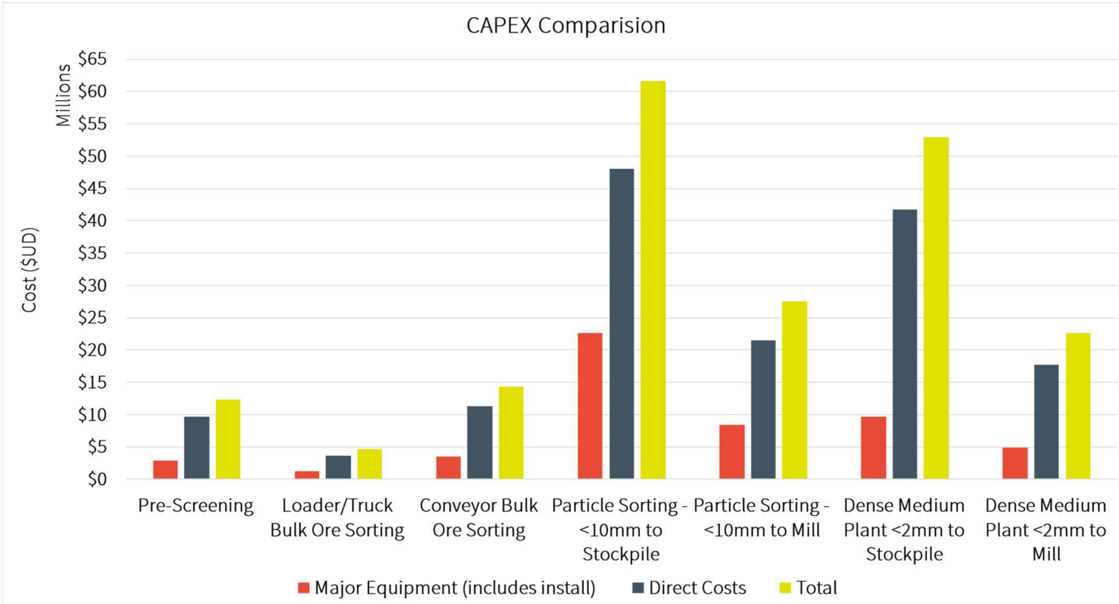

FIG 13 - Concept level CAPEX for pre-concentration circuits producing 250 t/h product in this case study

Case Study 3 – Open Pit Gold Project

The third case study is from a greenfield open pit gold project, processing a low-grade deposit at 0.87 g/t Au with a throughput of 1800 t/h. The aim was to utilise BOS as a low CAPEX option to treat low grade ore comprising one of three domains in the plant feed. This would upgrade the mill feed to reduce CAPEX for an equivalent gold production and reduce mill footprint in the heavily layout constrained site. Hatch was engaged to review previous bulk ore sorting test work and performance assumptions, and propose alternative BOS circuit designs, including in-pit sensing and diversion, shovel-based sorting, traditional conveyor-based sorting arrangements and combined bulk and particle sorting to “clean” the accepts stream or “scavenge” the rejects. Three areas of concern were identified in this project that would limit BOS performance and would be observed in similar applications.

Prior test work had been conducted with sensor suppliers to examine the potential for direct gold measurement using PGNAA (which is a significant development) as well as measurement via proxy elements. Measurement for Au correlated reasonably well with lab fire assay results (FIG 14 left), which initially suggested that direct Au measurement could be feasible. However, these measurements were obtained with a 1-hour sensing time, and upon further investigation, the repeatability of these results raised significant concerns. As per the repeatability measurements in FIG 14 right, shorter measurement durations of 20 and especially 2 minutes resulted in significant scatter in the grade readings obtained compared to the lab assay value, across various samples. In order to exploit the deposit heterogeneity, sorting should operate in the range of 30 secs – 2 min batches. Thus, at a feed grade of 0.87 g/t, the 2-minute result variation observed here would make direct Au measurement using PGNAA unfeasible due to the lack of precision, resulting in mischaracterised batches of ore to waste and vice versa.

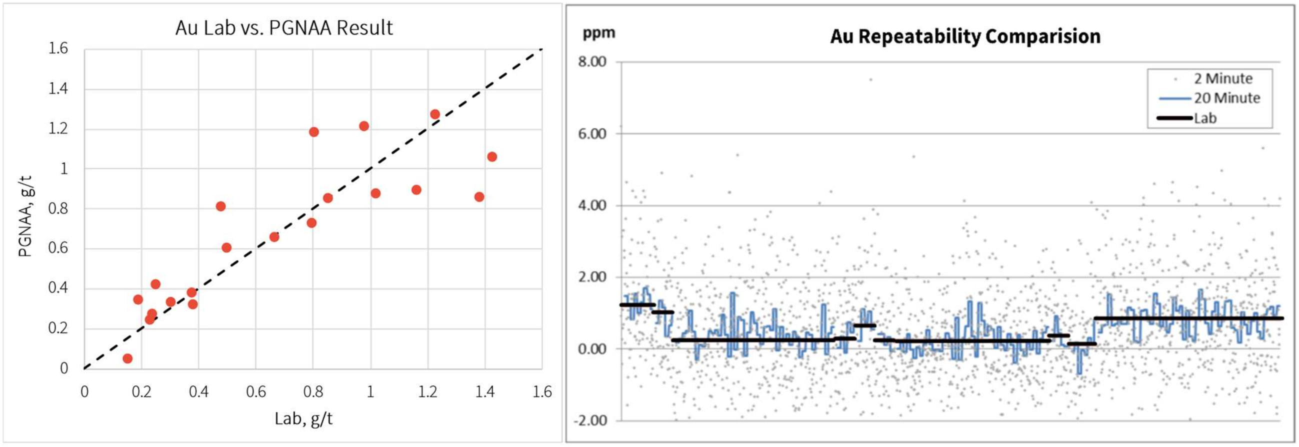

FIG 14 - PGNAA sensor vs. lab measurement (left) and repeatability (right)

A second method proposed for bulk sorting at this deposit was the use of proxy elements based on 3 000 ICP elemental datasets including Si, Mg, Al, Ca, Fe and K as predictors of gold grade. The deposit contains three distinct ore domains as follows:

- Geomet 1: Rich quartz vein with higher gold grade
- Geomet 2: Altered granitoid with higher gold grade
- Geomet 3: Tonalite with disseminated pyrite and lower gold grade

The three geometallurgically different domains each required a separate proxy model, (currently based on around 1000 ICP datasets each) and model regressions yielded R^2 values of 0.22 to 0.57, indicating the equations were a poor predictor of actual Au grade. Although further work was recommended to obtain additional elemental data sets to improve these models, the issue of sorting three different domains remains. Geomet 3, comprising 60% of the deposit, is the primary domain intended for BOS to exploit the lower grade nature of this material (half the grade of Geomet 1 & 2) and thus the potential for upgrade with major mass rejection. However, the three domains cannot be mined independently due to the proposed open pit mining method and the distribution of domains throughout the deposit. Thus, extremely stringent grade control block models and truck scheduling would be required to ensure only domain 3 ore is directed to the sorter, with domains 1 and 2 bypassing direct to the plant. In addition, for mixed domain areas of the pits, the sorter would have to be configured to select between different proxy relationships based on the batch of ore being processed.

The final limitation with BOS in this project was the initial methodology used to assess heterogeneity and therefore the potential mass rejection and Au recovery values. The initial analysis used composited drill core lengths of 1.5m up to 4.4m to represent the volume of ore seen by the sorter on a conveyor up to sorting of ore batches in a truck. The grade from this length was extrapolated into a cube and then each cube could be accepted or rejected with the application of a sorter cut-off grade. However, there are several significant issues with this method that overestimate the performance as follows:

1. Mining methods are typically horizontal in nature, and the variability along a length of drill core in 1 dimension vertically is not reflective of the 3D variability of batches of ore extracted by excavator buckets
2. Mixing during blasting, load and haul and materials handling was not considered in the analysis and would significantly reduce heterogeneity
3. The smallest scale at which heterogeneity can be maintained is the size of the excavator bucket which is used to extract the blasted material (29m^3 for this project) due to the mixing that occurs during mining. Attempting to calculate recovery at smaller scales from either drill core lengths or batch sizes passing through the BOS sensor will overestimate the mass rejection and upgrade that can be achieved.

To show the impact of using composite drill core lengths to estimate mass and gold recoveries for BOS (if Au could be directly measured), the authors compared the method used by the client to results from geostatistical analysis. Three variograms from zones of high, mid and low-grade material were used for geostatistical analysis. The mass and metal recoveries were obtained using the dispersion variance calculation described earlier in this paper, using the mine block model size and loading/hauling equipment sizes to estimate the impact on heterogeneity and thus BOS performance. The strength of this approach is two-fold; firstly, by calculating Gammabar (the average variogram) for both mining block (V) and sorter batch (v) sizes using all three directions of the semi-variogram, the variation in grade in three dimensions is captured. This enables a horizontal grade variation to be captured in the formation of some deposit. Furthermore, it prevents overestimation of BOS potential, for example where vein style mineralisation might result in significant assay variation over 10 or 20 meters of drill core, whereas the mining process which progresses horizontally through a bench wouldn't excavate the variability in this direction. Secondly, by selecting the small batch size based on the mining method e.g. excavator bucket size and truck tray volume, the dilution and mixing introduced through mining to be considered in the BOS performance calculation, preventing an overoptimistic result compared to down the hole compositing which does not capture this aspect.

For this case study, previous down the hole analysis had been used to calculate gold recovery from BOS and using an exponential equation $\left(Rec = 100 \times 1 - e^{-\frac{k}{100 \times Mass\ Pull}}\right)$, mass-metal recovery curves were fitted to the down the hole analysis result by varying the k value from 3.7 to 9.6. This change theoretically approximated sorting performance at different sorting batch sizes from a 12m bench to 1.5m pod on a conveyor. FIG 15 presents the combined mass and metal recovery curves for both methods of analysis, showing the down the hole analysis would indicate metal recoveries of 70 – 80 % in only 30 % of the mass, whereas the more rigorous geostatistical approach indicated only 50 % Au recovery in the same 30 % mass yield.

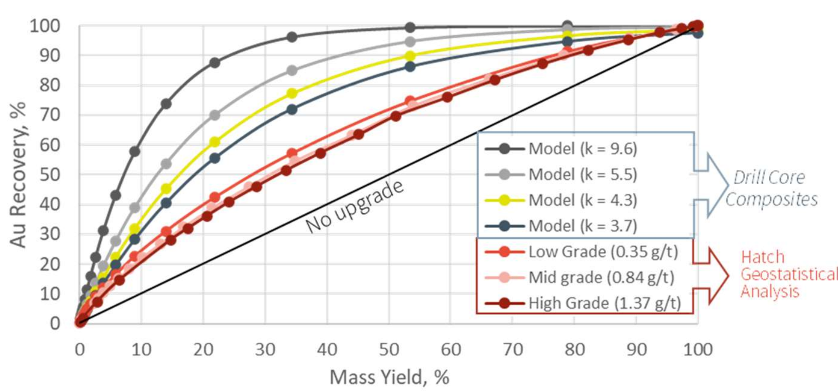

FIG 15 - Mass and gold recovery curves for drill core composites compared to geostatistical analysis

CONCLUSIONS

This paper has presented several recent projects examining the application and benefits of pre-concentration technologies. Bulk ore sorting was a particular focus for these studies due to its relative simplicity, ability to process material directly after mining with no further crushing or screening required upfront and lower CAPEX, OPEX and simplified layout requirements compared to other alternative pre-concentration technologies.

Three case studies were drawn from a variety of operations and commodities to highlight key factors or requirements for the successful implementation of bulk ore sorting. In these cases, one or even multiple instances of the following issues limited successful BOS application:

- Heterogeneity – suitable heterogeneity needs to be present in the deposit in situ and maintained through the mining process to enable effective separation at a truck, shovel or conveyor batch scale depending on the sorting technology being applied. A geostatistical approach is one of the most rigorous ways to quantify grade variation, because it allows for variation in three dimensions to be considered in the analysis and the selection of small v sorting batch size accounts for mixing in the excavation, load and haul stages to establish the grade variation as seen on the conveyor.

- Sensor suitability - all potential valuable or waste elements should be evaluated for sorting to determine which established bulk sorting sensors such as PGNAA/PFTNA, MR or XRF could be applied. Sometimes unconventional approaches may be required to target a gangue element or mineral, or a proxy mineral or element, if direct measurement of the valuable element, such as gold, is unfeasible.
- Mineralogy and liberation – consideration should be given to the value of all elements in the deposit and their recovery via BOS. In some applications, such as the Cu/Co case study, the correlations between valuable minerals may directly impact sorting performance, different elements may not be measurable using the same sensor or disseminated minerals may limit the upgrade potential.

A successful scoping study should consider bulk ore sorting alongside all the other potential technologies to determine which options can deliver a successful pre-concentration outcome. By investing in initial test work using minimal sample masses as part of the scoping level study, the value delivered through these assessments can be significantly improved by delivering orebody specific results, mass balances and CAPEX estimates. BOS certainly has an important role to play in the future of mining and minerals processing, however operations, vendors and consultants need to be aware there are other pre-concentration options which may be more suitable than BOS.

These studies are presented to support future successful applications of bulk ore sorting for the right deposit type with suitable heterogeneity. The selection of the optimum pre-concentration route needs to be evaluated considering the deposit characteristics and in an integrated analysis with mining and downstream processing. These technical aspects also need to align with positive financial metrics. Mining and processing teams cannot operate independently, as maximum benefits from BOS will only be obtained when a shift in mining practices is made to retain in situ heterogeneity as much as possible as opposed to blending it out.

REFERENCES

Adair, B., Revell, R., King, D. & Keeney, L., 2020. *A 'no tailings dam future' - preconcentration strategies 'down the sizes and across the value chain'.* AusIMM, pp. 5-17.

Bazin, C., Grant, R., Cooper, M. & Tessier, R., 1994. *Prediction of metallurgical performances as a function of fineness of grind*. Ottawa

Beal, C. & Singh, S., 2020. *Improved mining recovery with bulk ore sorting and MR technology.* AusIMM.

Brooker, P. (1991). A geostatistical primer. World Scientific

De Wet, N., 1994. Homogenizing/blending in South Africa - an update. 14(1), p. 93.

Duffy, K., Valery, W., Jankovic, A. & Holtham, P., 2015. *Integrating Bulk Ore Sorting into a Mining Operation to Maximise Profitability*. Perth, AusIMM, pp. 273-287.

Duffy, K., Valery, W., Jankovic, A. & Holtham, P., 2015. Resource Efficient Mining Processes of Tomorrow. Sydney, AusIMM, pp. 227-238.

Ehrig, K. et al., 2020. *Preconcentration via waste rejection based on fragment size - Olympic Dam Case Study*. pp. 336-344.

Harding, B. & Deutsch, C., 2019. *Change of Support and the Volume Variance Relation.* [Online] Available at: https://geostatisticslessons.com/lessons/changeofsupport

Isaaks, E. & Srivastava, M., 1989. *An Introduction To Applied Geoostatistics.* New York: Oxford University Press.

Pyle, L., Valery, W., Holtham, P. & Duffy, K., 2020. *Techno-economic comparison of pre-concentration options for a Ag-Pb-Zn deposit*. Cape Town, XXX International Mineral Processing Congress.

Regino, R. et al., 2020. *A Comparision of Two Circuit Applications for Implementation of Coarse Particle Flotation.*

Reple, A., Chieregati, A. C., Valery, W. & Prati, F., 2018. *Bulk ore sorting cut-off estimation methodology: Phu Kham Mine case study*. Minerals Engineering, 4, Volume 149, pp. 105498-105501.

Runge, K., Tabosa, E. & Holtham, P., 2014. *Integrated Optimisation of Grinding and Flotation Circuits.*

Schofield, C., 1980. *Homogenisation/Blending Systems Design and Control for Minerals Processing*. TransTech Publications.

Scott, M., Rutter, J., du Plessis, J. & Alexander, D., 2020. *Operational deployment of sensor technologies for bulk ore sorting at Mogalakwena PGE Mine.* AusIMM.

Valery, W., Duffy, K., Holtham, P., Tabosa, E., Andrade, B., 2020*. Conducting comprehensive evaluations of preconcentration options*. Brisbane, Australia, pp. 345-355.

Valery, W. et al., 2016. *Evaluation of Bulk Ore Sorting for Pre-concentration of Copper Ore at PanAust Phu Kham Operation*. Rio de Janeiro, Brazilian Mining Association, pp. 145-154.

Vollert, L., Akerstrom, B., Seaman, B. & Kohmuench, J., 2019. *Newcrest's industry first application of Eriez HydroFloat technology for copper recovery from tailings at Cadia Valley Operations*. British Columbia.