Relationship between Kansanshi Ore Mineralogy and Copper Recovery Across Different Processing Circuits

Lucy Little*, Quintin Mclennan, Andre Prinsloo, Kashela Muchima, Billy Kaputula, Chiza Siame

First Quantum Minerals Limited, Kansanshi Mine, Solwezi, Zambia

**Corresponding author: lucy.little@fqml.com*

The variability of copper mineralisation at Kansanshi mine significantly affects the efficiencies of the processes used to recover copper. Quantifying this variability is therefore essential to monitor and optimize the performance of the processing circuits used, which include three different flotation circuits (Sulphide, Mixed and Oxide) and an atmospheric leach circuit. An on-site automated scanning electron microscope with energy-dispersive X-ray spectrometry (Auto-SEM-EDS) is used to analyse routine weekly composite samples for each circuit, providing detailed analysis of bulk mineralogy, copper deportment to different minerals, mineral liberation and associations. Weekly mineralogical trends are used to assess changes in process performance, and characterisation of the copper losses has inspired a number of projects to target the specific particle types responsible for the main copper losses. This paper describes the relationship between mineralogy and plant performance at Kansanshi, and discuss some of the advantages and challenges of on-site process mineralogy.

INTRODUCTION

Kansanshi is a copper and gold mine located in the Northwestern Province of Zambia, with an annual production in the region of 250 000 tons of copper and 160 000 ounces of gold. Ore mined from the epigenetic-vein-hosted type deposit is classified into three ore types based on the extent of oxidation and supergene enrichment. Sulphide, Oxide and Mixed ore are fed to separate minerals processing circuits tailored for recovery of the dominant copper minerals in each. After comminution to a target grind size of 80% passing 150 µm, each ore type is subjected to flotation to recover the copper sulphide minerals (chalcopyrite, bornite, chalcocite, covellite). The Oxide float tails and a large proportion of the Mixed float tails are then treated by atmospheric leaching to extract copper from the remaining copper-hosting minerals, which include cuprite, malachite, chrysocolla and delafossite. Some native copper is recovered by flotation and some of it is recovered by leaching. A small portion of the combined concentrate from all circuits is subjected to high pressure leaching and the rest is sent to the Smelter. Sulphuric acid produced during smelting is used for the atmospheric leach.

With assistance from Dr Will Goodall and colleagues from MinAssist and iMinSolutions, Kansanshi set up an on-site mineralogical section within its Processing department in 2015 to provide applied mineralogy services for continuous plant performance improvement. Most features of the set-up that Goodall outlined in 2015 have been maintained to date, with the Zeiss MinSCAN being used for automated mineralogical analysis of weekly composite samples of the key streams for each circuit. In this work, Sulphide, Mixed and Oxide flotation are considered as three separate circuits, and the atmospheric leach that processes

Oxide and Mixed flotation tails is considered as a fourth circuit. Detailed weekly mineralogy reports for each circuit have helped to explain variance in performance, and have underpinned a number of successful recovery and grade improvement projects. Kalichini *et al*. (2017) described some of the key successes of the Kansanshi mineralogy program, focusing on the value delivered by the program in the Sulphide circuit.

Work involving Kansanshi ore mineralogy that has been reported to date has typically focused on mineralogy or processing associated with a particular circuit at Kansanshi *(Corin et al., 2017; Kalichini et al*., 2017; Jacobs, 2016; Pacquot & Ngulube, 2015; Kalichini, 2015; Pérez-Barnuevo *et al*., 2013). The objective of this paper is to provide a broader perspective, analysing variation in mineralogy across the four circuits and relating this to the operating strategies and performance of each.

METHODOLOGY

The data presented in this paper are from routine weekly mineralogical analysis performed on-site using the Zeiss MinSCAN (Hill 2014). Automated sample cutters on the cyclone overflows, final tails and final concentrate streams are used to collect two-hourly samples for chemical analysis. From these samples, daily composites are prepared, which are in turn combined to form weekly composites. The plant sampling equipment and sample preparation procedures are continuously being improved with the aim of ensuring full representativeness. This is a work in progress.

After being wet screened into four fractions, the weekly composite samples (4 circuits by 3 streams by 4 fractions), are further split using a micro-riffler, mixed with graphite and then mounted in resin. After setting, these blocks are polished and carbon coated for scanning electron microscopy (SEM) analysis with the MinSCAN. The blocks are analysed using a random grid pattern of fields that are mapped with a 2–14 µm energy-dispersive spectroscopy (EDS) point spacing depending on the size fraction. Mineral classification, performed using the *Mineralogic* software, is based on a mineral list that was developed by iMin Solutions. Optical microscopy is used intermittently for corroboration of SEM mineral identification (Figure 1), and this has helped to identify and resolve occasional issues with SEM mineral classification.

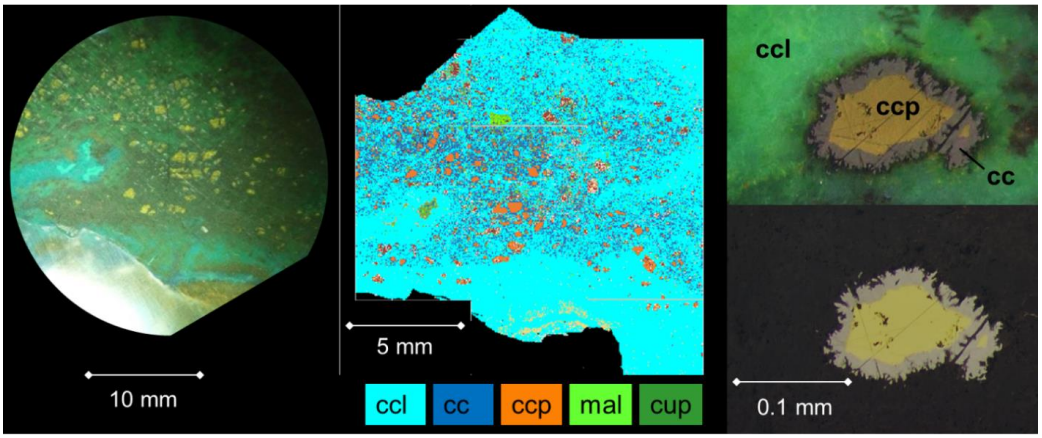

Figure 1: Photomicrographs and a MinSCAN false colour map of a hand-sample (rock) taken from the Oxide SAG mill discharge grate. The major component of the rock is chrysocolla (ccl), but it hosts disseminated chalcopyrite (ccp) grains that have partially been replaced by chalcocite (cc). Finer sulphide grains have undergone complete replacement to secondary copper sulphides, whereas coarser grains retain the chalcopyrite core.

Calculations for copper distribution by-mineral-by-size incorporate total copper assays from the Kansanshi chemical laboratory for weighting of the copper deportment to each size fraction. This reduces the errors associated with particle segregation during block preparation. In November 2017, the analysis method for tails samples was changed from X-ray mapping to a bright phase search to increase the number of copper mineral particles analysed and thus reduce the uncertainty associated with losses of sulphide minerals, native copper, malachite and cuprite.

Data from the period December 2017 to May 2018 (22 weeks) were selected for description of the bulk mineralogy, copper deportment, copper mineral recovery and mineral associations across the four circuits. This was due to slight modifications in the mineral list clustering and improvements in the analysis set-up that were implemented in November 2017. However, data for 2016 and 2017 describe similar mineral behaviour across the four circuits to the selected period data.

BULK MINERALOGY

Figure 2 shows the bulk mineralogy of the ore fed to the three flotation circuits, as well as that of the flotation concentrates obtained. The major non-sulphide gangue minerals are quartz, feldspar (albite), mica (biotite, muscovite), and calcite, which together comprise over 80 mass% of the ore fed to the three circuits. Dolomite, clay (kaolinite), iron oxides (magnetite, haematite and goethite) and others (rutile, ilmenite and zircon) comprise another 10–20 mass%. Note that graphite is not shown in Figure 2 as it is not directly quantifiable with the EDS method, but minor quantities are expected to be present, and at times "carbonaceous material" does cause problems in the flotation circuits. Additionally, the mica cluster currently includes amphibole and chlorite minerals that are present in minor amounts. Pyrite, pyrrhotite and other sulphides (mostly galena) are all minor components of the feeds, with pyrite having the biggest impact on concentrate quality, accounting for 10–20% of the final concentrate mass.

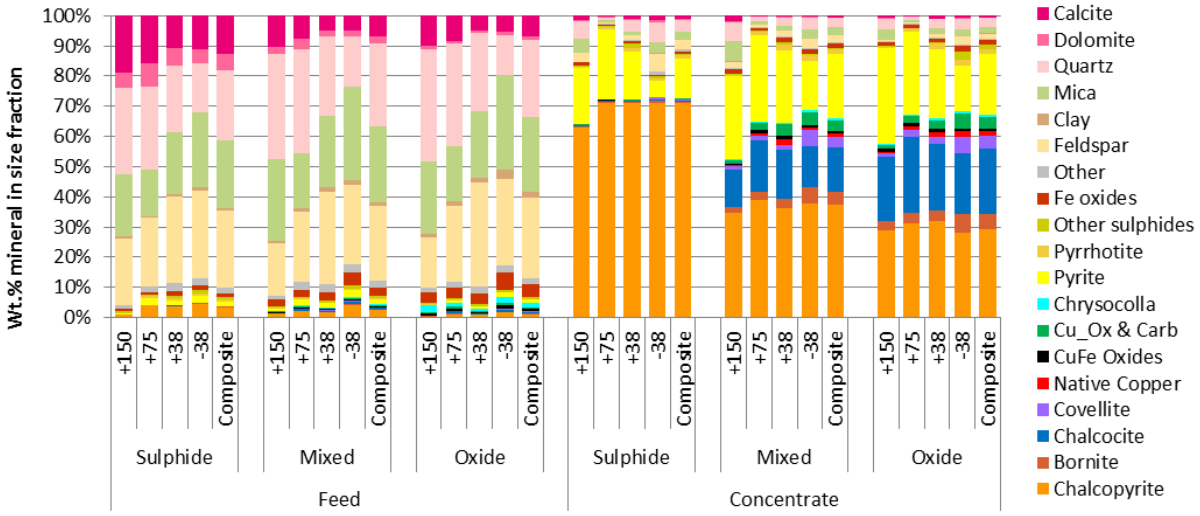

Figure 2: Bulk mineralogy by size for feed and concentrate samples of the three circuits (average of 22 weeks).

The bulk mineralogy by size for each circuit shows clear variation in the breakage rates and grain sizes between the major gangue minerals, with quartz and calcite preferentially deporting to the coarse fractions. In the deposit, a substantial proportion of the copper and gold mineralisation is hosted within the network of quartz and calcite veins that crisscross the deposit. The chalcopyrite mineralisation within these veins is mostly coarse, so it is easily liberated and preferentially recirculated within the grinding

circuits, and therefore it does not follow the same distribution by size as the quartz and calcite. Across the three flotation circuits, the proportions of the gangue acid-consuming minerals (calcite and dolomite), decrease from Sulphide to Mixed to Oxide. The relative abundance of these minerals and variation thereof is of key interest for the leach circuit and has at times been used as a means of controlling the acid balance between the smelter and the leach circuit.

The extent of oxidation and weathering across the three flotation circuits is apparent in the ratio of sulphide to oxide minerals in the feeds, with increasing proportions of iron oxides, secondary copper minerals and chrysocolla. The main differences between the concentrates across the three circuits is apparent in the copper sulphide minerals, where supergene enrichment has led to replacement of chalcopyrite with secondary copper sulphides such as bornite, chalcocite and covellite in the Mixed and Oxide ores (Figure 1). Djurleite and digenite have also been identified (Jacobs, 2016), but these have been clustered with chalcocite. The similarity in bulk mineralogy of the Mixed and Oxide concentrates does not fully reflect the differences in copper mineralogy between these two ores because the flotation recovery of chrysocolla is negligible. The difference in chrysocolla content is more clearly demonstrated in the following sections.

COPPER DEPORTMENT

The distribution of copper to different mineral hosts was originally identified as one of the key mineralogical variables to track at Kansanshi, and this has been the most useful mineralogical measurement for explaining variation in grade and recovery over time. Figure 3 shows how the copper in the feed to each circuit is recovered to the flotation concentrate or lost to the flotation tails depending on the copper deportment to different minerals. Figure 3 also describes the leaching of copper from various minerals in two leach trains through comparison of the copper distribution in the leach feed to that in the solids residues discharged from the two leach trains. It is predominantly the copper sulphide minerals that are recovered to the concentrates, and chrysocolla and copper oxides and carbonates that are leached.

Of the three flotation circuits, the Sulphide circuit has the highest recovery (>90%) due to the simple copper mineralization, with ~90% of the copper occurring in chalcopyrite. In the Mixed ore, chalcopyrite hosts ~35% of the copper in the feed, and in the Oxide ore this proportion is normally less than 20%. In the Oxide circuit, the flotation recovery is less than 50%, as ~50% of the copper is contained in chrysocolla, cuprite and malachite which report to the Oxide tails. Fortunately, these minerals leach well, so it is the losses of chalcopyrite that is most detrimental to the combined float and leach recoveries of the Oxide and Mixed circuits. In the Mixed circuit, controlled potential sulphidisation (CPS) with addition of sodium hydrogen sulphide (NaHS) is used to improve flotation recovery of the secondary copper sulphide, copper oxide and copper carbonate minerals. NaHS is also added to the Oxide circuit and occasionally to the Sulphide circuit for a similar purpose, but dosages are not currently controlled by CPS. Projects supported by mineralogy are underway to optimise NaHS dosages using CPS for both the Oxide and Sulphide circuits.

Historically, little or none of the Mixed float tails was sent to the leach circuit, so more effort was made to recover the copper oxide and carbonate minerals by flotation in the Mixed circuit, with multiple CPS stages and regular trials for new reagents designed to improve recovery of these minerals (Pacquot & Ngulube, 2015; Corin *et al.*, 2017). Recently, due to the abundance of acid available from the Smelter and quantitative mineralogy describing the losses in the Mixed flotation tails, the leach circuit has been reconfigured to increase capacity and allow for processing of up to one hundred percent of the Mixed flotation tails.

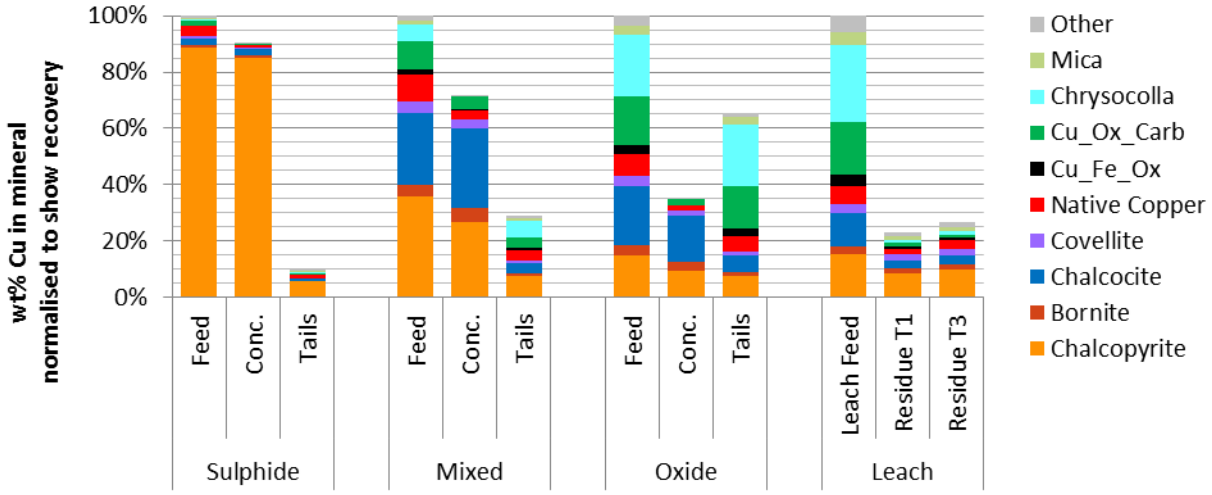

Figure 3: Normalised copper distribution by mineral for the key streams of each circuit, weighted by total copper in each stream relative to the total copper in the feed to the circuit. Residues T1 & T3 describe the final counter-current decantation thickener underflow streams of the two leach trains used in the current circuit configuration.

The mineralisation that leads to difficulty in achieving high flotation recoveries in the Mixed and Oxide circuits has a positive effect on concentrate grade. For the Sulphide circuit, to achieve a target grade of 24% total copper, the concentrate has to contain 70% chalcopyrite by mass as chalcopyrite has a relatively low copper content (34.8%). On the other hand, chalcocite has a copper content of ~80%, so for the Mixed and Oxide concentrates it is possible to achieve grades of over 40%. For this reason, the operating strategy for the Sulphide circuit has a greater emphasis on pyrite and non-sulphide gangue rejection than the other circuits, with a chalcopyrite selective collector and Jameson cells used for cleaning the final concentrate (wash water reduces entrainment). The effects of these are evident in the bulk mineralogy of the concentrate (Figure 2), which shows more copper minerals and less pyrite in the Sulphide concentrate than in the Mixed and Oxide concentrates, while the copper grades of the three concentrates are 22%, 28% and 30%, respectively (average year to date).

COPPER MINERAL RECOVERIES

The ability to calculate recoveries of individual copper minerals within a circuit provides a valuable tool for assessing operating factors influencing both flotation and leach performance, as well as for understanding how ore variability affects performance. The recoveries and leach efficiencies of the major copper hosting minerals are presented in Table I and the copper mineral recoveries by size are presented in Figure 4. It is apparent that it is not only the distribution of copper to different minerals in the three circuits that influences recovery, but recovery of each mineral also varies across the different circuits. The recovery of chalcopyrite is highest in the sulphide circuit (94%), moderate in the Mixed circuit (78%), and poor in the oxide circuit (57%). This variation is likely a result of differences in the surface chemistry of the chalcopyrite grains, which may be influenced by multiple variables. Key factors identified that could influence the chalcopyrite grain surfaces include the stream's bulk mineralogy (affecting iron and copper species in solution), the grinding environment (different quality steel media used), and the pulp potential (modified by addition of NaHS) (Heyes & Trahar 1979; Bruckard et al. 2011; Chen et al. 2014; Greet et al. 2004; Whiteman et al. 2016; Lotter et al. 2016).

Table I: Copper mineral recoveries across the three flotation circuits and two leach trains.

	Flotation recovery (%)			Leach efficiency (%)	
	Sulphide	Mixed	Oxide	Train 1	Train 3
Chalcopyrite	94	78	57	37	34
Bornite	84	86	70	43	40
Chalcocite	69	89	75	69	65
Covellite	71	76	62	33	24
Native Copper	40	63	37	67	63
Copper oxides & carbonates	43	55	14	93	94
Delafossite & Cu Fe oxides	50	32	10	75	76
Chrysocolla	7	6	1	94	94

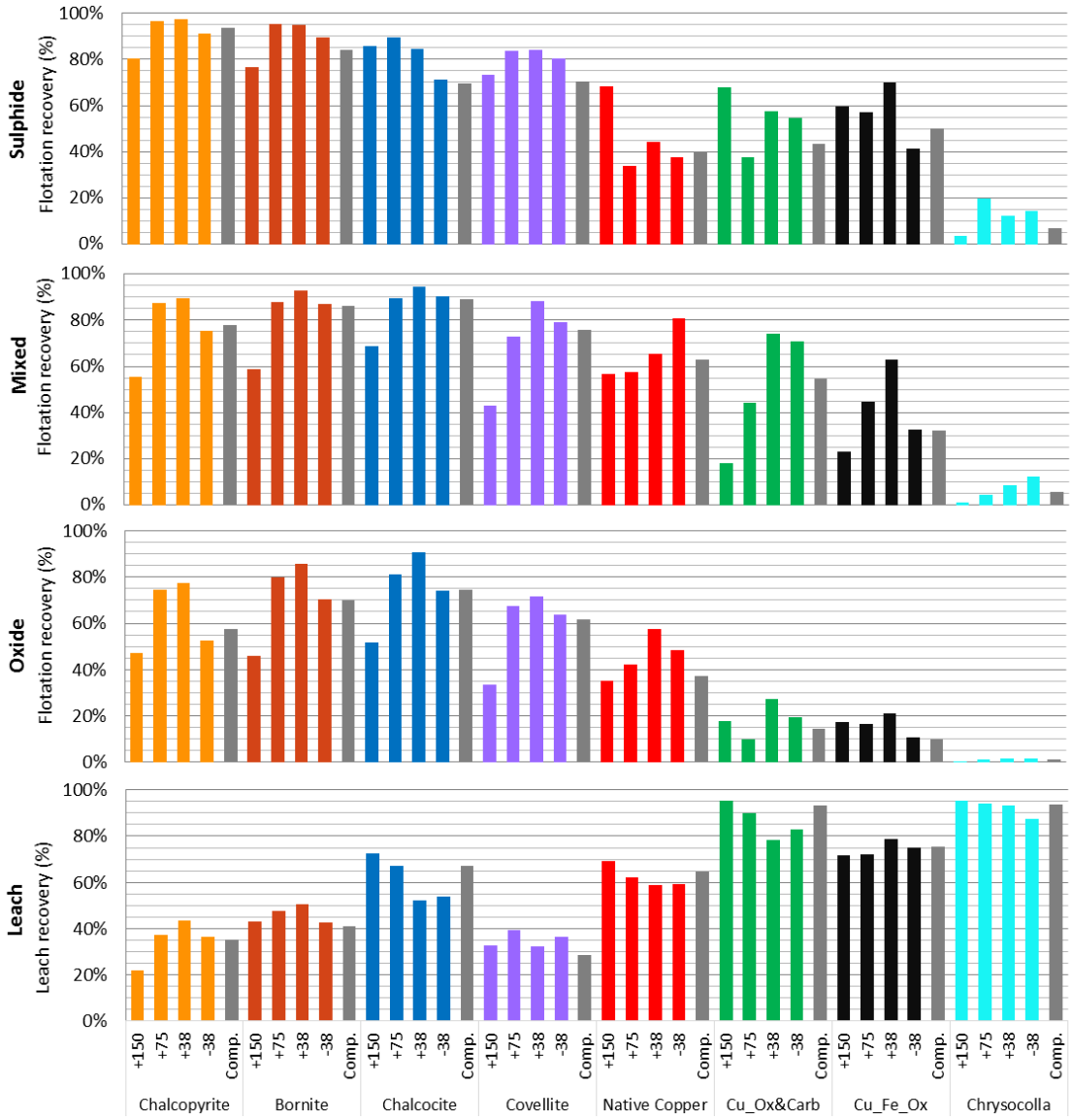

Figure 4: Recovery of copper minerals by size for each circuit. Comp. = composite.

Of the secondary copper sulphides, bornite has the highest recoveries in the sulphide circuit with similar behaviour to chalcopyrite, whereas chalcocite has the highest recoveries in the Mixed and Oxide circuits. This is consistent with observations in the literature that the Eh ranges for optimal flotation of chalcopyrite and chalcocite differ due to the difference in semi-conductor types, with chalcocite floating better at more negative Eh (Lotter *et al*., 2016). Additional CPS stages are likely responsible for higher recoveries of chalcocite, native copper, and copper oxides and carbonates in the Mixed circuit (Pacquot & Ngulube, 2015), and this reasoning has motivated CPS projects for the Oxide and Sulphide circuit to further improve flotation recoveries. Variation in the different Cu mineral head grades across the three circuits could also account for some of the variation in mineral recoveries across circuits, with the Sulphide ore having the highest chalcopyrite head grade and very low head grades for the other copper minerals. The YTD chalcocite head grades for the Sulphide, Mixed and Oxide circuits are 0.01%, 0.28% and 0.33%, which perhaps describes the difference in chalcocite recovery between Sulphide and Mixed, but not between Mixed and Oxide.

The effects of particle size on flotation performance have been well documented (Feng & Aldrich, 1999), and the traditional "optimal particle size range for flotation" with a P_{80} of 150 µm has been targeted since the plant was commissioned. The size-by-size recovery data (Figure 4) seem to justify this approach, with flotation recoveries of the sulphide minerals higher for the −150/+75 µm and −75/+38 µm fractions than for the +150 µm and −38 µm fractions. However, in Figure 5 it is evident that losses in the −38 µm fraction are typically greater than those in the other size fractions, indicating that perhaps a coarser and steeper particle size distribution would be preferable.

Many projects have been implemented to reduce the fines losses. In the Sulphide circuit, Jameson cells (Glencore Technology, 2017) were installed in August 2017, and ProFlote (paramagnetism to agglomerate fines) is currently being trialed (FLSmidth, 2018). To reduce fines generation in the Mixed circuit, the discharge mechanism of the ball mill was converted from grate to overflow in mid-2017, which led to a substantial increase in recovery. High shear stators have also been trialed for improving fines recovery.

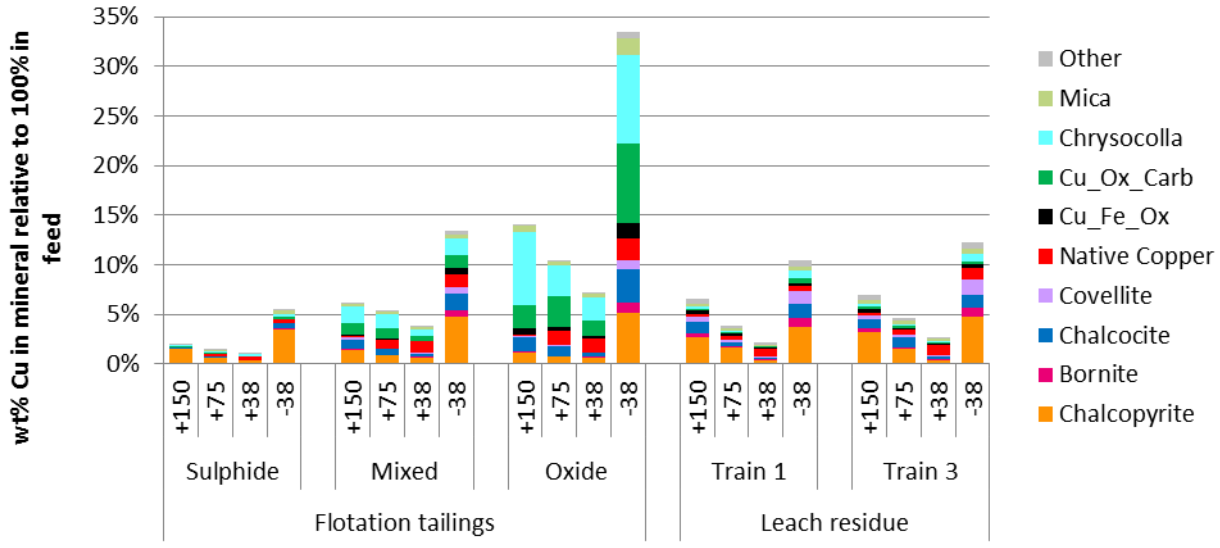

Figure 5: Normalised copper losses by mineral and by size as a percentage of the total copper fed to each circuit.

MINERAL ASSOCIATIONS

Apart from mineral abundance and copper distribution, understanding the liberation and associations of key minerals is important for explaining mineral behaviour in processing and identifying opportunities for improvements (Evans & Morrison, 2016; Fandrich *et al*., 2007; Albijanic *et al*., 2014; Whiteman *et al*., 2016). A commonly occurring association texture, that of rimming of chalcopyrite with chalcocite, is shown in Figure 6 (Pérez-Barnuevo *et al*., 2013; Jacobs, 2016). The associations of chalcopyrite, chalcocite, pyrite and quartz with other minerals in the feed and flotation concentrate streams of each circuit are quantified in Figure 7. In each graph, the colour of the selected mineral is used to indicate the associations of that mineral with "background," which is an indication of the extent of liberation. By this measure, of the four minerals, chalcopyrite and quartz have the greatest extent of liberation. As expected, in all of the feed streams liberation increases with decreasing particle size.

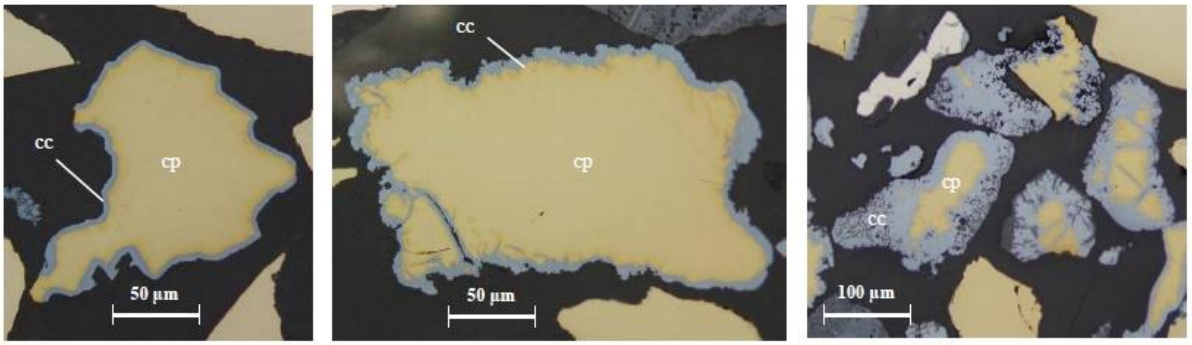

Figure 6: Reflected light photomicrographs of chalcocite (cc) rimming chalcopyrite (cp) particles on the Mixed ore (Kottgen & Bastin 2009).

Figure 7 shows that chalcopyrite and pyrite have significant associations with quartz, mica and feldspar in the coarse fractions of the feeds, and it is clear that these associations have a negative impact on recovery as the concentrates indicate much lower associations with these non-sulphide gangue minerals. Chalcocite has negligible associations with gangue minerals and is almost entirely associated with other copper minerals, predominantly covellite, bornite and chalcopyrite. In the Sulphide circuit, chalcocite associated with bornite and chalcopyrite is more likely to be recovered than liberated chalcocite, while liberated chalcocite appears to have the highest recoveries in the Mixed and Oxide circuits. Associations of chalcocite with covellite, chrysocolla, cuprite and malachite (Cu Ox & Carb) seem to negatively impact chalcocite recovery.

A large proportion of pyrite recovered to the concentrates is liberated, and the remainder is associated with pyrrhotite, chalcopyrite or mica. Similarly, a large proportion of the quartz recovered is either liberated or associated with other non-sulphide gangue minerals, which implies recovery by entrainment. Nevertheless, association of quartz with copper minerals and pyrite does account for a significant proportion of quartz recovered in the coarse size fractions.

Across the three circuits, the main differences observed are in the associations of chalcopyrite and quartz with secondary copper sulphides, which is due to weathering and supergene enrichment. It was noted in the previous section that chalcopyrite has lower recoveries in the Oxide and Mixed circuits than in the Sulphide circuit. Figure 7 suggests that in the Oxide and Mixed circuits, chalcopyrite has slightly poorer liberation and more associations with other minerals than in the Sulphide circuit, which likely contributes to the lower chalcopyrite recovery, but does not fully explain the difference observed.

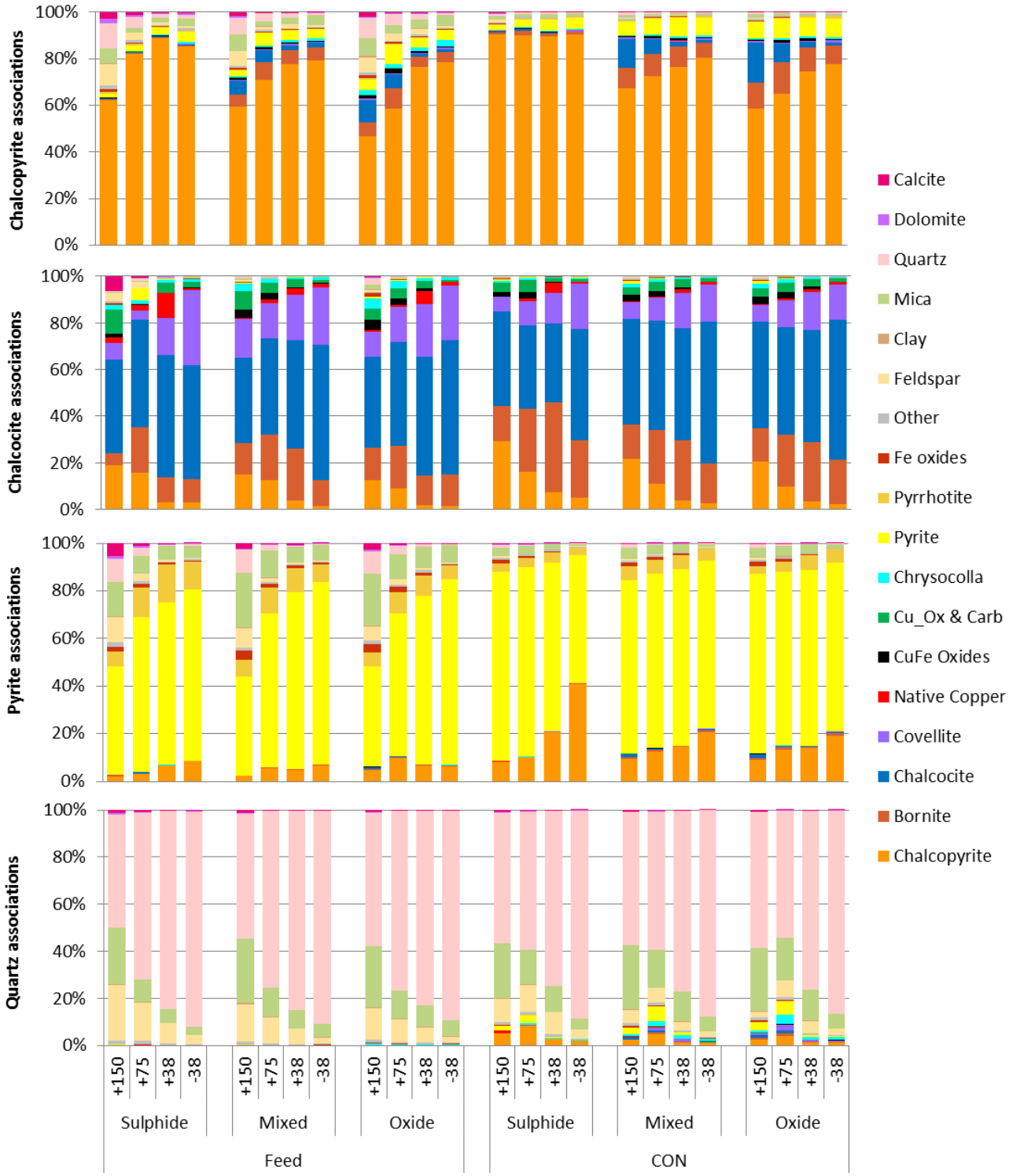

Figure 7: Associations distributions of chalcopyrite, chalcocite, pyrite and quartz with other minerals in the feed and flotation concentrate of each circuit. The colour for the selected mineral is used to show the association of that mineral with "background", which is an indication of extent of liberation.

MINERALOGY USED TO ASSESS PLANT PERFORMANCE

The average recoveries of each copper mineral for the year 2016 were used to calculate predicted recoveries for each circuit based on variation in feed copper deportment for 2017. These mineralogy-predicted recoveries are compared to measured recoveries in Figure 8, which shows recovery versus mineralogy-predicted recovery for the four circuits in 2017. This graph (going forward) provides a useful tool to identify periods of high and low recovery that can be explained by variation in feed mineralogy, thus improving interpretation of the impacts of operation and circuit changes. Figure 8 also illustrates that the variability of feed mineralogy is much greater for the Mixed and Oxide ores than it is for the hypogene Sulphide ore, and this, in turn, leads to high variability in flotation recovery as compared to the Sulphide circuit.

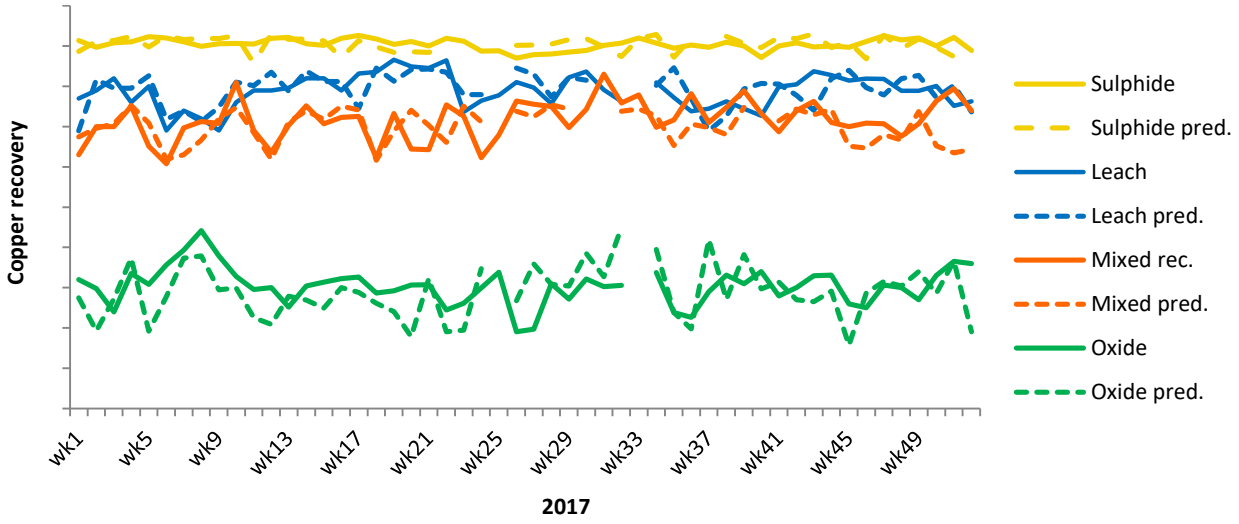

Figure 8: Total copper recovery vs. mineralogy-predicted-recovery for each circuit in 2017

CONCLUSIONS

The relative abundance of copper minerals within the ores fed to each of the processing circuits at Kansanshi has a strong impact on process performance. The fresh, hypogene Sulphide ore with ~90% of the contained copper hosted in chalcopyrite has consistently high flotation recoveries with minimal variation in flotation performance. In the Oxide and Mixed circuits, the copper mineralogy is much more variable depending on the extent of oxidation and supergene enrichment, which causes large fluctuations in both copper recovery and flotation concentrate grade.

Quantifying mineralogy using automated SEM provides an understanding of when changes in process performance can be explained by changes in mineralogy. At Kansanshi, it has delivered significant value through highlighting opportunities for grade and recovery improvement projects that have included reagent suite optimisation and circuit modifications. Key circuit modifications that have been guided by process mineralogy have included the change of the Mixed ball mill discharge mechanism, the installation of Jameson cells, and CPS for the Sulphide circuit (still in progress).

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