

# Separation efficiency improvement of a low grade copper-gold flotation circuit

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## Abstract

Newcrest's Telfer gold mine, located in the South Western area of the Great Sandy Desert in the Paterson Province of Western Australia, processes low grade copper-gold ores from open pit and underground sources in two parallel processing trains. The copper flotation circuits upgrade copper content by a factor of 80-160 times from feed to a saleable copper concentrate containing approximately 50% of the contained gold in feed (remaining gold is recovered both as a gravity product within the primary grinding circuit and also via leaching of a gold-containing pyrite concentrate).

The copper concentrate grades achieved from the flotation circuit at Telfer have typically fallen well short of the theoretical achievable grades despite the majority of copper being hosted in high copper bearing minerals: chalcopyrite, chalcocite, bornite and others. The recovery of gold-containing pyrite to the copper concentrate is encouraged to achieve a higher total gold recovery provided that a saleable copper concentrate grade is still produced.

Mineralogical diagnosis of feed and product streams from the copper flotation circuit have shown that the copper bearing minerals are well liberated and thus further liberation of the copper minerals was not selected as a method of improving Telfer's concentrate grade or recovery from current main dome ore sources. Furthermore, repeated mineralogical analyses have demonstrated that there is significant dilution in the copper concentrate resulting from the presence of fine, well liberated non-sulphide gangue minerals.

This paper presents a diagnosis of the flotation circuit that resulted in a proposal for a circuit modification that afterwards was indeed implemented, aiming at improving the existing flotation circuit with respect to the rejection of non-sulphide gangue, which is achieved by improved separation efficiency of the copper flotation circuit. The techniques used for diagnosis include: quantitative mineralogical evaluation of key streams, flotation circuit surveys, floatability component model fitting and simulation, and the application of dilute batch flotation tests.

## INTRODUCTION

Telfer is a gold/copper operation located in the Pilbara region of Western Australia. Open pit mining (Main Dome) recommenced in 2003, followed by an underground mine (Telfer Deep) in mid-2006. Ore mineralogy varies significantly from mainly chalcocite in the open pit ore to predominately chalcopyrite in the underground ore. The ore processing plant consists of two parallel trains, currently treating a total of 21 million tonnes of ore per annum including approximately six million tonnes from the underground mine. Train 1 is receiving a blend of the underground and open pit ores, while Train 2 is treating open pit ore alone. Details of the mine geological and ore mineralogical information, the initial process plant design criteria and operating strategies and a summary of the commissioning phase, can be found in previous publications by Goulsbra et al. (2003) and Benson et al. (2007).

Ore is processed through both trains in a variety of configurations. The predominant configuration is sequential flotation, where copper bearing minerals are recovered to a saleable copper concentrate, followed by re-activation and flotation of the pyrite which is leached with cyanide to recover gold.

Both trains produce a copper concentrate with significant quantities of non-sulphide gangue minerals. The target concentrate grade is approximately 16% Cu, with copper minerals, chalcopyrite and chalcocite, being the most abundant copper bearing minerals in the feed to the plant. Stoichiometrically, these minerals contain 34.6 and 79.9% Cu respectively (up to five times the copper grade produced at Telfer).

A reduction in the non-sulphide gangue content of the concentrate will result in an improvement of the copper concentrate grade and of the ability of the plant to recover additional pyrite (containing gold) to the copper concentrate, thereby improving gold recovery from the operation. Zheng, Crawford & Manton (2009) presented details on how a reconfiguration of Train 1 was completed to assist rejection of some of this gangue. While this modification was successful, further improvements are still possible.

This paper presents diagnostic investigations carried out to determine the quantity and nature of this non-sulphide gangue reporting to the copper concentrate. Subsequently, a modification to the cleaning circuit is proposed, that aims at rejecting a significant quantity of the non-sulphide gangue.

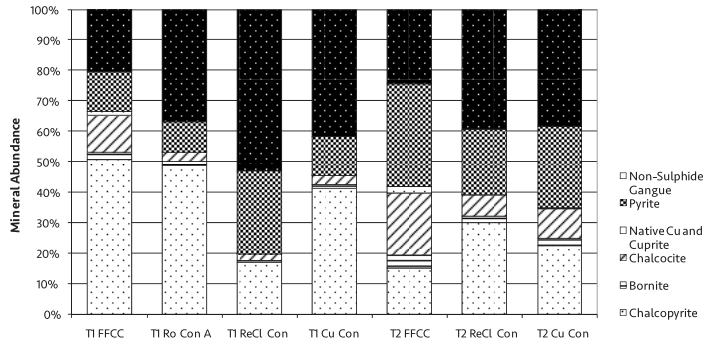
## MINERALOGY OF COPPER CONCENTRATE

Since processing recommenced at Telfer, there have been several size and mineralogical quantifications of monthly composites from both operating trains. **Figure 1** below shows a summary of the mineral abundance in the copper concentrate streams from all trains. These data are taken from a recent (October 2010) quantification that was completed by G&T Metallurgical Services Ltd. (Ma & Johnston, 2011). It is evident in the mineral abundance that approximately 40% (on a mass basis) of the copper concentrate produced is made up of non-sulphide gangue. Although not presented here, similar mineral abundances have been found in other mineralogical campaigns carried out over the last two years at Telfer.

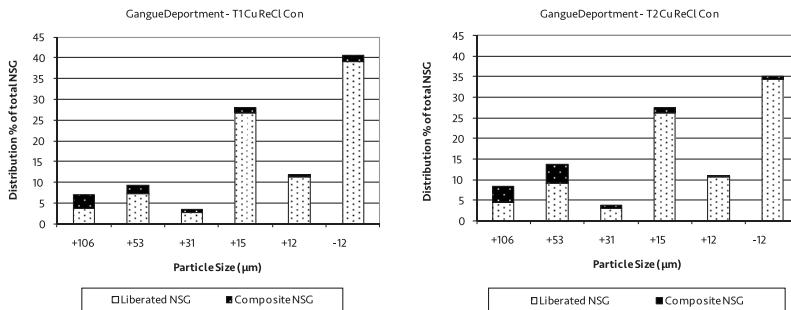
The greatest source of gangue in the copper concentrate is the re-cleaner concentrate streams and the Train 1 rougher concentrate A.

**Figure 2** shows the size distribution and liberation characteristics of the non-sulphide

gangue contained in the two re-cleaner streams. Note that liberated gangue is defined as those particles containing more than 98% of the gangue surface area (measured in 2D) as measured by G&T in a QEMSEM (Ma & Johnston, 2011). Firstly, it is evident that the non-sulphide gangue present in the copper recleaner concentrate streams is very well liberated in most size fractions. Secondly, the gangue is finely distributed with 80% of the total gangue from these streams containing particles of less than 30  $\mu\text{m}$  in size.



**Figure 1** Mineral abundance in concentrate streams from the Telfer plant in October 2010<sup>1</sup>



**Figure 2** Non-sulphide gangue liberation and size distribution, October 2010 (Ma & Johnston, 2011)

Analysis of the mineralogical data reveals that theoretically, a final concentrate assaying about 20 to 25% copper could be achieved in both trains by rejecting half of the liberated gangue currently reporting to final concentrate. This theoretical improvement would be equivalent to an improvement in overall copper and gold recovery by moving further down the grade-recovery curve of the operation.

The mineralogical deportment and size of the gangue presented in these two graphs suggest that the non-sulphide gangue is being recovered to the flotation concentrate by the mechanism of entrainment. The entrainment process is size dependant, with more recovery by entrainment occurring at finer particle sizes due to their lower settling velocities in flotation froth. Johnson, McKee & Lynch (1975) and Savassi et al. (1998)

1 In this figure, FFCC refers to the flash flotation cleaner concentrate; Ro Con A refers to the first two rougher cell concentrates which are currently sent to final concentrate – as described in Zheng, Crawford & Manton (2009); ReCl Con refers to the re-cleaner concentrate; Cu Con refers to the final combined copper concentrate from each respective train. T1 and T2 refer to the two operating trains – Train 1 and Train 2.

provide further details on the entrainment mechanism. A common expression used to describe the entrainment process is the use of an entrainment classification function,  $ENT_i$ , which is dependant on the size distribution of the particles in the slurry phase:

$$M_{ent,i} = ENT_i \cdot R_w \cdot M_{tail,i} \quad (1)$$

Where,  $ENT_i$  is the entrainment classification number for particles of size class  $i$ ;  $M_{ent,i}$  is the mass flow rate of particles in the  $i$  size class collected in the concentrate by the entrainment mechanism;  $M_{tail,i}$  is the mass flow rate of particles in the  $i$  size class in the tailings stream; and  $R_w$  is the water recovery of the flotation cell. Values of  $ENT_i$  for different size classes can be found in Johnson, McKee & Lynch (1974) or these can be determined from a non-floating fully liberate tracer mineral in the ore body.  $ENT_i$  typically ranges from zero for particles greater than 53  $\mu\text{m}$  and approaches a value of one for ultra-fine particles.

Once the ore has been ground and a size distribution of particles is presented to a flotation cell, the only parameter that can be manipulated to reduce the entrainment is water recovery. If additional water is added to dilute the contents of the flotation cell, and the flow rate of water to the concentrate remains the same, the water recovery will be reduced. The downside of diluting flotation banks is the resulting loss of residence time which can have a negative effect on valuable mineral recovery.

## FLOTATION CIRCUIT DIAGNOSTICS

Surveys of each flotation train were conducted to investigate the potential for diluting the cleaner circuits further, with the objective to assist in entrainment rejection and to provide a circuit balance. This balance is used to build a flotation model that is used to investigate further configuration changes to the circuit. Dilution cleaning tests were also carried out with the aim of confirming whether the non-sulphide gangue in the concentrate streams was being recovered by entrainment or was hydrophobic and therefore recovered by true flotation.

### Mass balance results

A survey and mass balance was completed on each flotation train. The %solids (w/w) and calculated residence times of each flotation bank are shown for both trains in the table below.

**Table 1** Estimated densities and residence times of flotation banks

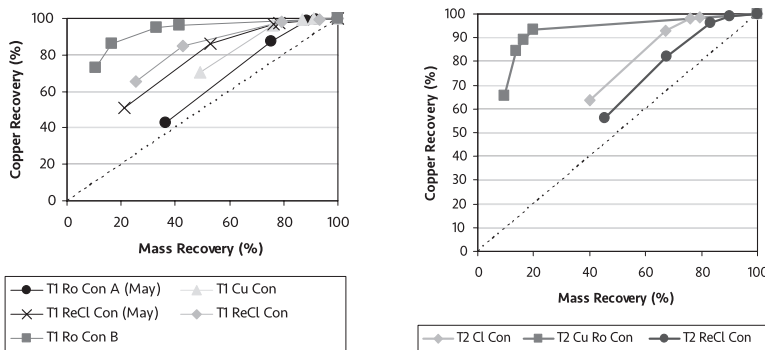
	Train 1		Train 2	
	%solids	$\tau$ (min)	%solids	$\tau$ (min)
Cu Ro	22.0	10.2	34.3	15.0
Cu Cl	13.0	10.9	10.5	13.6
Cu Cl Scav	9.0	14.9	9.2	14.4
Cu ReCl	13.4	17.2	10.1	20.7

As can be seen in **Table 1**, the cleaner and recleaner banks are being operated with sufficient residence time to achieve flotation exhaustion of floatable minerals. In order to improve cleaning efficiency by further dilution of the cleaner and recleaner banks, additional flotation capacity would be required in the cleaning circuit.

### Dilute cleaning batch flotation tests

Plant samples were collected, diluted and floated immediately to assess the NSG rejection potential of these streams. In all tests, a pulp density in the batch flotation test of ~5% was targeted to minimise the amount of non-sulphide gangue (NSG) reporting to the flotation concentrates by entrainment. Raw water was used as make-up dilution water for these tests, and only plant frother (DSF004) was added as necessary to maintain the froth in the batch flotation cell.

**Figure 3** below shows the resulting copper-mass selectivity of these dilute batch flotation tests conducted on various concentrate streams around the plant.



**Figure 3** Copper recovery as a function of mass recovery - dilution cleaning tests

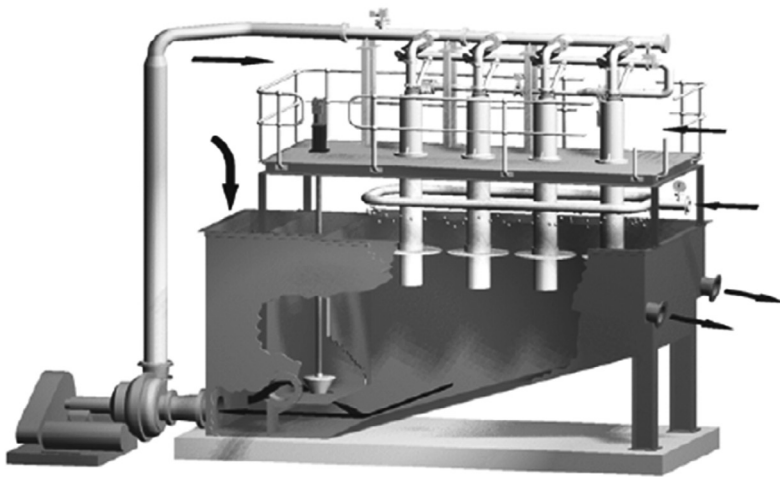
It is evident that with all streams re-floated, there is a significant upgrading achieved with an additional dilute cleaning stage, and in all cases, high copper recoveries were achieved after four concentrates with a cumulative flotation time of 9-10 minutes used in each test. For all the streams tested, it was possible to reject greater than 20% of the mass of the concentrates while recovering 90-99% of the valuable minerals (copper and gold). It can be concluded from these tests that the non-sulphide gangue present in the concentrate streams is predominantly non-floating and is being recovered by entrainment. Sulphur recovery was higher, similar to the copper result shown above, confirming that the improved selectivity was a result of non-sulphide gangue rejection.

### FLOTATION PLANT RECONFIGURATION

Several alternative configurations were explored to improve the gangue rejection of the cleaning circuits at Telfer. As this is a Brownfield application, practical issues related to the installation of additional cleaning capacity and reconfiguration of the existing equipment were important in selecting the most effective improvements to the cleaner circuit performance at Telfer.

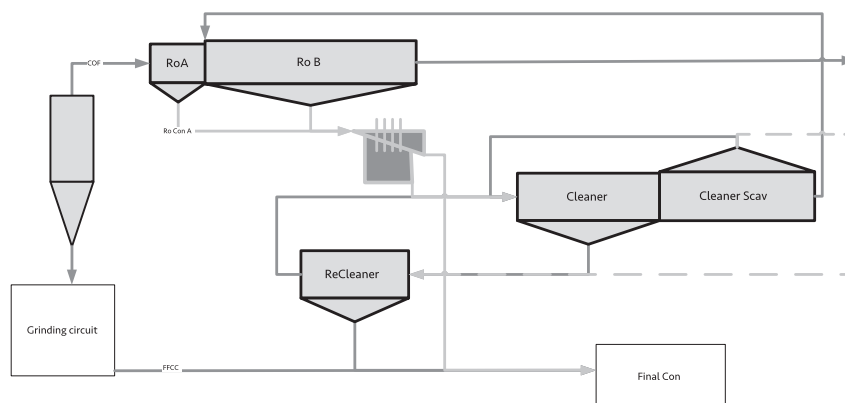
Barns, Colbert & Munro (2009) demonstrated the application of a Jameson cell at Prominent Hill, where a Jameson cell was installed as a 'cleaner scalper' unit, in order to reduce the load on the mechanical cleaning circuit and improve fine grained fluorine entrainment.

Jameson cells described by Evans, Atkinson & Jameson (1995) are highly efficient flotation machines that require a smaller footprint than conventional mechanical flotation cells and enable the efficient use of froth washing to improve gangue rejection. A schematic figure of the latest Jameson cell technology presented by Young et al. (2006) is shown below. Typically these cells achieve more than 50% recovery of valuable mineral per stage, with negligible gangue entrainment due to the froth wash water that is operated to achieve a net downward water flow in the froth.



**Figure 4** The Jameson cell with internal recycle to stabilise downcomer feed rate, after Young et al. (2006)

The proposed circuit reconfiguration for improving NSG rejection at Telfer is shown in **Figure 5** – a Jameson cell is to be installed as a scalper unit ahead of the cleaning circuit, targeting the removal of 50% of the valuable (i.e., copper and gold containing) minerals at high concentrate grades (through effective froth washing and high intensity bubble-particle contact). The cleaner-scavenger concentrate is also to be redirected to the re-cleaner, allowing the cleaner bank to be operated at a higher dilution (5–10% solids w/w) without compromising cleaner bank residence time. In the case of Train 1, the first two rougher cell concentrates are to be redirected through the new Jameson cells to assist in gangue rejection of this stream. In Train 2, all the copper rougher concentrate currently reports to the cleaning circuit.



**Figure 5** Jameson cell configurations with concentrate reporting to final concentrate

In order to estimate the potential improvement in the operating grade-recovery curve of the Telfer operation, a flotation model was calibrated with recent plant survey data, and used to predict the performance of the modified circuit shown above.

## Model

A flotation model was developed for both trains based on the AMIRA P9 flotation modelling methodology as presented by Harris et al. (2002). In this methodology the feed is split into a number of floatability fractions that describe the observed distributed flotation rate. The recovery of a single floatability fraction across a single perfectly mixed cell is described by:

$$R = \frac{P \cdot S \cdot Rf \cdot \tau(1 - R_c) + ENT \cdot R}{1 + P \cdot S \cdot Rf \cdot \tau(1 - R_c) + ENT \cdot R} \quad (2)$$

where  $R$  is the recovery of fraction,  $P$ , the floatability of the fraction,  $S_b$ , the superficial gas velocity of the cell,  $Rf$ , the froth recovery of the floatability fraction in the cell,  $ENT$ , entrainment parameter (size dependent) and  $Rw$ , the water recovery across the cell.

A total of three components were used to describe the flotation of each mineral species: fast floating, slow floating and non-floating.

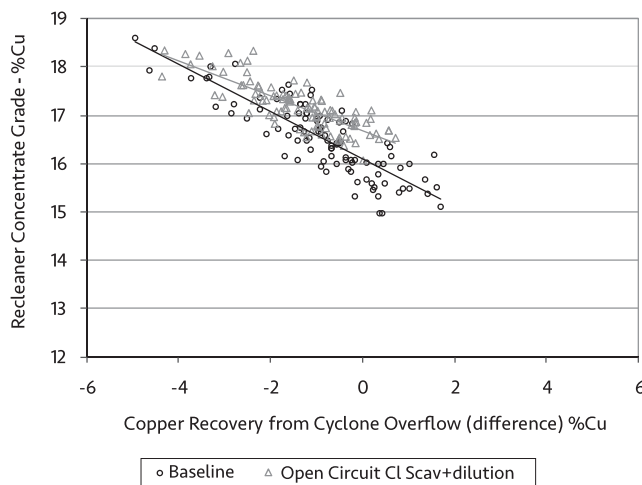
In order to simplify the model, the terms  $P^*S_b^*R_f$  were lumped into a single first order rate constant  $k^*C$  where  $k$  is a first order rate constant conserved around the entire Cu circuit (for each floatability class) and  $C$  being a scale-up number assigned to each physical cell or flotation bank. This  $C$  parameter is actually a measure of how hard the cell is pulled by changing froth depth or aeration rate. The model adequately describes the entrainment mechanism, critical to evaluating cleaning circuits through the use of the entrainment classification number,  $ENT$ , which was estimated based on the size distribution of the tailing streams from each bank.

The feed stream to each bank was split into three floatability fractions: fast, slow and non-floating fractions for each of the elements assayed for (Au, Cu, Fe and S), and model parameters were fitted to the plant data.

## Simulation results

Monte Carlo simulations were conducted of the as-is flotation circuit models and with the modifications shown in **Figure 5**. The Monte Carlo simulations were conducted by sampling the flotation cell scale-up parameters, randomly, from a normally distributed scale-up number with standard deviation of 10% and 20% for the rougher and cleaner/re-cleaner cells respectively, and simulating the resulting parameter set. This was repeated 100 times for each scenario that was modelled. This manipulation of the cell scale-up parameter is akin to operators changing the level and the air flow rate set-points randomly in the plant.

The impact of redirecting the cleaner scavenger concentrate from the cleaner feed to the recleaner feed and diluting the cleaner bank from 10% solids to 7% solids, without installation of the Jameson cell, is shown for Train 2 in the figure below (the copper recovery is shown as a change from current recovery).

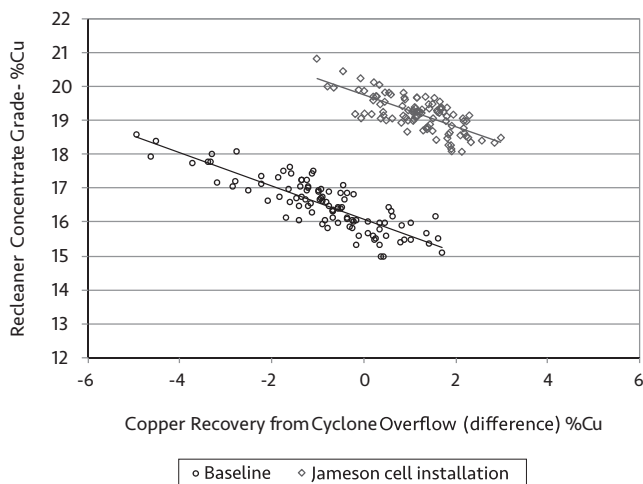


**Figure 6** Modelled effect of open-circuiting cleaner scavenger concentrate and dilution of the cleaner bank from 10% solids to 7% solids on Train 2 at Telfer

This modification shows some improvement in gangue rejection, however further benefit is possible combining this change with the additional installation of Jameson cells as shown in **Figure 5**.

The Jameson cells were modelled using the selectivity obtained in the dilute cleaning tests on the rougher concentrate streams, as shown in **Figure 3**, and assuming that at least 50% copper and gold recovery can be achieved in the Jameson cell. This recovery value was based on similar installations of Jameson cells in similar duties, such as Prominent Hill (Barns, Colbert & Munro, 2009) and Phu Kham (Young & Crnkovic, 2011). The results of the Monte Carlo simulation results are shown in **Figure 7**.





**Figure 7** Model results for Train 2 Jameson cell installation

It can be seen in **Figure 7** that the resulting improvement in copper grade and recovery is consistent with the mineralogical predictions that the concentrate grade could be raised to 20-25% Cu, by rejecting half of the liberated gangue species that are currently reporting to the final concentrate. Although not shown, gold grade and recovery showed improvements similar to those of copper.

## Implementation

On the basis of the mineralogical circuit analysis, dilute cleaning tests and modelling outputs, it was decided that piloting of Jameson cells was unnecessary and capital expenditure for the installation of one Jameson cell per train has been approved. The Jameson cells have been sized to treat a maximum feed rate of 60 tph (solids) at a solids content of 15% w/w and a feed grade of 3-7% Cu. The recycle of the cell has been designed at 50% and the Jameson Cell was designed with an eight downcomer configuration – E3432/8.

The cleaner scavenger concentrate will be re-routed to allow for the stream to be directed to cleaner feed or recleaner feed, and additional water addition and density control will be installed to allow for the optimisation of the cleaning circuit post Jameson cell installation.

Installation and commissioning of these cells are expected to be completed by the end of December 2011.

## CONCLUSIONS

It has been shown that there is a significant portion of non-sulphide gangue reporting to the final concentrate from both trains (~40%) at Telfer. In both trains, this non-sulphide gangue has been shown to be well liberated and contained in relative fine fractions suggesting its deportment to final concentrate by the mechanism of entrainment. This was confirmed by completing dilute batch flotation tests on various concentrate streams

where 80% mass rejection was achieved with minimal loss of valuable (copper and gold) species.

Flotation modelling was completed on both trains to simulate the benefit of operating the cleaning circuit at a lower density – by diverting the cleaner scavenger concentrate to the recleaner to avoid loss of residence time in the cleaning circuit – and with the installation of a Jameson cell at the front of each cleaning circuit. The estimated benefit of this modification to the circuit is consistent with theoretical predictions that are based on rejection of half of the liberated gangue from the final concentrate producing a copper concentrate grade of 20 to 25% Cu.

The use of mineralogical circuit data, together with the application of a flotation model, allowed for the redesign of the cleaning circuit as well as demonstrating the need of capital expenditure required to implement the modification at Telfer.

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