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Operating strategies to maximise gold recovery at Telfer

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ABSTRACT

On average, the difference in recovery between copper and gold is 15% among the large scale copper/gold operations. At Telfer, a number of operating strategies have been implemented together with a sequential flotation circuit design to maximise gold recovery. The main operating strategies include targeting a primary grind size optimum for copper recovery, designing and operating the main flotation circuit as copper and pyrite sequential flotation, targeting a minimum saleable concentrate copper grade, allowing a portion of pyrite/gold recovered into the copper concentrate and leaching the pyrite concentrate to extract the gold. These operating strategies have lifted the recovery of both copper and gold above 90%. There are opportunities to further improve the metallurgical performance at Telfer, including a single stage of cleaning of the copper rougher concentrate, regrinding of the copper scavenger concentrate prior to cleaning and regrinding of the pyrite rougher concentrate followed by additional copper/gold flotation prior to pyrite leaching.

1. Introduction

A review on some of the large porphyry/skarn copper operations reveals that there is a relatively large gap in recovery between copper and gold, as shown in Table 1 and Fig. 1.

On average, the difference in recovery between copper and gold is 15%. Although this difference depends primarily on ore mineralogy especially gold deportment in the ore, other factors such as treatment method, flowsheet configuration and operating strategy may also have a strong influence. At Telfer, these factors are considered in the operating strategy to maximise gold recovery.

2. Telfer ore and circuit configuration

Telfer mine operates both underground and open pit mines (Goulsbra et al., 2003). The underground ore contains predominantly chalcopyrite with a relatively large quantity of pyrite while chalcocite is the main copper mineral in the open pit ore. Gold is mostly associated with copper minerals and pyrite. There are two parallel process trains in the ore treatment plant. More than half of the Train 1 feed comes from the underground mine with the remaining ore from the open pit mine. The underground ore has a higher grade and contributes over 80% of the copper and 70% of the gold in the Train 1 feed. Table 2 lists the main minerals in the Train 1 feed based on plant monthly composite samples (Brown and Ma, 2007).

In addition to flash flotation and gravity gold concentration in the grinding circuit, the main flotation circuit consists of two separate sections. Each section has its own rougher/scavenger and cleaner circuit. Depending on flotation mode, these two sections can be used to float copper minerals and pyrite in sequence or together (Benson et al., 2007). While the copper concentrate is the final product, pyrite concentrate when produced is further leached in the CIL circuit to extract gold. Fig. 2 is a schematic representation of the current Train 1 circuit flowsheet.

A more detailed description of the Telfer Train 1 flotation circuit can be found in a previous publication by Zheng et al. (2009).

3. General operating philosophy

Since the main economic value is derived from gold for the Telfer operation, the operational focus is on maximising gold recovery. To achieve this goal, the general philosophy for circuit design and operating strategy is:

1. Targeting a primary grind size optimum for copper recovery.
2. Operating the main flotation circuit in a sequential mode.
3. Targeting a minimum saleable copper concentrate grade.
4. Leaching the pyrite concentrate to extract the gold.

There are also a number of options considered to further enhance metallurgical performance, including:

1. A single stage of cleaning for the copper rougher concentrate.
2. Regrinding of the copper scavenger concentrate prior to cleaning.

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Table 1
Comparison of copper and gold recovery in copper/gold operations.

Operations	Throughput head grade			Recovery			Throughput head grade			Recovery		
	k tonnes	% Cu	g/t Au	Cu	Au	Diff.	k tonnes	% Cu	g/t Au	Cu	Au	Diff.
	2009						2008					
Aitik – Sweden	9258	0.28	0.13	89.2	55.8	33.4	17,813	0.30	0.14	88.4	48.8	39.5
Alumbrera – Argentina	18,919	0.49	0.56	82.5	73.7	8.8	37,502	0.50	0.55	83.7	76.1	7.6
Batu Hijau – Indonesia							37,818	0.47	0.28	80.6	75.2	5.4
Cadia Hill – Australia	17,163	0.19	0.71	86.0	75.6	10.4	16,792	0.18	0.94	88.8	82.1	6.7
Ernst Henry – Australia							11,406	1.07	0.53	90.9	75.2	15.7
Freeport/Grasberg – Indonesia	43,362	1.11	1.32	90.6	82.9	7.7	70,409	0.83	0.66	90.1	79.9	10.2
Kennecott/Bingham Canyon – US	26,030	0.69	0.48	89.6	70.9	18.6	49,134	0.58	0.35	83.5	66.6	17.0
Northparkes – Australia	2789	0.71	0.24	88.9	69.2	19.6	5244	0.54	0.26	87.6	73.7	13.9
Ok Tedi – Papua New Guinea	10,546	0.83	1.06	85.8	69.0	16.8	21,663	0.85	1.02	86.7	72.5	14.2
Osborne - Australia	2549	1.93	0.65	94.5	79.0	15.5	8020	1.83	0.59	93.8	78.4	15.4
Ridgeway – Australia	5860	0.56	1.52	85.8	80.9	4.9	5775	0.67	1.93	89.6	84.7	4.9
Tintaya – Peru	3354	1.22	0.24	82.0	65.0	17.1	7110	1.38	0.27	85.2	59.5	25.8
	2007						2006					
Aitik – Sweden	18,178	0.32	0.14	86.8	46.3	40.5	18,481	0.40	0.25	89.5	50.7	38.8
Alumbrera – Argentina	38,607	0.56	0.67	83.4	74.0	9.4	36,350	0.56	0.71	88.5	77.3	11.2
Batu Hijau – Indonesia	46,782	0.60	0.44	86.1	81.9	4.2	47,026	0.55	0.37	87.3	79.5	7.8
Cadia Hill – Australia	17,817	0.18	0.58	89.3	79.7	9.6	15,501	0.14	0.65	84.3	76.7	7.6
Ernst Henry – Australia	11,114	0.94	0.47	91.7	73.4	18.3	10,301	0.89	0.44	91.6	72.4	19.1
Freeport/Grasberg – Indonesia	77,599	0.82	1.24	90.5	86.2	4.3	83,731	0.85	0.85	86.1	80.9	5.2
Kennecott/Bingham Canyon – US	47,525	0.53	0.38	84.2	68.4	15.9	47,857	0.63	0.49	88.1	69.4	18.7
Northparkes – Australia	5,297	0.91	0.62	89.4	74.6	14.8	5,789	1.53	0.64	94.0	79.5	14.5
Ok Tedi – Papua New Guinea	25,771	0.76	0.85	86.4	70.8	15.6	27,561	0.82	0.88	86.0	70.5	15.5
Osborne – Australia	8,270	2.09	0.75	93.7	80.2	13.5	3,147	1.85	0.96	88.2	72.2	16.0
Ridgeway – Australia	5,694	0.72	2.00	90.6	85.7	4.9	5,538	0.71	2.40	91.3	85.8	5.5
Tintaya – Peru	6,767	1.44	0.36	86.0	49.5	36.5	6,556	1.35	0.33	88.5	57.6	30.8

Note: 1. Information sourced from company annual reports, including Xstrata, Rio Tinto, Freeport-McMoRan, Newmont, Barrick, Ok Tedi and Newcrest. 2. Year reported refers to financial year which may differ from calendar year depending on the countries.

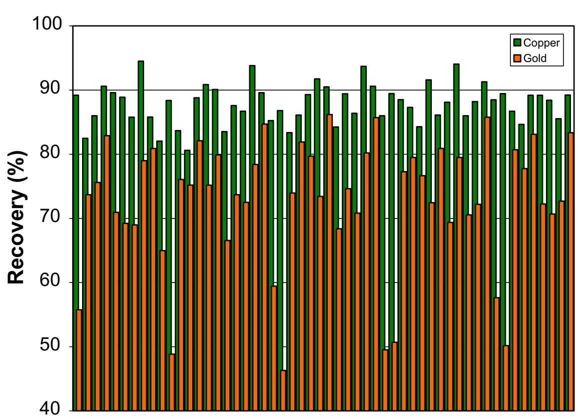

Fig. 1. Comparison of copper and gold recovery in copper/gold operations.

3. Regrinding of the pyrite rougher concentrate followed by additional flotation of copper and gold prior to leaching.

4. Optimum grind size

For a given ore, mineral liberation is generally directly related to particle size. Flotation could only proceed if the particle is sufficiently liberated. Meanwhile, particles of different sizes react differently to the hydrodynamic conditions in a flotation cell, affecting the probabilities of bubble-particle collision, attachment and detachment. Hence, particle size has a strong influence on recovery.

The relationship between grind size and recovery is studied using a modelling technique developed by Bazin et al. (1994). The procedure is to predict particle size distribution at different grind sizes according to the Rosin–Rammler model proposed by Edwards and Vien (1999). By plotting the cumulative percent passing size distribution against the cumulative percent passing metal distribution and fitting a cubic spline function to the data (Ahlberg et al., 1967), the cumulative metal distribution can be predicted at any given grind size. Fundamentally, this relationship accounts for both preferential breakage and classification segregation effects observed in the plant. The overall flotation recovery can then be predicted by combining the newly calculated metal distribution with the size-by-size recovery obtained from the plant survey. The model of Bazin et al. (1994) has been independently validated by Runge et al. (2007) and proved to be robust. Fig. 3 shows a typical size-by-size recovery profile in the Train 1 copper flotation circuit. Based on this size-by-size recovery profile, recovery at various grind sizes is simulated and plotted in Fig. 4. Note that the cyclosizer cut size (C5) shown in Fig. 3 is approximately 6.4 μm for copper sulphides and 2.8 μm for gold.

The optimum grind size for copper recovery is estimated to be close to 80 μm in Fig. 4. Meanwhile, within the particle size range simulated, gold recovery continues to increase as the grind size decreases. Simulation based on a wide range of survey or monthly composite data produced consistent results with the optimum grind size for copper recovery likely in a range of 75–85 μm and gold recovery improving with a finer grind.

Table 2
Train 1 feed typical mineral composition.

Minerals	% Mass distribution
Chalcopyrite	0.61
Bornite	<0.01
Chalcocite/covellite	0.034
Cuprite	<0.01
Native copper	<0.01
Pyrite	2.11
Goethite	1.29
Other NSG	95.9

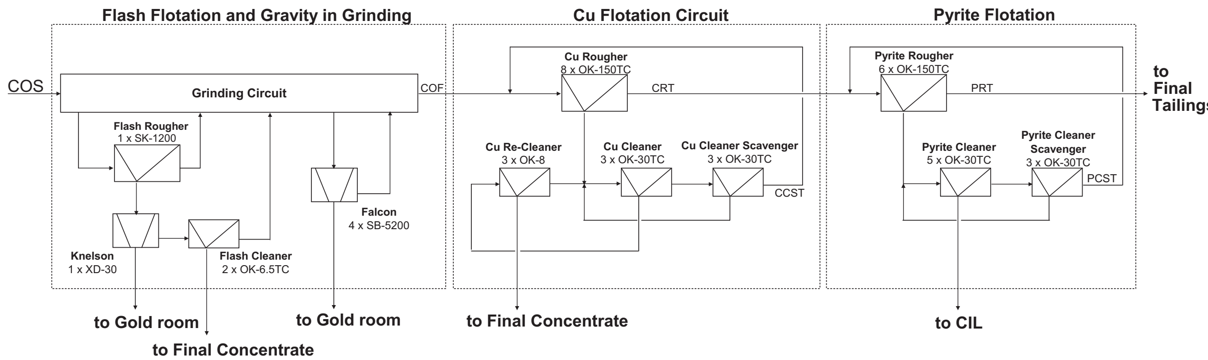

Fig. 2. Telfer Train 1 circuit flowsheet.

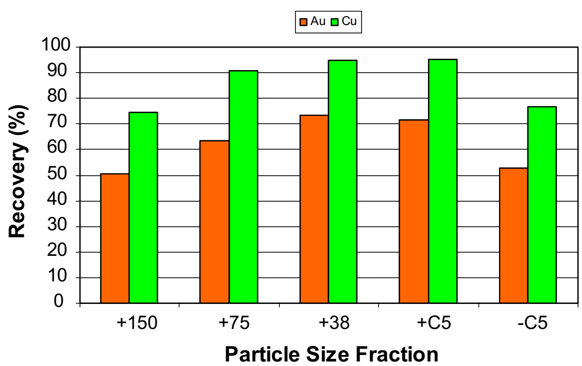

Fig. 3. Size-by-size recovery of copper and gold in Train 1 copper flotation circuit.

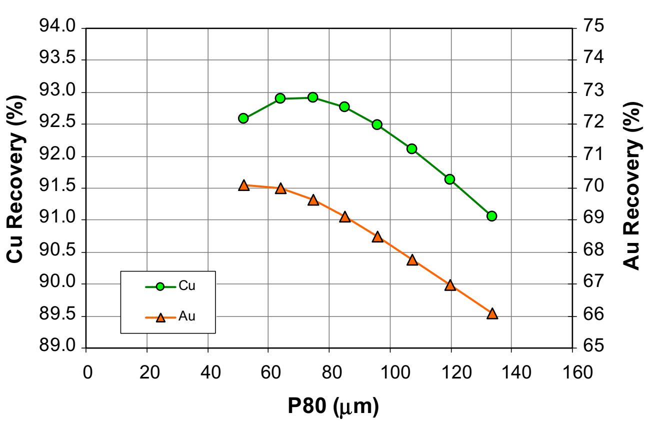

Fig. 4. Simulated grind size versus recovery of copper and gold in Train 1 copper flotation circuit.

It is common to see a plant design with a relatively coarse primary grind followed by a regrind on the rougher/scavenger concentrate. One of the metallurgical arguments for such an arrangement is to reduce the potential risk of valuable minerals being over-ground. However, coarse particles are generally difficult to float due to poor liberation and/or unfavourable cell hydrodynamic and froth transportation conditions. This is evident in operations such as Escondida and Freeport.

Coleman et al. (2006) reported that in the two surveys conducted at the Laguna Seca and Los Colorados concentrators at Escondida, copper losses in the +106 μm size fraction were greater than those in the −53 μm size fraction in the rougher tailings. The rougher copper recovery was measured as 83.1% and 85.9% at a feed size P_{80} of 172 μm and 180 μm at Laguna Seca and Los Colorados concentrators respectively. It can be estimated that the losses in the +106 μm size fraction were equivalent to approximately 7% of the total copper in the feed. Huls and Hill (2005) proposed a coarse particle recovery process (CPR) to re-float the coarse fraction of the rougher tailings at Escondida. It was acknowledged that CPR flotation would not be achieved in conventional flotation cells and would require extreme operating conditions. To date, the CPR process has not been implemented.

A similar phenomenon can be observed at Freeport (Zheng et al., 2007a). At a feed size P_{80} of approximately 140 μm at the Freeport C4 concentrator, more than half of the copper lost in the rougher tailings was in the +106 μm size fraction. With increase in feed size among the four concentrators at Freeport, the proportion of copper loss also increased. This can be explained by the size-by-size recovery results: copper recovery in the −20 μm size fraction was higher than that in the +100 μm fraction at Freeport.

At Telfer, the plant flotation feed had an average P_{80} of 95 μm, significantly finer than the other major copper/gold operations. This gives Telfer an advantage to achieve a higher recovery despite no regrinding of rougher/scavenger concentrate.

5. Sequential flotation versus copper only flotation

Sequential flotation produces two separate concentrates of copper and pyrite. The pyrite concentrate is then further treated in a CIL circuit to extract gold. In addition to losses of valuables in the final pyrite flotation tailings, the other main losses include copper in the pyrite concentrate and gold in the leached tailings. Nevertheless, by producing an additional pyrite concentrate, sequential flotation has a number of advantages over the copper only flotation even though the pyrite flotation circuit can be utilised to extend copper flotation during copper only flotation.

Pyrite is generally abundant in the Train 1 feed ore and better liberated than copper minerals. This can be seen in the production data at Telfer. Train 1 produced more pyrite concentrate than the combined copper concentrate despite the Train 1 pyrite rougher bank having two cells less than the copper rougher bank. Meanwhile, on average, the pyrite concentrate contained more than 90% sulphides compared with more than 20% non-sulphide gangue in the copper concentrate. As a result of abundant, well liberated pyrite in the ore, pyrite flotation generally produced more stable froth and improved composite recovery.

Based on the production data of the first 6 months of year 2009 when Train 1 was operated in a sequential mode, approximately 2.1% of the total copper reported to the pyrite concentrate. The value of copper recovered in the pyrite concentrate currently cannot be counted as revenue. However, once the pyrite concentrate is available, it provides opportunities to recover not only gold but also more copper.

Because of the reasons mentioned above, Telfer Train 1 flotation circuit is designed and always operated in a sequential flotation mode unless the CIL circuit is not available. In addition pyrite, especially the proportion as composite with copper minerals, is often not actively depressed in the copper flotation circuit. This is aimed to reduce copper reporting to the pyrite circuit and more importantly to improve gold recovery. Higher copper content in the pyrite concentrate increases the chance of higher cyanide consumption and/or lower gold recovery in the CIL circuit. Furthermore, with unground feed, gold recovery in the CIL circuit is relatively limited at approximately 75%. Operating cost per unit of gold produced in the pyrite flotation and CIL circuit is also higher than copper flotation.

When operating the circuit in a copper only flotation mode, a minimum acceptable concentrate grade is usually targeted to maximise gold recovery. This is similar to a bulk flotation strategy which produces only one combined copper and pyrite/gold concentrate. Given sufficient circuit capacity, bulk flotation and sequential flotation should achieve the same total recovery of copper and pyrite/gold. However, it is difficult to achieve a saleable concentrate copper grade by bulk flotation on a copper sulphide ore or a mixture of sulphide/oxide ore due to pyrite dilution. Bulk flotation generally can only be used for oxide copper ores with low pyrite content. Depending on the oxide copper and pyrite content, the reagent scheme can be modified to suit not only for copper only flotation or bulk flotation but also for copper only flotation with additional pyrite recovery or bulk flotation with incomplete pyrite recovery.

6. Gold associated with pyrite

The proportion of the gold associated with pyrite is estimated by conducting sequential flotation tests in the laboratory. Train 1 ore contains predominantly chalcopyrite. Chalcopyrite is known to be floatable without collector in a moderate oxidising environment while pyrite does not float naturally under an alkaline condition (Heyes and Trahar, 1977; Gardner and Woods, 1979; Janetski et al., 1977; Ahlberg et al., 1990). Therefore, a collectorless flotation followed by another stage of flotation in the presence of a collector, relatively pure copper concentrate can be obtained separately from pyrite. Gold distribution in chalcopyrite and pyrite can then be estimated. Fig. 5 shows the recovery of chalcopyrite, pyrite and gold obtained from a laboratory flotation test. The sample was taken directly from the plant SAG mill trommel undersize stream and ground in a laboratory ball mill. The final grinding product/laboratory flotation feed had a P_{80} size of 99 µm, close to that of the plant cyclone overflow. The pyrite flotation stage used 8 g/t of PAX.

Nearly 90% of the copper with 71% of the gold in the sample was recovered by collectorless flotation alone. The flotation kinetics of chalcopyrite appeared to be fast. The recovery reached a plateau during the collectorless flotation. At the same time, the total pyrite recovery from the collectorless flotation was 3.6%, the same as the non-sulphide gangue. This implies that pyrite was non-floatable in the absence of a collector and recovered into the copper concentrate by entrainment.

Pyrite floated in the presence of PAX. After 6 min, an additional 5.3% of the copper and 22% of the gold were recovered together with 74.7% of the pyrite. Although not all the copper and gold recovered in this stage with collector were associated with pyrite, the results from this sequential flotation test can still be used as indicators for how much copper and gold are associated with pyrite and what recoveries are to be expected when the plant circuit is operated in a sequential flotation mode or a copper only flotation mode.

Based on the plant production data, the Train 1 pyrite flotation circuit recovered 2.1% of the copper and 9.2% of the gold in the mill feed. It should be noted that a portion of the pyrite is allowed to be recovered into the copper concentrate at Telfer. Typically, the Train 1 final copper concentrate contains more than 15% pyrite.

7. Current operation performance

Under the current operating strategies i.e. relatively fine primary grind size, copper/pyrite sequential flotation and allowance of partial pyrite recovery into the copper concentrate, Telfer achieved a copper recovery of 91.5% and a gold recovery of 90.7% at a head grade of 0.28% copper and 1.4 g/t gold in Train 1 during the first 6 months of year 2009. During the same time period, the copper concentrate contained an average of 18.7% copper and 60.7 g/t gold. Figs. 6 and 7 compare the gold and copper recovery at Telfer with the other operations listed in Table 1.

It should be noted that recovery alone does not represent the optimum metallurgical objective. The optimum metallurgical objective should take into account recovery, concentrate grade and mill throughput depending on ore properties and other site specific conditions, and can only be determined ultimately by the economic return. This is beyond the scope of the current paper. This paper simply provides a methodology and a working example for an option of maximising gold recovery in a specific copper/gold ore at Telfer. One of the important characteristics of the Telfer ore is that it has a higher gold to copper ratio as shown in Fig. 8.

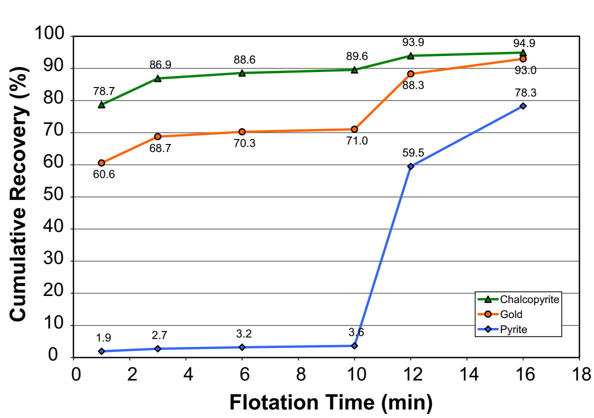

Fig. 5. Laboratory sequential flotation of chalcopyrite and pyrite.

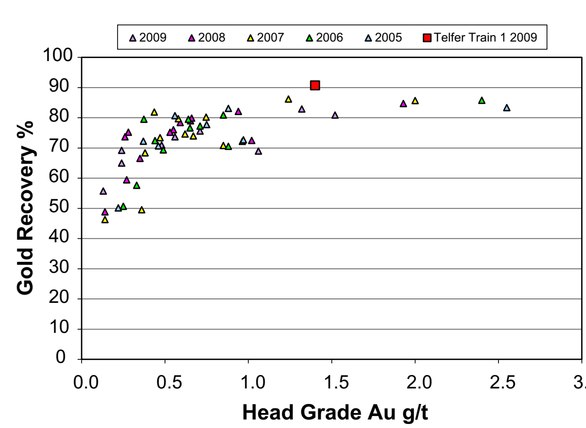

Fig. 6. Gold recovery versus feed grade from various operations.

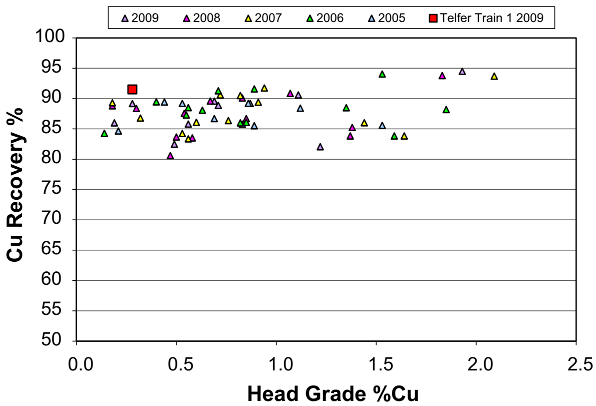

Fig. 7. Copper recovery versus feed grade from various operations.

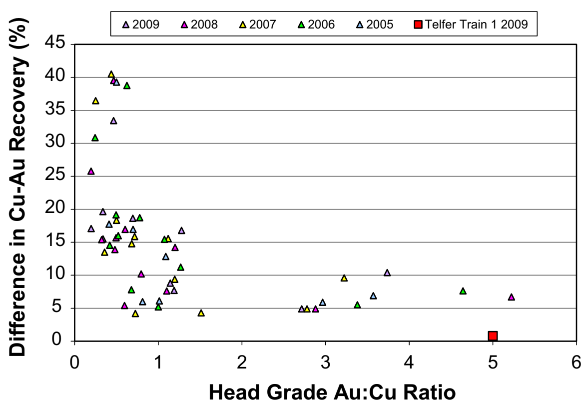

Fig. 8. Difference in copper and gold recovery as a function of head grade gold to copper ratio.

Due to the relatively high gold to copper ratio in the ore and gold providing the main revenue for the operation at Telfer, emphasis has been on maximising recovery while allowing a relatively low copper concentrate grade to be produced. However, the current studies indicate that there are opportunities to increase the recovery further and improve concentrate grade without sacrificing recovery. The main opportunities include an additional single stage of cleaning of the copper rougher concentrate, regrinding the copper scavenger concentrate prior to cleaning and regrinding the pyrite rougher concentrate for additional copper/gold flotation with flotation tailings for CIL gold leaching.

8. Copper rougher concentrate cleaning

The copper rougher consisting of two cells preferentially recover liberated fast floating copper mineral particles. The amount of composite material in the rougher concentrate is relatively minor. The main dilutant is entrained non-sulphide gangue, which typically constitutes more than 35% of the total mass of the Train 1 copper rougher concentrate. A single stage of cleaning using a laboratory flotation cell reduced the non-sulphide gangue content from 38% in the initial copper rougher concentrate sample to 19% and increased the copper concentrate grade from 20% to 26.4% at a copper recovery of 94.5% (Zheng et al., 2009). It is anticipated that a higher concentrate grade can be achieved in an industrial scale cleaner cell, especially if a column type of flotation cell is used, at a higher recovery (Zheng et al., 2007b). The tailings from the copper rougher cleaner can then be combined with the copper scavenger concentrate for further regrinding and cleaning.

9. Copper scavenger concentrate regrinding

After the liberated, medium size copper mineral particles have been removed in the copper rougher, the copper scavenger cells pick up proportionally more composite particles as well as slow floating fines. Regrinding of the copper scavenger concentrate together with classification would provide an effective means to liberate the copper minerals and gold from the composites and likely improve concentrate grade in the cleaner circuit. The preliminary results from the laboratory regrinding/flotation on the Train 1 copper scavenger concentrate indicate that the final concentrate copper grade increased from 15% Cu without regrinding to 21% Cu with regrinding and gold from 27 g/t to 54 g/t with a head grade of 4.7% copper and 8.8 g/t gold in the initial Train 1 copper scavenger concentrate sample (Zheng et al., 2009).

10. Copper separation from pyrite concentrate

There is an incentive to separate copper from the pyrite concentrate as much as possible prior to leaching as copper in the pyrite concentrate could increase cyanide consumption and reduce gold recovery in the CIL circuit. Currently, pyrite concentrate is not being reground prior to CIL processing. An experimental investigation was conducted in the laboratory, involving regrinding the pyrite concentrate, separating copper by flotation and leaching the flotation tailings.

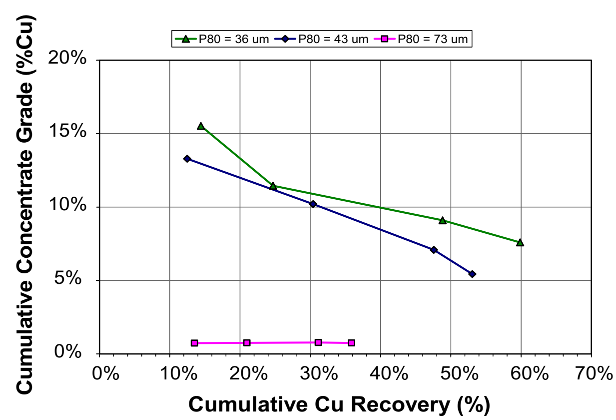

Fig. 9. Comparison of copper recovery and grade in flotation of pyrite concentrate samples without additional collector at various grind sizes.

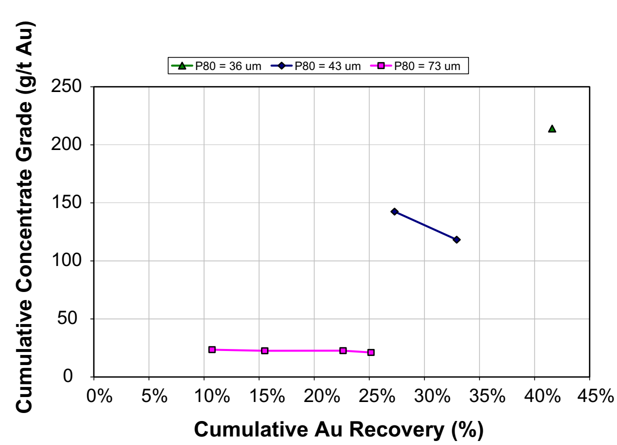

Fig. 10. Comparison of gold recovery and grade in flotation of pyrite concentrate samples without additional collector at various grind sizes.

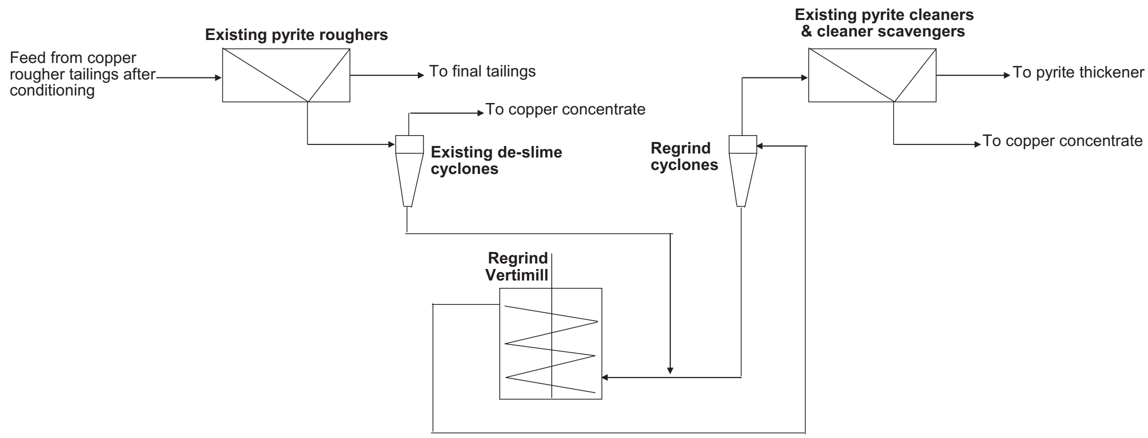

Fig. 11. Pyrite flotation circuit incorporating regrinding and additional copper/gold flotation on pyrite concentrate.

A fresh pyrite concentrate slurry sample was taken from the plant. The sample had a head assay of 5.3 g/t gold, 0.13% copper and 42.4% sulphur. P_{80} was measured as 73 µm. Three flotation tests were conducted on one original sample and two after being reground in a laboratory rod mill to a P_{80} of 43 µm and 36 µm respectively. No collector was added in the flotation test. The slurry pH was adjusted to 10.5. Flotation was conducted for 10 min with four concentrates collected. The results of the flotation tests at three different grind sizes are compared in Figs. 9 and 10 for copper and gold, respectively.

As can be seen in Figs. 9 and 10, relatively limited recovery and poor concentrate grade were obtained when flotation was performed on the original pyrite concentrate sample without regrinding. Regrinding significantly improved both recovery and concentrate grade. Gold was more sensitive to grind size than copper.

At the concentrate grade achieved at the finest grind, the new concentrate produced from the pyrite concentrate can be blended into the existing plant copper concentrate. This would increase the overall plant Train 1 copper recovery by more than 1% while lowering the final combined concentrate copper grade by less than 0.5%. The grade of gold in the final combined concentrate would not decrease, as gold grade in the re-float concentrate of the pyrite concentrate sample was more than two times higher than the average gold grade of the existing plant copper concentrate i.e. 215 g/t versus 65 g/t.

Regrinding of the pyrite concentrate would also improve the gold recovery in the CIL circuit. The preliminary test results indicate that by reducing the CIL feed size from a P_{80} of 75 µm to 35 µm, the leaching recovery of gold increased by more than 10%. The potential increase in cyanide consumption in the CIL circuit due to finer feed would be offset by a decrease in the total amount of gold and copper in the feed to the CIL circuit after the additional flotation. In addition, maintenance costs associated with the CIL circuit would likely decrease due to finer CIL feed and less wear.

The laboratory experimental test results and the anticipated improvement in the plant as a result of pyrite concentrate regrind are supported by the recent mineralogy study (Wightman, 2009). It was found that:

- Gold in the pyrite concentrate is present as native gold and gold alloy (electrum) which is flotation recoverable and cyanide leachable.
- Approximately 80% of the gold is locked with pyrite.
- More than half of the gold grains present are less than 10 µm.

Based on the above metallurgical justifications, it is proposed that the existing pyrite flotation circuit can be modified to improve copper concentrate production. Fig. 11 shows the proposed pyrite flotation circuit configuration.

No additional flotation cells are required in the proposed circuit modification. Regrinding is to be carried out on the pyrite rougher concentrate. Based on the plant operational data, pyrite rougher concentrate normally contains over 80% sulphides. The reduction in concentrate mass after the cleaning stage is insignificant. The existing pyrite cleaner circuit can hence be used for the additional copper/gold flotation with the tailings going to the existing pyrite thickener and onwards to the CIL circuit.

With this supplemental copper/gold flotation circuit available in the pyrite flotation circuit, it will also reduce the risk of losing copper when fluctuations and disturbances occur in the main copper flotation circuit.

11. Summary

A number of operating strategies have been implemented together with a sequential flotation circuit design to maximise recovery of the gold in the copper/gold ore at Telfer. The main operating strategies include targeting a primary grind size optimum for copper recovery, designing and operating the main flotation circuit as copper and pyrite sequential flotation, targeting a minimum saleable concentrate copper grade, allowing a portion of pyrite/gold recovered into the copper concentrate and leaching the pyrite concentrate to extract the gold. These operating strategies have lifted the recovery of both copper and gold above 90% in Train 1 in the first 6 months of year 2009.

Opportunities to improve final copper concentrate grade exist, including a single stage of cleaning of the Train 1 two cell copper rougher concentrate and regrinding of the copper scavenger concentrate prior to cleaning. There is also a potential to increase both copper and gold recovery further by regrinding the pyrite rougher concentrate and utilising the existing pyrite cleaning circuit for additional copper/gold flotation. A schematic representation of the future process route is shown in Fig. 12. Potential improvement in recovery and concentrate grade is estimated in Table 3.

Other benefits to be gained from the proposed sequential flotation circuit configuration and operating strategies are a potentially reduced risk of losing copper due to fluctuations and disturbances in the main copper flotation circuit, lower maintenance and operating costs for the CIL circuit and lower reagent (collector, lime and cyanide) consumption.

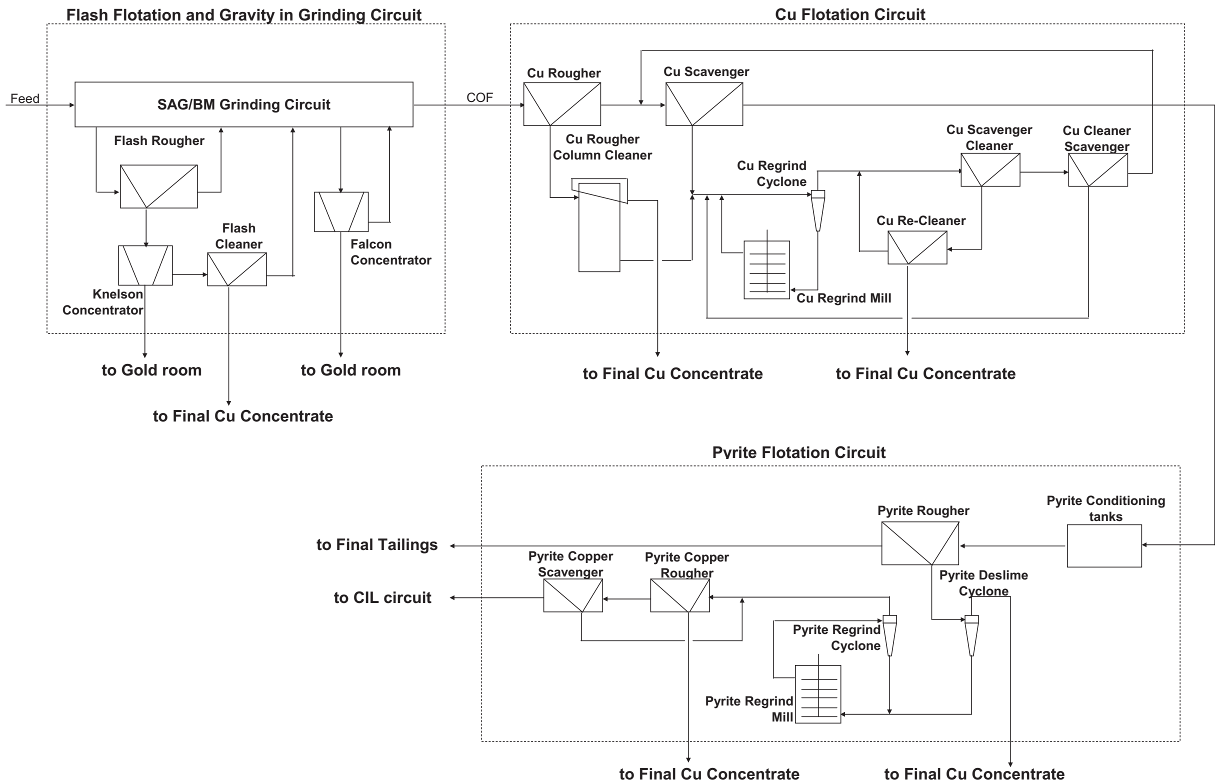

Fig. 12. Sequential flotation circuit configuration.

Table 3
Estimated potential metallurgical improvement with the proposed circuit.

	Present			Potential		
	Cu recovery %	Au recovery %	Concentrate % Cu	Cu recovery %	Au recovery %	Concentrate % Cu
Gravity		21.7			21.7	
Flash flotation	15.7	22.2	19.5	15.7	22.2	19.5
Copper flotation	76.7	41.0	18.6	76.7	41.0	21.0
Pyrite flotation		9.2		1.3	9.2	14.0
Pyrite flotation/CIL		6.9			8.3	
Total	92.4	91.8	18.8	93.6	93.2	20.7

Notes:
1. The present data in Table are based on the first 6 months of production results in 2009 when the circuit was operating in a sequential mode.
2. Increase in concentrate grade from the copper flotation circuit is estimated based on reduction of non-sulphide gangue content in the first two rougher cell concentrate from 38% to 15%.
3. Additional copper recovery from the pyrite flotation circuit is estimated based on 2.1% of the copper in the ore currently being recovered into the pyrite rougher concentrate and 60% of this copper to be recovered into a separate copper concentrate after regrinding the pyrite rougher concentrate.
4. Additional gold recovery from the pyrite flotation and CIL circuit is estimated based on an improvement in recovery from 75% to 90%.

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