Expansion of the Mount Isa Mines Copper Concentrator Phase One Cleaner Circuit Expansion

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ABSTRACT

Mount Isa Mines Limited operates one of the world's largest underground mines in North West Queensland. It's mining and smelting complex produces copper anode, crude lead bullion (containing silver), and zinc concentrate. Copper is recovered in the Copper Concentrator from chalcopyrite ore, converter smelter slag and rotary holding furnace (RHF) slag via grinding and flotation operations.

During 2002 a project to optimise cleaner circuit capacity was commenced. Its aims were to minimise copper losses and minimise slag/chalcopyrite cross contamination by reconfiguring existing flotation equipment and the addition of new flotation capacity. To minimise cross contamination a Jameson Cell was installed and dedicated to cleaning of slag concentrates, which had formerly been conducted by three flotation columns operating in series. The flotation columns were re-configured to operate in parallel and dedicated to upgrading chalcopyrite concentrate. In addition a Jameson Cell was installed to reject entrained copper from a prefloat concentrate consisting of talc and carbonaceous pyrite.

This paper discusses the logic behind the cleaner circuit expansion, recent test work relating to the Jameson Cell and the installation and commissioning phases. Difficulties relating to the commissioning are reviewed and the solutions to problems are highlighted. The benefits the reconfigured circuit has provided to Mount Isa Mines are also discussed.

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JUSTIFICATION

The circuit capacity during commissioning of the Copper Concentrator in 1973 is now inadequate due to:

- increase in throughput rate and variability;
- increase in head grade and head grade variability resulting from Enterprise Mine ore;
- increase in talc in feed; and
- higher concentrate grade targets; meaning 15 per cent of installed roughing capacity is now lost to preflotation.

Chalcopyrite ore treatment justification

From 1975 to 1980 the Copper Concentrator treated 154 000 t Cu in feed per year. In 2001/02 it treated 230 000 t contained Cu in feed; a 50 per cent increase with 15 per cent less roughing capacity.

Ongoing metallurgical investigations and full plant demonstrations showed that a two per cent increase in copper recovery could be achieved if flotation capacity is increased. These demonstrations show that increased recovery is achieved when one line of grinding and the equivalent of double the flotation capacity is used. The concentrator flow sheet is shown in Figure 1 and illustrates the parallel grinding and primary flotation with a common cleaning circuit.

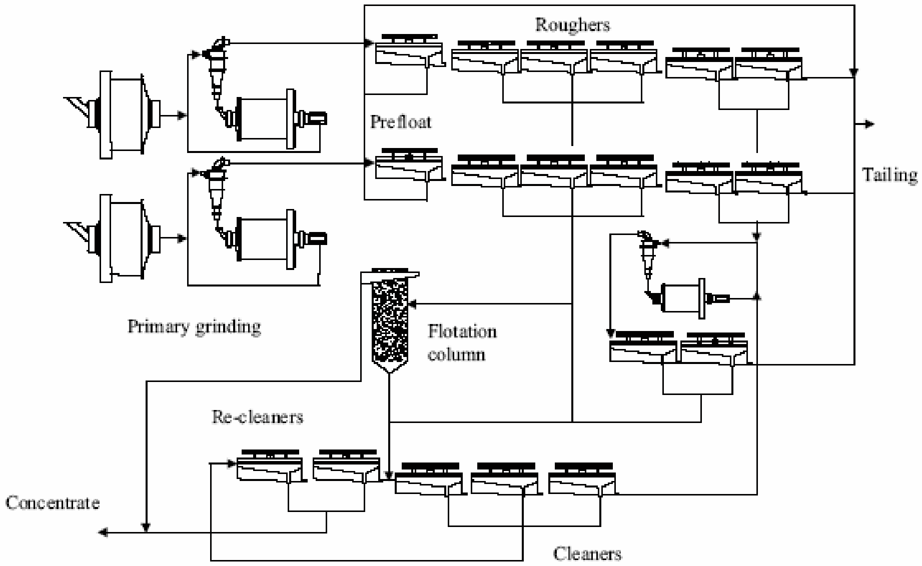

FIG 1 - Pre-expansion parallel milling flow sheet.

Over time there has been an increase in the amount of naturally floatable species of carbonaceous pyrite and talc in ore. This would normally decrease concentrate grade for a given recovery; however at the same time the concentrator has been required to increase concentrate grade from 25 per cent to 27 per cent Cu. A preflotation circuit to remove naturally floating species has been necessary to achieve this. Fifteen per cent of rougher scavenger capacity has been reallocated to form the preflotation rougher circuit. There is no facility to clean preflotation rougher concentrate, causing higher than necessary copper losses by entrainment.

Slag treatment justification

Converter slag and RHF slag are both produced within the copper smelting complex of Mount Isa Mines and contain economically recoverable quantities of copper as chalcocite and metallic copper. When the Copper Concentrator capacity periodically exceeds that of the mine the opportunity is taken to treat converter and RHF slag.

The contamination of the chalcopyrite flotation with as little as 2 wt per cent slag results in total collapse of the flotation froth and loss of recovery. This collapse can be overcome with uneconomic additions of either sodium isobutyl xanthate (SIBX) or methyl isobutyl carbinol (MIBC). The mechanism by which slag removes the collector from the surface of the chalcopyrite particles is under investigation. This phenomenon is common to both converter and RHF slag. For this reason the flotation of ore and slag are completed independently of each other minimising cross contamination.

Performance at both Hilton Concentrator and in the Copper Concentrator indicated that froth washing techniques were the most effective forms of cleaning to achieve high-grade concentrates for smelting. Figure 2 shows the historical performance of converter slag in the Copper and Hilton Concentrators.

Plant test work was performed over the past five years to confirm the additional flotation requirements in the Copper Concentrator. Trials were conducted on a pilot scale and where possible on full plant scale. The level of both pilot plant and full plant investigations all but eliminated any significant technical risk. More recently, work was performed to confirm the proposed flotation flow sheet.

Proposed circuit expansion

Five areas were identified that would contribute to the increase in recovery. These areas were:

1. Jameson Cell for cleaning preflotation rougher concentrate;
2. Jameson Cell for slag cleaning;
3. changing column pipe work from series to parallel operation;
4. tank cells for scavenging; and
5. tank cells for cleaner scavenging.

This paper discusses the first three changes including the installation of two Jameson cells and changes to the copper columns. Benefits include the reduction of slag contamination in chalcopyrite flotation currently resulting in 0.3 per cent lower copper recovery per year and reduction in copper lost to preflotation (currently 0.5 - two per cent copper).

PROCESS DESCRIPTION

Mt Isa copper treatment

Chalcopyrite ore and converter slag are treated to produce copper concentrates at the Copper Concentrator of Mount Isa Mines Limited, Mount Isa, Qld. The concentrator was commissioned in 1973, replacing the original No. 1 Lead-Zinc Concentrator, for the processing of all the chalcopyrite ore coming from Mount Isa mine (Lumsdaine, O'Hare, 1980).

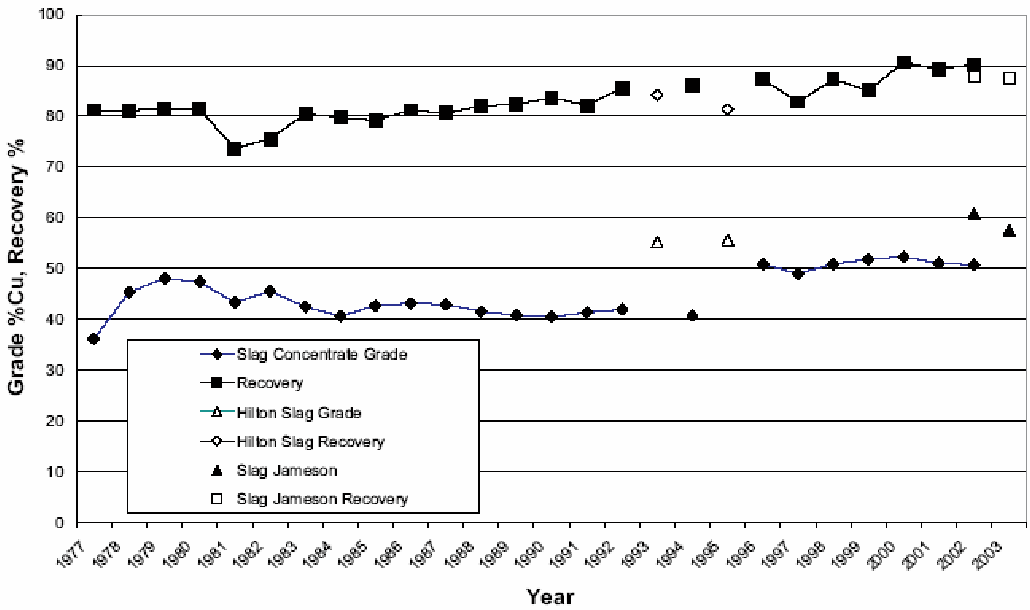

FIG 2 - Historical smelter slag metallurgical performance.

At present production ore is sourced from the southern 1100, 3000 and 3500 orebodies. Currently 50 per cent of mill feed is from the 3000 and 3500 orebodies. Chalcopyrite is the only significant copper mineral and occurs as a replacement deposit in a silica-dolomite host rock. Sulfide gangue consists of pyrite (FeS_2) and minor amounts of pyrrhotite and cobaltite. The ore averages eight per cent sulfur.

Future ore production will draw increasingly larger tonnages from the 3000 and 3500 orebodies. The host rock is very similar to that of the 1100 orebody but is slightly more siliceous in nature. Silica assays range from 60 to 70 per cent SiO_2 and the ore is both abrasive and hard with a current Bond Work Index of approximately 22kWh/t. Copper mineralisation is disseminated through the silica-dolomite. Mineralogical examination indicates liberation is about 75 per cent at a P_{80} of 150 μm. Flotation feed sizing is normally in the range 80 per cent passing 150 μm, depending on milling rate and ore type.

Smelter slag treatment

Treatment of converter slag in the 1970's and 1980's was on a batch basis during periods of ore shortage. The batch processing of Copper Smelter converter slag began in 1976 (Bartrum *et al*, 1978). Parallel processing of slag and ore commenced in 1992 and the flotation circuit was modified to simplify parallel processing in 1998 when it was determined that Copper Smelter Rotary Holding Furnace (RHF) slag also had economic retreatment value. Slag was treated through the concentrator using the existing flow sheet. In 1991 the comminution circuit consisting of secondary crushing, screening, open circuit rod milling and closed circuit ball milling was replaced with an AG/SAG/ball mill circuit. With the introduction of AG/SAG milling the treatment of slag could be performed in parallel with ore using one line of grinding, primary flotation and the flotation columns (Barnes *et al*, 1993). This flow sheet is shown in Figure 3. During 1993 and 1995 the nearby Hilton Concentrator was converted from treating lead zinc to converter slag to reduce the stockpile of slag that had developed due to high output from the mine and low concentrator throughput. For a period of this time Hilton Concentrator operated Jameson cells in the slag cleaner duty (Dawson, Harbort, 1996).

Preflotation

Periodically amounts of naturally floatable species of carbonaceous pyrite and talc occur in the Mt Isa copper ore. Early treatment in the Copper Concentrator consisted of flotation, followed by depression with ambiguous results (Hoffman, 1965; Lyon, Fewings, 1971). Investigation by Grano, Ralston and Smart, in 1990 identified the natural floatability of carbonaceous pyrite. An evaluation of using preflotation prior to roughing, rather than the flotation/depression route was conducted.

The preflotation circuit operated in the Copper Concentrator from 1992-1996. Originally the preflotation rougher concentrate was cleaned in the three flotation columns operating in series. Griffin (1993) reported that this circuit increased the concentrate grade from 25 per cent Cu to 27.2 per cent Cu for a 1.5 per cent loss in copper recovery. The loss consisted of 0.6 per cent Cu lost to the prefloat concentrate in the form of carbonaceous copper and 0.9 per cent loss due to the decrease in scavenger capacity.

In 1996 the duty of the columns was changed from preflotation cleaning to primary cleaners treating copper rougher concentrate and producing up to 50 per cent of the final concentrate. This circuit enabled consistently high concentrate grades to be achieved without the preflotation circuit. As the ore was sourced increasingly from the deeper copper orebodies 3000 and 3500 the level of talc increased making concentrate targets harder to achieve. The prefloat circuit was reinstated with the prefloat rougher concentrate being sent straight to final tail.

Work on a pilot scale using Jameson Cells for one stage preflotation and for preflotation cleaning had been performed over a number of years. A series of pilot tests were conducted in 2000 to confirm parameters for engineering design.

Jameson Cell operating principles

The principles of Jameson Cell operation have been discussed by numerous authors including Jameson (1998), Jameson *et al* (1998) and Evans *et al* (1995). Recent developments have been reviewed by Harbort *et al* (2003) and Carr *et al* (2003). Operation can be described with reference to Figure 4.

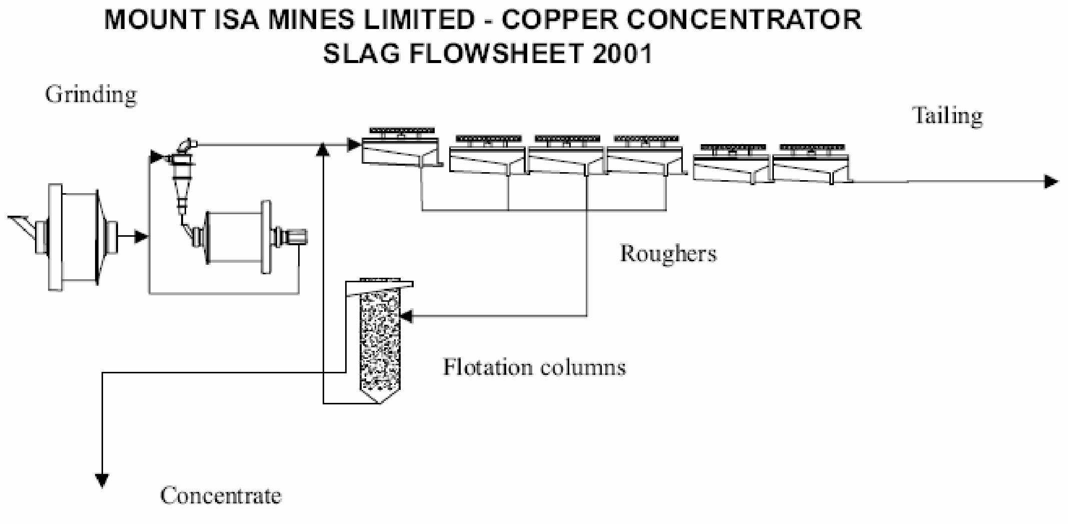

FIG 3 - Pre expansion slag circuit.

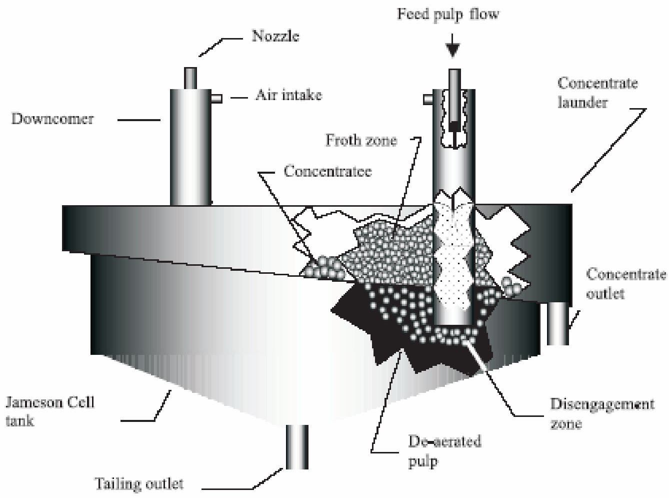

FIG 4 - Jameson Cell schematic.

The Jameson Cell can be divided into three main zones:

- The downcomer where primary contacting of bubbles and particles occurs. Feed pulp is pumped into the downcomer through an orifice plate, creating a high-pressure jet. The plunging jet of liquid shears and then entrains air, which has been naturally aspirated. Due to a high mixing velocity and a large interfacial area there is rapid contact and collection of particles.
- The tank pulp zone where secondary contacting of bubbles and particles occurs and bubbles disengage from the pulp. The aerated mixture exits the downcomer and enters the pulp zone of the flotation tank. The velocity of the mixture and large density differential between it and the remainder of pulp in the tank results in recirculating fluid patterns, keeping particles in suspension without the need for mechanical agitation.
- The tank froth zone is where materials entrained in the froth are removed by froth drainage and/or froth washing.

DESIGN

Jameson Cells

For cleaning of both converter slag and RHF slag a model E2532/6 Jameson Cell was installed. The model designation refers to a rectangular cell, 2.5 m in length × 3.2 m wide, equipped with six downcomers and an internal launder. A wash water system was included that consisted of a set of perforated trays suspended above the froth. At the design phase it was envisaged that operation on converter slag would require six downcomer operation with RHF slag only requiring three. As such the cell was designed with an internal baffle allowing a 50 per cent turndown. The design specification for both duties, plus commissioning information is given in Table 1 and 2. The new slag circuit is shown in Figure 5.

For cleaning of prefloat concentrate a model E2514/3 Jameson Cell was installed. The model designation refers to a rectangular cell, 2.5 m in length × 1.4 m wide and equipped with three downcomers, as shown in Figure 6. A submerged wash water system was included that consisted of a set of perforated pipes that were suspended in the froth zone. The cell was designed with adjustable concentrate lips that allowed the froth cross sectional area to be reduced by 50 per cent to achieve higher superficial air rise velocities. The design specification for prefloat cleaning, plus commissioning information is given in Table 3. The new chalcopyrite circuit is shown in Figure 7.

TABLE 1

Design and commissioning data for the Jameson Cell converter slag cleaner.

Converter slag	Design	Commissioning
New feed solids tph	148 - 188	200 - 252
New slurry flow m^3/hr	255 - 350	300 - 378
Downcomer feed m^3/hr	448	480 - 500
Downcomer feed pulp density kg/l	1.4	1.4 - 1.7
Tailing recycle (%)	40	0 - 30
Air flow m^3/hr	140	115
Jg (cm/sec)	0.43	0.45

TABLE 2

Design and commissioning data for the Jameson Cell RHF slag cleaner.

RHF slag	Design	Commissioning
New feed solids tph	51 - 95	124 - 202
New slurry flow m^3/hr	91 - 168	156 - 311
Downcomer feed m^3 per hr	224	400 - 500
Downcomer feed pulp density kg/l	1.4	1.35 - 1.65
Tailing recycle (%)	50	40 - 61
Air flow m^3/hr	45	110
Jg (cm/sec)	0.36	0.44

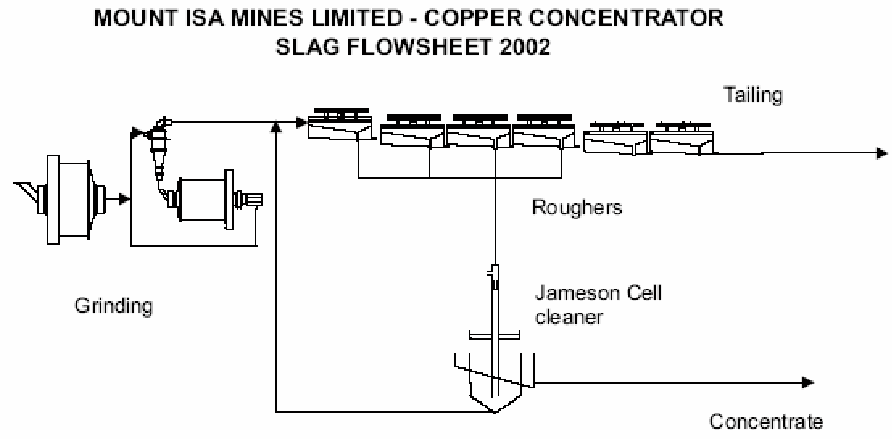

FIG 5 - Post expansion slag circuit.

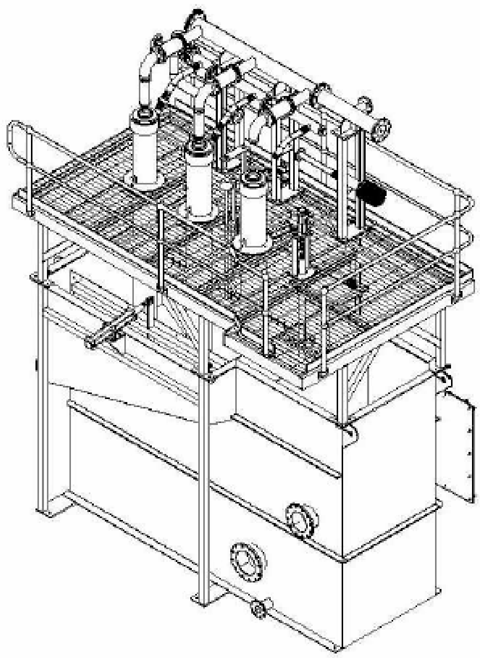

FIG 6 - Prefloat cleaner Jameson Cell.

Both Jameson Cells were fitted with an internal tailing recycle mechanism where a small amount of tailing was directed into the feed to provide constant pump feed and downcomer feed pressures.

A requirement of the installation was to minimise equipment and operator labour to operate the circuit as well as to complete construction and commissioning with a minimum of downtime. The location of the equipment required a design that would require the minimum involvement of the flotation operator during normal operation. For this reason the equipment was located approximately 12 m above ground level to allow gravity to be used to transport tailings to the head of the rougher circuit and for concentrate to travel to the final tailings and concentrate pump boxes. The feed pumps were located on a suspended concrete slab 6m above the ground at the same level as the existing flotation banks.

TABLE 3

Design and commissioning data for the prefloat cleaner Jameson Cell.

Prefloat cleaning	Design	Commissioning
New feed solids tph	54 - 80	5 - 20
New slurry flow m^3/hr	140 - 210	35 - 80
Downcomer feed m^3/hr	224	240 - 250
Downcomer feed pulp density kg/l	1.27	1.2
Tailing recycle (%)	20	64 - 93
Air flow m^3/hr	134	100 - 200
Jg (cm/sec)	0.6 - 1.0	0.9 - 2.3

Instrumentation and control

Both Jameson cells were fully instrumented with airflow control, level control, vacuum and slurry pressure measurement. A redundant 75 KW Variable Frequency drive was used on the slag Jameson cell to avoid delays in construction of a new Motor Control Cabinet and to provide the ability to vary the pump speed when changing from RHF to Converter slag. Two independent tail plugs were used on the slag Jameson cell to allow turndown for RHF operation without loss of level control. Pneumatic cylinders were used to control the angle of the adjustable launder on the prefloat cell as the simplest method of movement. The control room can move the launders over a range of approximately 600 mm, with automatic adjustment to the level calibration for the change in cell geometry undertaken in the plant DCS. Inline samplers of both concentrate streams were installed to allow online measurement of copper grade through the existing Courier 30XP on stream analyser and to provide shift samples for assay. The prefloat concentrate line passes through a U tube before discharging into the final tailings stream to allow flow measurement with a conventional magnetic flow meter.

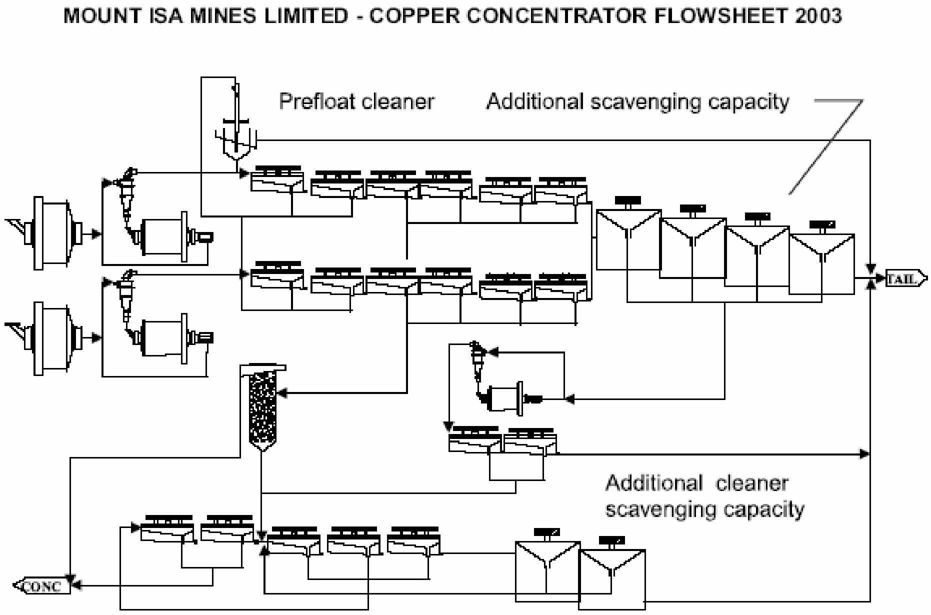

FIG 7 - Mount Isa Copper Concentrator post expansion.

A closed circuit Pan-Zoom-Tilt camera was installed above the cells allowing the control room operator to be able to see the froth surface, pulling rate of the cells and the level in the feed boxes. This has proved to be considerably useful in diagnosing problems with the cell without having to recall the flotation operator from other areas of the plant.

COMMISSIONING

Columns

The flotation column alterations consisted of re-piping feed, tailing and concentrate streams so that they could be operated in parallel, rather than in series. It did not require installation of additional equipment and as such the commissioning of the columns was relatively trouble free.

Slag cleaner

The maximum commissioning tonnage of new feed to the Jameson Cell on converter slag treatment exceeded the maximum design tonnage by some 34 per cent. On RHF slag treatment it exceeded the maximum design tonnage by some 113 per cent. This is due to a number of reasons including higher grinding tonnages and copper content in feed. The initial impact of the higher feed tonnages was high pulp densities in downcomer feed, which appeared to reduce Jameson Cell copper recovery and increase the re-circulating load in the circuit, causing further increases in feed. To minimise losses all six downcomers were operated during treatment of RHF slag. At times during converter slag treatment the tailing recycle mechanism was overloaded with feed bypassing the cell and going directly to tailing.

Air entrainment was found to be detrimentally affected by the high pulp densities. At pulp densities of 1.55 kg/l to 1.7 kg/l air entrainment was only half the design air requirement. Significant improvements in air entrainment were achieved at a pulp density of 1.45 kg/l with design air entrainment or better being achieved at the design pulp density of 1.4 kg/l. To ensure optimum feed densities during commissioning the Jameson Cell feed pump speed was increased by 20 per cent and dilution water was added to the feed box.

Figure 8 shows a comparison of the concentrate grade and recovery achieved following pulp density control with that achieved during the original test work in June 2000. During commissioning a maximum copper recovery of 63.5 per cent at a concentrate grade of 63.8 per cent Cu was achieved. This compares well with the original test series. It should also be noted that during commission numerous mill feed disruptions occurred with feed to the flotation circuit being disrupted for a significant portion of operating time. This resulted in attempts at complete optimisation of parameters and recovery being limited. It was expected that target copper recoveries of 70 per cent were achievable.

During the design phase discussion were held on whether the addition rate of SIBX to the roughers resulted in Jameson Cell instability and whether either reductions in rougher addition rate or a change to another collector would enhance Jameson Cell and overall slag treatment performance. There was some tenuous evidence from original test work that indicated that SIBX addition was detrimental and that increases in SIBX resulted in further deterioration of Jameson Cell performance. Operation during commissioning indicated that high SIBX addition would not affect Jameson Cell stability.

During original test work variation of MIBC had very little effect on vacuum generated and air entrained within the downcomer. During commissioning increased MIBC addition to the Jameson Cell feed was found to have a significant beneficial effect on downcomer vacuum and operating stability. The optimum MIBC addition was approximately 20 ppm MIBC in downcomer feed which is in the upper range of Jameson Cell installations.

Prefloat cleaning

The maximum tonnage of new feed to the Jameson Cell was substantially less than the maximum design tonnage at all times and specifically when only one milling line was treating chalcopyrite ore. This was due to lower than expected grades of prefloat material in run of mine ore. This resulted in levels of tailing recycle above design. Usually increased rates of tailing recycle are considered beneficial to operation as it increases the recovery of the valuable mineral without significant effect on concentrate diluents. This was not necessarily the case with the prefloat Jameson Cell. Although the true flotation talc recovery increased with increasing tailing recycle, true flotation pyrite recovery decreased and above a certain point copper recovery increased. A full understanding of the decline in FeS_2 recovery is not possible without detailed surface analysis of floated and non-floated particles. It is possible that only strongly carbonaceous pyrite is being recovered within the Jameson Cell. Alternatively multiple passes through the downcomer at high tailing recycle rates may have resulted in some passivation of the pyrite surface.

Immediately following start up a number of concerns were raised with the long-term efficiency of the submerged wash water system. It was noted that at a wash water to concentrate water ratio of 0.5, the low velocity water jets exiting the wash water pipes only penetrated the froth to a horizontal depth on 50 mm, compared to a required penetration of 250 mm. This generated sluggish froth zones between the wash water pipes, with low apparent recovery of prefloat concentrate. A number of comparisons with submerged versus 'top of froth' were conducted. These included the insertion of froth tracers in the froth zones and collection of timed lip samples. This allowed froth velocities, volumetric flow and tonnage produced to be evaluated between the two washing systems.

The submerged wash water piping resulted in a 20 per cent decrease in horizontal froth velocity, at a Jg of 1.25 cm/sec, with the two systems only showing equivalent horizontal froth velocities at a Jg of 2.0 cm/sec. The 'on top' wash water achieved a 50 per cent higher flow over the lip and a 50 per cent higher solids mass pull than the submerged wash water system. Although submerged wash water systems were shown to have major benefits in other applications it was considered that the area of wash water piping in the pre-float cell was too high a percentage of the froth area and the system was replaced with wash water trays.

The strength of the Jameson Cell air entrainment would generally indicate that sufficient frother is present to provide acceptable operation. There is however some indication that MIBC may be acting as a weak collector for naturally floating material as well as a frothing agent. A sample of prefloat rougher concentrate was collected and floated under nitrogen in a standard Mount Isa Mines prefloat bench flotation test, with no additional MIBC added. The test was then repeated with additional MIBC.

Plots of true flotation recovery (ie total recovery less quartz recovery) versus time were then generated to determine the effect of additional MIBC. The bench test indicated that additional MIBC addition to the Jameson prefloat cleaner would have only a minimal effect of talc recovery. Figure 9 shows the response of FeS_2. The bench test with no additional MIBC shows a talc recovery after eight minutes of seven per cent, or less than half that achieved on average by the Jameson Cell prefloat cleaner. With additional MIBC the eight-minute FeS_2 recovery increased dramatically to 47.5 per cent with the curve indicating further recovery gains were possible with time. There are a number of possible reasons for the improvement including smaller bubble size, a more stable froth or the MIBC acting as a weak collector, enhancing the flotation of partially carbonaceous pyrite.

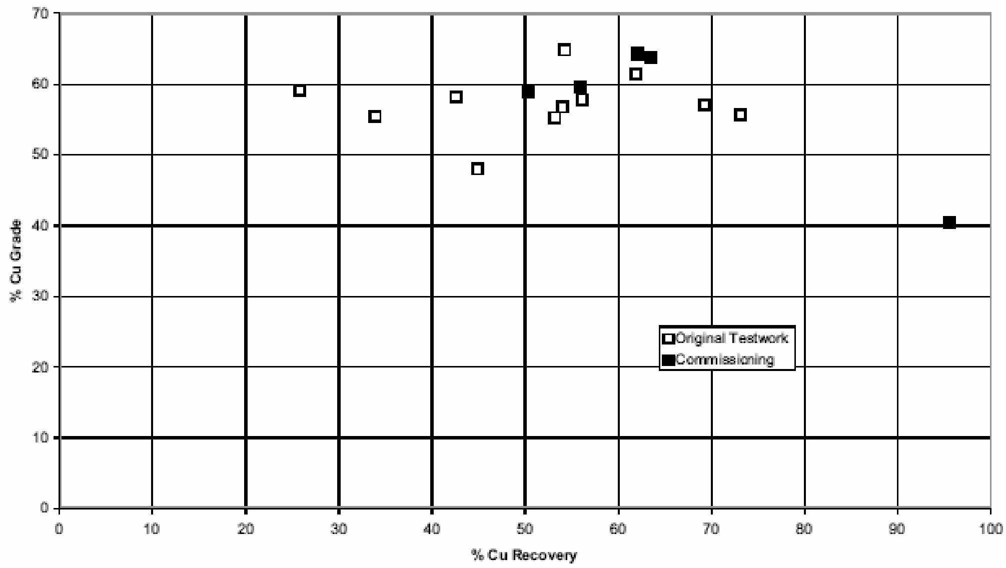

FIG 8 - Converter slag concentrate grade versus copper recovery.

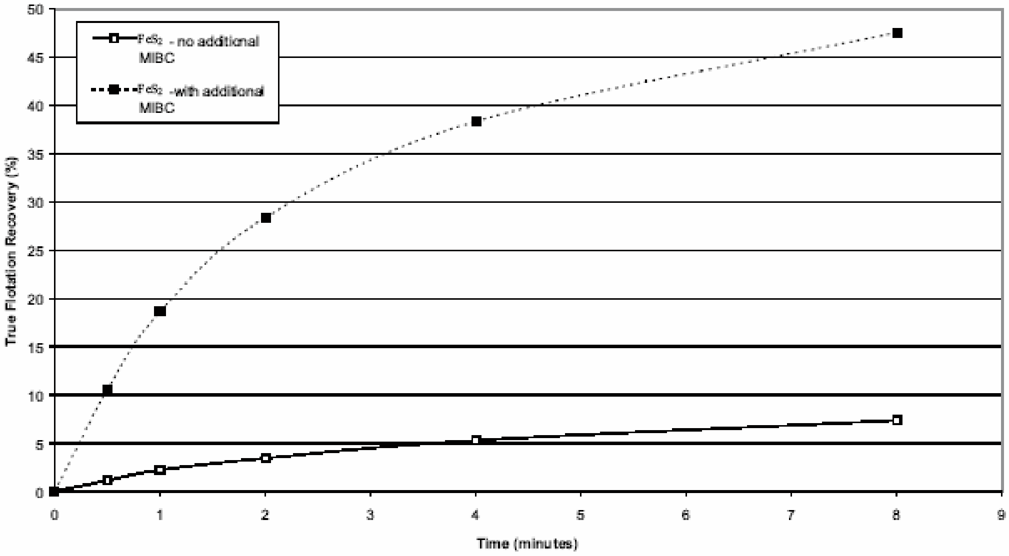

FIG 9 - Flotation of pyrite with and without additional MIBC.

The potential existed that additional MIBC addition to the Jameson prefloat cleaner could have a significant effect on improving FeS_2 recovery. Although additional MIBC is a promising route for increasing FeS_2 recovery there are some concerns. The current level of MIBC entering the Jameson Cell in rougher concentrate appears to be approaching saturation levels. Periodically it was observed that MIBC was causing bubble size within the downcomer to be so small that bubbles could not disengage from the pulp within the tank. This resulted in the tank 'foaming' with no effective froth/pulp interface, loss of level control and severe loss of recovery and concentrate quality. An initial trial of additional MIBC was aborted when this occurred.

Figure 10 shows total talc recovery versus copper rejection from concentrate for the commissioning period, the original test work and for bench scale tests taken during commissioning, respectively. Targeted performance was to achieve a copper rejection of 90 per cent, at a talc recovery of 70 per cent. The upper envelope for commissioning surveys indicates a minor shortfall from this point, with an interpolated copper rejection of 90 per cent at a talc recovery of approximately 67 per cent. At 90 per cent copper rejection talc recoveries of 50 per cent and 47 per cent were achieved in the bench scale tests and original test work respectively.

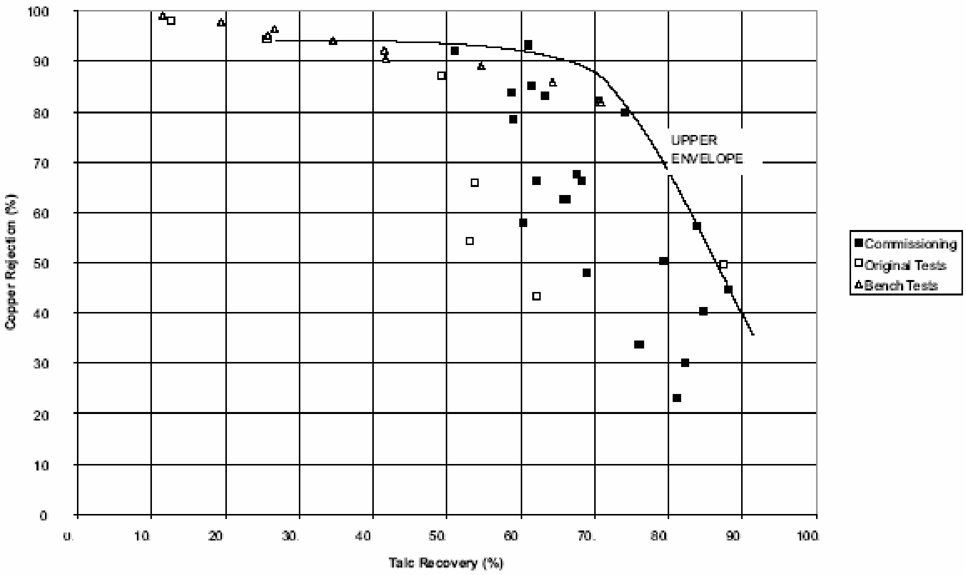

FIG 10 - Talc recovery versus copper rejection.

Figure 11 shows total FeS_2 recovery versus copper rejection from concentrate for the commissioning period, the original test work and for bench scale tests taken during commissioning, respectively. Targeted performance was to achieve a copper rejection of 90 per cent, at an FeS_2 recovery of 50 per cent. The commissioning surveys indicate a significant shortfall from this point, with a copper rejection of 90 per cent at an FeS_2 recovery of only 17 per cent. At 90 per cent copper rejection FeS_2 recoveries of 30 per cent and 28 per cent were achieved in the bench scale tests and original tests respectively. The commissioning results show a strong linear relationship between copper rejection and FeS_2 recovery indicating some degree of carbonaceous chalcopyrite being present and also FeS_2 recovery gains being due to entrainment rather than true flotation. There is some initial evidence that carbonaceous pyrite that did not float within the Jameson Cell has not been refloated in the prefloat roughers and does not report to final concentrate.

POST COMMISSIONING

To provide fewer fluctuations in feed tonnage to the slag cleaner blending of converter and RHF slag was commenced. This has resulted in a major reduction in the per cent solids entering the Jameson Cell and has produced very stable operation. A number of changes to the reagent addition were also trialed. All MIBC is added to the Jameson cell rather than the flotation feed to improve the air entrainment. Sufficient frother remains in the Jameson Cell tailing to meet rougher frother requirements. SIBX addition is used in the roughers to control rougher recovery. Additions greater than 200 g/t SIBX are used regardless of head grade whilst dilution water and wash water are used to control Jameson concentrate grade.

The prefloat cleaner was successfully refitted with wash water trays, which resulted in more effective rejection of entrained chalcopyrite and removal of talc.

The original butterfly valves supplied for commissioning have been replaced with V-notch knife gate valves, which have provided substantially better airflow control. The improved flow characteristics and pressure drop on the new valves has allowed the removal of substantial signal damping required for the air flow measurement and provided more consistent air flow control.

The expansion of the Copper Concentrator has been multifaceted and to quantify the benefit of one circuit change in isolation to the others is difficult. A number of qualitative benefits were rapidly evident. The dedication of the flotation columns for chalcopyrite cleaning only has resulted in a major decrease in slag contamination, allowing significantly more stable chalcopyrite flotation. The Jameson Cell slag cleaner operates with a tank residence time of only 3.5 minutes, compared to sixty minutes for the columns in series. The small volume of the Jameson Cell has allowed it to be rapidly stabilised during start-ups and following inadvertent shutdowns, releasing operators for other duties. Dumping of the cell, if required also results in much less spillage and cross contamination. The Copper Concentrator has been operating almost exclusively on the parallel milling circuit since December 2000 with the target of eliminating the slag stockpile. The prefloat Jameson Cell has reduced pyrite in concentrate and allowed the concentrator to handle increased talc head grades without increasing the copper loss, Figure 12. Run of mine talc head grades have been amongst the highest ever recorded, yet concentrate grades are still in excess of 25 per cent Cu.

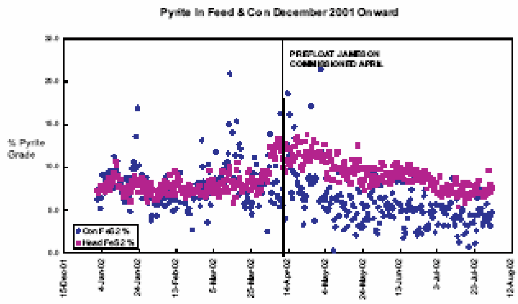

FIG 12 - Pre and post expansion pyrite in feed and copper concentrate.

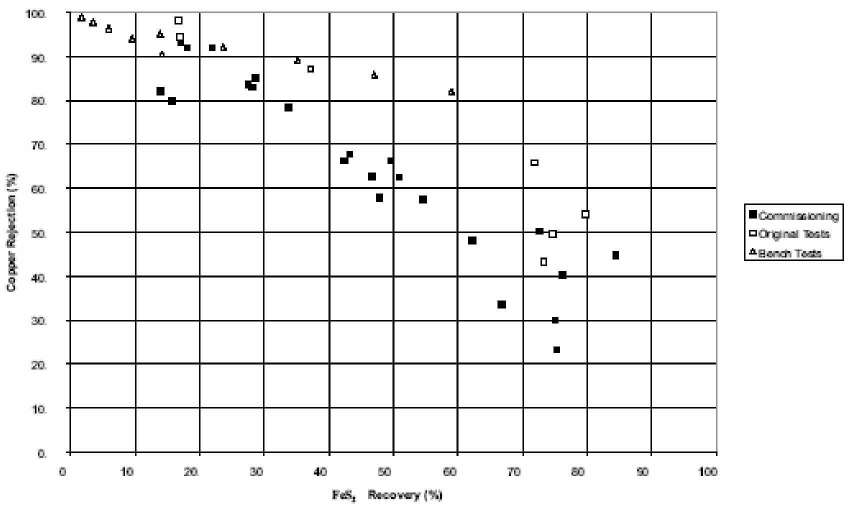

FIG 11 - Pyrite rejection versus copper recovery.

CONCLUSIONS

The slag Jameson Cell has allowed the Copper Concentrator to operate on the parallel milling circuit for extended slag campaigns due to the lower risk of cross contamination and the release of the columns back to the chalcopyrite circuit allowing higher ore treatment without loss in recovery. Also Jameson Cells have proved to be effective in their design role of copper recovery/rejection with a minimum operator involvement to control and very forgiving with variations in feed rate due to the nature of the internal tailing recycle. The prefloat Jameson Cell has reduced pyrite in concentrate and allowed the concentrator to handle increased talc head grades without increasing the copper loss.

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REFERENCES

Barnes, C D, Lumsdaine, I, O'Hare, S M, 1993. Copper converter slag treatment at Mount Isa Mines Limited, Mount Isa, Qld, *The AusIMM Proceedings*, 1993(1):31-37.

Bartrum, J, Schache, I S, Watsford, R M S, Slaughter, P J, 1978. Converter slag flotation practice at Mount Isa Mines Limited, in *Proceedings Mill Operators' Conference*, pp 113-120 (The Australasian Institute of Mining and Metallurgy: Melbourne).

Carr, D, De Bono, S, Harbort, G J, Lawson V, 2003. Jameson Cell fundamentals: a revised perspective. Presented at *Flotation 03*, Helsinki, March 2003.

Dawson, W J, Harbort, G J , 1996. Jameson Cell recovery of copper from converter slag, in *Proceedings of Hidden Wealth – Unlocking values from low grade and refractory ores and wastes*, Minerals Engineering International.

Evans, G M, Atkinson, B W, Jameson G J, 1995. The Jameson Cell, *Flotation Sci Eng*, 1995:331-363.

Grano, S, Ralston, J, Smart, R St C, 1990. Influence of electrochemical environment on the flotation behavior of Mt Isa copper and lead-zinc ore, *International Journal of Mineral Processing*, 30 (1990), pp 67-97.

Griffin, L K, 1993. Results of the Second Extended Preflotation Plant Trial in the Copper Concentrator, Mount Isa Mines Technical Memorandum, MIM.

Harbort, G J, Manlapig, E V, DeBono, S K, 2003. A discussion of particle collection within the Jameson Cell downcomer, *IMM Trans / AusIMM Proceedings, Section C: Mineral Process Extr Metall, January/April*, pp C1-C10.

Hoffmann, C W, 1965. A review of developments in flotation treatment of Mount Isa chalcopyrite ores, in *Proceedings of 8th Commonwealth Mining and Metallurgy Congress Australasia*, 1965, Paper no 103, pp 1133-1149.

Jameson, G J, 1988. A new concept in flotation column design, *Column'88 – Proceedings of an International Symposium on Column Flotation* (Ed: K V Sastry) pp 281-289 (SME).

Jameson, G J, Belk, M, Johnson, N W, Espinosa-Gomez, R, Andreaditis, J P, 1988. Mineral flotation in a high intensity column, in *Proceedings Chemeca 88, 16th Australian Conference on Chemical Engineering*, Sydney, pp 507-510.

Lumsdaine, I, O'Hare, S M, 1980. Copper concentrator practice at Mount Isa Mines Limited, Mount Isa, Qld, in *Mining and Metallurgical Practices in Australasia* (Ed: J T Woodcock) pp 649-653 (The Australasian Institute of Mining and Metallurgy: Melbourne).

Lyon, G C, Fewings, J H, 1971. Control of the flotation rate of the carbonaceous material in Mount Isa chalcopyrite ore, *The AusIMM Proceedings*, No 237, March 1971.