Flotation data for the design of process plants Part 2 – case studies

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The Cadia Hill Gold Mine and the Hellyer copper, zinc and lead plant present different problems with widely different target grind sizes, liberation characteristics. The contrast is between Hellyer's sequential flotation circuit versus Cadia's 'simple' copper–gold flotation circuit. The common theme is that pyrite forms the principal floatable concentrate diluent. Hellyer ore contains finely disseminated chalcopyrite, sphalerite, galena and tetrahedrite. The flotation plant design was based on extensive benchscale test work (including locked cycle tests) on drill core and many months of operation of a 30 t h^{-1} 'pilot plant' using the modified Cleveland Tin Mine processing plant. The Cadia concentrator was designed based on an extensive benchscale variability test work programme. Data from approximately a dozen locked cycle tests conducted on drill core and 2 weeks of continuous pilot plant trials of samples obtained from an adit into the orebody were used as the basis of design. The orebody is a low grade monzonite porphyry with disseminated chalcopyrite/bornite/pyrite (0·17%Cu, 0·73 g t^{-1} Au). Methods used to establish the appropriated flotation circuits for the treatment of the Hellyer and Cadia ores, as well as the interpretation of the test work data for these particular operations, are discussed. The predicted versus actual operational plant is reviewed and the comparison was found to be satisfactory.

Keywords: Locked cycle tests, Residence time, Finely disseminated, Flotation circuit design, Hellyer Project, Cadia Project

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Introduction

The two case studies described in this paper outline the following stages of design: initial benchscale flotation test work to evaluate the orebody, pilot tests using mined ore bulk samples and final design and operation of the selected flowsheet.

In both case studies, the main characteristics of variable mineralogy, flotation kinetics and orebody representation posed some challenges. For Hellyer, the slow floating nature of the minerals, the fine particle size and the number of valuable minerals (lead, zinc, copper and silver) to be recovered meant that a complex flowsheet resulted. At the Hellyer site, use was made of an existing tin flotation plant and various flotation cell types were tested in a large scale 'pilot plant'.

The Cadia Hill ore was tested at laboratory scale and validated in a pilot plant using four mined ore samples.

Both of the final flowsheet made use of regrind stages to allow for recovery of the fine minerals and in the case of Hellyer, this allowed the recovery of each valuable mineral to separate copper, lead and zinc concentrates, with a final 'catch all' bulk concentrate to collect lead and zinc that failed to report to their respective primary concentrates. Typical regrind particle 80% passing (P80) sizes were 38 μm for Cadia and 10–25 μm for Hellyer.

Project optimisation work was aimed at understanding how mineralogical variation impacted on the metallurgical performance of the Hellyer ore. Similar work is continuing at Cadia.

Case study one: a complex copper, lead and zinc flotation circuit (the Hellyer Project)

The Hellyer Project was developed by Aberfoyle Limited and proceeded from discovery in 1983 to production in 6 years.[1] The 15×10^6 t orebody consisted of fine grained massive sulphide containing copper, lead, zinc and silver minerals with gold in a predominantly pyrite matrix.

Sulphide mineralisation was intersected in 1983. Preliminary metallurgical test work commenced mid-1984 and an adit to provide access to the orebody was initiated in 1985. Laboratory batch test work providing two options for further evaluation was completed by the end of 1985. Pilot plant test work on underground samples commenced in June 1986 and detailed design for the Hellyer concentrator started in March 1987. Plant construction began in January 1988 and the first flotation concentrate from the initial 10^6 t/year concentrator was produced in March 1989.[2,3]

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Mineralogy

The mineralogy of the Hellyer ore is complex and the orebody was classified in several categories on the basis of texture, typically associated with distance from the vent zone. The sulphide mineralisation was unaltered leading to the very fine grain size and high level of sulphide intergrowth and association. Grain size generally increased and pyrite associated decreased towards the top of the orebody, and the barite cap. Gold was principally associated with arsenopyrite and pyrite with minor electrum. Thus, arsenopyrite was predominantly associated with pyrite with minor associations with sphalerite and galena. Chalcopyrite was strongly associated with sphalerite and often included in sphalerite grains. Silver occurred as tetrahedrite, in solid solution in galena and very fine grains in pyrite. The typical occurrence of minerals is shown in Table 1.

Principal conclusions from the early mineralogy[4] were that a fine grind was required for sulphide liberation, although even with fine grinding it would be difficult to achieve high concentrate grades and recoveries. Misplaced minerals would lead to loss of payable metals and gold would be upgraded to a refractory arsenopyrite/pyrite flotation tailings. Mineralogical variation within the orebody would probably result in only minor metallurgical differences.

Benchscale flotation test work

Benchscale batch flotation test work and mineralogical investigations were conducted at the Aberfoyle laboratory in Tasmania and Amdel in Adelaide. Preliminary roughing tests were performed on drill core composites. This work showed that very fine grinding, of approximately 40 μm, was required to obtain reasonable recoveries. Unfortunately, cleaning of the rougher concentrate failed to produce acceptable concentrate grades. Further testing was carried out at a laboratory in Canada where an acceptable metallurgical reagent regime was developed. A key step to this was the use of a laboratory mild steel mill for grinding rather than a conventional stainless steel mill. An extensive flotation test work programme followed at the Aberfoyle laboratory in Tasmania. At the completion of this programme two flowsheets producing either one bulk or four separate concentrates, were proposed for flotation locked cycle test work.

The bulk flotation flowsheet was aimed at generating a combined or bulk lead, zinc, silver, gold and copper product for processing in Imperial Smelter Furnace smelters. The four product flowsheet produced individual products of copper–silver, lead, zinc and bulk concentrates. The purpose of the two flowsheets related to the proposed development of the Hellyer orebody. The four product flowsheet was designed to be implemented at the new Hellyer concentrator to be built beside the mine. However, mine development and concentrator design and construction would take a number of years. It was decided to convert the nearby Luina tin concentrator in the interim to produce bulk concentrate at a rate equivalent to the future Hellyer concentrator bulk production rate. This had the advantage of retaining concentrator workforce from Luina, as the tin reserves were exhausted, as well as establishing a foothold in the variable bulk concentrate market. It also allowed further metallurgical development of a difficult ore.

Table 1 Hellyer mineralogy

Mineral	Occurrence, %
Pyrite	40–70
Sphalerite	15–25
Galena	6–12
Arsenopyrite	1–3
Chalcopyrite	0·8–2
Tetrahedrite	0·1
Non-sulphides	5–25

The key conclusions from the benchscale test work were that the minerals were slow floating and long residence times were required to maximise selectivity and recovery and a regrind of the lead and zinc rougher concentrates to finer than 20 μm was required to optimise performance.

The Hellyer ore benchscale process flowsheet was similar to that of other copper, lead and zinc concentrators. However, the metallurgy was considered sufficiently difficult due the fine particle liberation sizes and the slow flotation response to warrant pilot plant operation. An attempt to use the common North American practice of combined copper–silver–lead roughing with subsequent separation of copper–silver and lead was not successful. In particular a satisfactory lead concentrate grade could not be achieved.

Locked cycle batch flotation test work

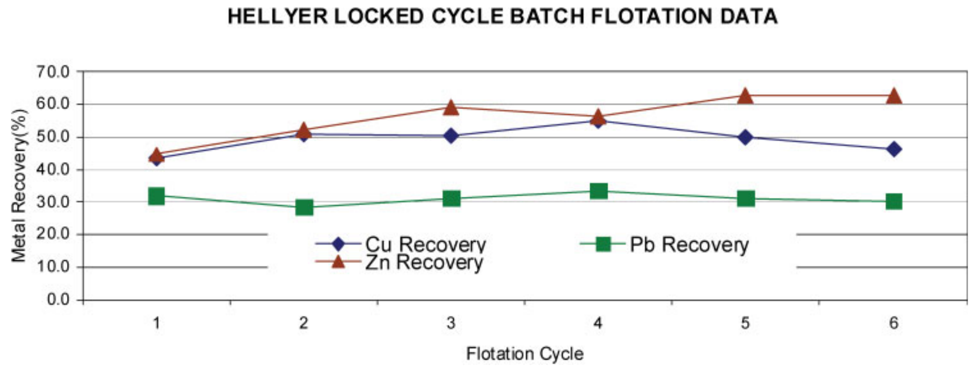
Locked cycle flotation tests on the four product sequential circuit confirmed that retaining production of bulk concentrate permitted higher grade separate lead and zinc concentrates. This enhanced economic returns and greatly increased circuit stability. The locked flotation tests also confirmed the slow flotation rates and the deleterious effects of recirculating loads. It was found beneficial to keep the lead and zinc roughers in open circuit to minimise retention times and maintain circuit stability. The flowsheet development philosophy was for a simple, conventional reagent regime in order to speed development. Reagent optimisation would be a continuing part of plant operations. Depressants such as sulphur dioxide were discarded for the easier to handle sodium metabisulphite.

High circulating loads in locked cycle test work for lead and zinc cleaning resulted in a requirement to produce a bulk lead/zinc concentrate from the lead and zinc cleaner tailings to enable the production of saleable grade concentrates. The reagent scheme required was typical of other base metal concentrators. Typical locked cycle test data showing recoveries and grades are provided in Figs. 1 and 2.

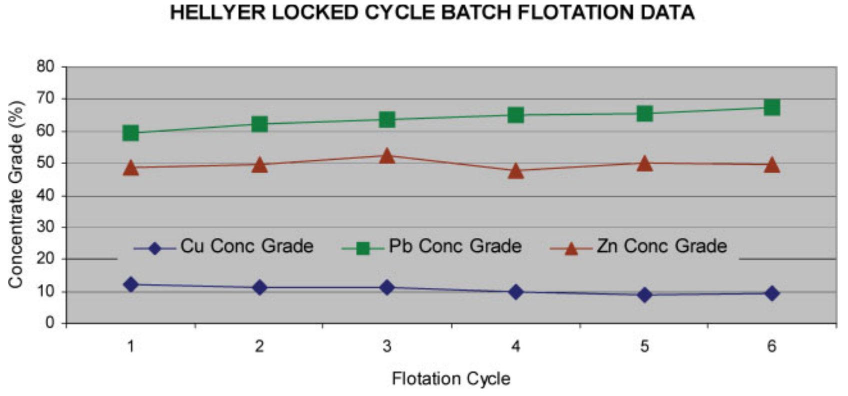
Even though there were only six cycles, the flowsheet for the processes was complex and time consuming. For the six cycles there are 27 rougher floats, 6 scavenger floats, 48 cleaner floats and 18 regrind steps. One cycle takes 280 min to complete and there are 312 reagent addition steps. The documented predicted grades and recoveries from the locked test are shown below in Table 2.

Pilot plant

Aberfoyle operated the Cleveland Tin Mine and processing plant at Luina, 50 km North West of Hellyer. In the mid-1980s, the dramatic fall in the tin



1 **Hellyer recoveries**



2 **Hellyer concentrate grades**

price resulted in closure of the tin mine that paralleled the development of Hellyer. Aberfoyle decided to use the Cleveland facility to pilot the bulk and sequential flotation flowsheets developed at benchscale. The decision to conduct the 6 to 10 $t\ h^{-1}$ pilot trials at Cleveland over the typical 0·25 to 1 $t\ h^{-1}$ pilot plant at facilities like Amdel was taken due to its proximity to Hellyer and because this plant became available.

The principal difference between the benchscale test work and the pilot plant operation was the increased quantity of less than 7 μm fines produced in the pilot plant. At benchscale only 22% of the mass was less than 7 μm for a grind of 40 μm. At pilot scale this increased to as much as 50% less than 7 μm and was typically 30–35% less than 7 μm in piloting and during later Hellyer plant operation. The greatest impact of this change in size distribution on flotation was seen in the increase in fine galena (relatively soft and high relative density) and the slower lead flotation response. Reagent additions increased marginally due to the higher fines content. The slow flotation rates required low pulling rates (aeration rate) in flotation to maximise the selectivity of the valuable minerals over pyrite. Pilot plant flotation times were typically at least double that observed in the benchscale test work. A total of 15 000 t ore was treated through the pilot plant before an upgrade of the Cleveland plant. Major ore textures were tested with few significant differences in performance observed.

At the successful completion of the pilot plant trial, the Cleveland plant was upgraded to 30 $t\ h^{-1}$ and operated producing a bulk concentrate while the Hellyer plant was designed and constructed. During this period, flotation cells and impeller types were evaluated focusing on the particularly slow flotation rates and low air requirements. In addition, column flotation and Jameson cells were evaluated in roughing and cleaning duties due to their increasing application in other flotation plants.[5] This work led to the installation of 50 m^3 rougher and 30 m^3 cleaner tank cells fitted with low power input flotation impellers in the majority of flotation circuit. A small flotation column fitted with a Jetflote (Jameson cell) downcomer feed system was installed for copper concentrate cleaning.[6]

Metallurgical recovery predictions

Predicted metallurgical recoveries for Hellyer were determined from operating results obtained from the pilot plant. Grade–recoveries from all stable periods of operation of two or more shifts were plotted and recoveries selected for the circuits were determined from detailed surveys conducted during the pilot plant operations.

It had been anticipated from laboratory work that recoveries would vary with mineralogical texture and a number of these textures were tested in the pilot plant. However, the metallurgical performance for all textures tested was similar so a constant grade–recovery was attributed to the whole of orebody. It was recognised that performance would vary with head grade. As the head grade in the mine schedule was reasonably

Table 2 Predicted grades and recoveries from locked test

	Copper	Lead	Zinc
Grade, %	10·3	63·6	49
Recovery, %	48	31	62

constant for the first 5 years no allowance was made for head grade variation in the predicted grade–recovery relationship. It was anticipated that improvements in metallurgy over time would offset any deterioration of performance as head grade decreased near the end of the life of Hellyer.

Design criteria and description of the Hellyer flotation circuit

The Hellyer plant has a complex flotation flowsheet with four distinct final concentrates being produced and multiple cleaning and regrinding stages.

Flotation feed was conditioned with reagents and then fed to the head of the first copper–silver rougher cells. Concentrate generated from the first three Maxwell MX14 (50 m^3) roughers was fed to a 10 m (height) × 0·9 m (diameter) column cell to produce a final silver rich copper concentrate containing 12–16% copper. Column cell tailings were returned to the conditioning tank ahead of the copper–silver rougher cells.

The tailing from the copper–silver roughers was conditioned with lime to increase the pH before adding cyanide prior to the lead roughing circuit. Flotation collector was stage added to each of the seven Maxwell MX14 lead rougher cells. The lead rougher concentrate fed a 335 kW Kubota Tower Mill and the product was further cleaned in a three stage cleaning circuit using Maxwell MX12 (32 m^3) cells. The first cleaner tailing reports to the bulk flotation circuit.

The tailing from the lead roughers was conditioned with copper sulphate and lime, before zinc rougher flotation. Collector was again stage added to each of the eight MX14 Maxwell zinc rougher and scavenger cells. Additional copper sulphate and lime was added midway through scavenging. The zinc scavenger tailing, containing less than 1% zinc, was the final tailings. The zinc scavenger concentrate reported to the zinc regrind circuit. This stream reported to a Kubota Tower Mill in closed circuit with a second set of cyclones. This stream was later preclassified before regrind due to differences in flotation performance caused by the regrind environment.[7] The regrinding improved liberation of sphalerite from the pyrites gangue. The reground material was reconditioned with copper sulphate, lime and collector. This stream reported to the first of three zinc cleaning stages. Lime was added to the first and third zinc cleaning stages to raise the pH. The zinc concentrate produced from the third cleaner contained 50% zinc with a recovery of 63–68% (but as low as 45% during initial operation).

Before the bulk rougher stage, the combined lead and zinc first cleaner tailings were reground in the bulk regrind circuit consisting of cyclones enclosed circuit with a 335 kW Kubota Tower Mill. This regrind step was necessary to reduce the number of composite sphalerite–pyrite and galena–pyrite particles to enable more selective flotation. The product size was about 20–25 μm and was reconditioned with copper sulphate. Lime was added to the bulk roughers and collector added to each of the six bulk rougher cells. The bulk rougher concentrate reported to the first of two bulk cleaning stages. The bulk scavenger tailing reported with the zinc scavenger tailing to the final tailings sump and was then pumped to the tailings dam. The final bulk concentrate contained approximately 35% zinc and 15% lead.

The pilot plant residence times were typically twice the benchscale residence times. For the Hellyer plant operation, the residence time was three times that required at bench scale. The slow flotation in the plant was linked to high levels of surface passivation (typically metal hydroxides) but this was not known until after the Hellyer concentrator had been commissioned. The passivation resulted in particles that were slow to further oxidise as noted in the early benchscale test work.

The Hellyer schematic flowsheet is shown in Fig. 3 (lead regrind not included).

Some relevant Hellyer flotation operating data are shown in Table 3.

Plant operation and metallurgical development

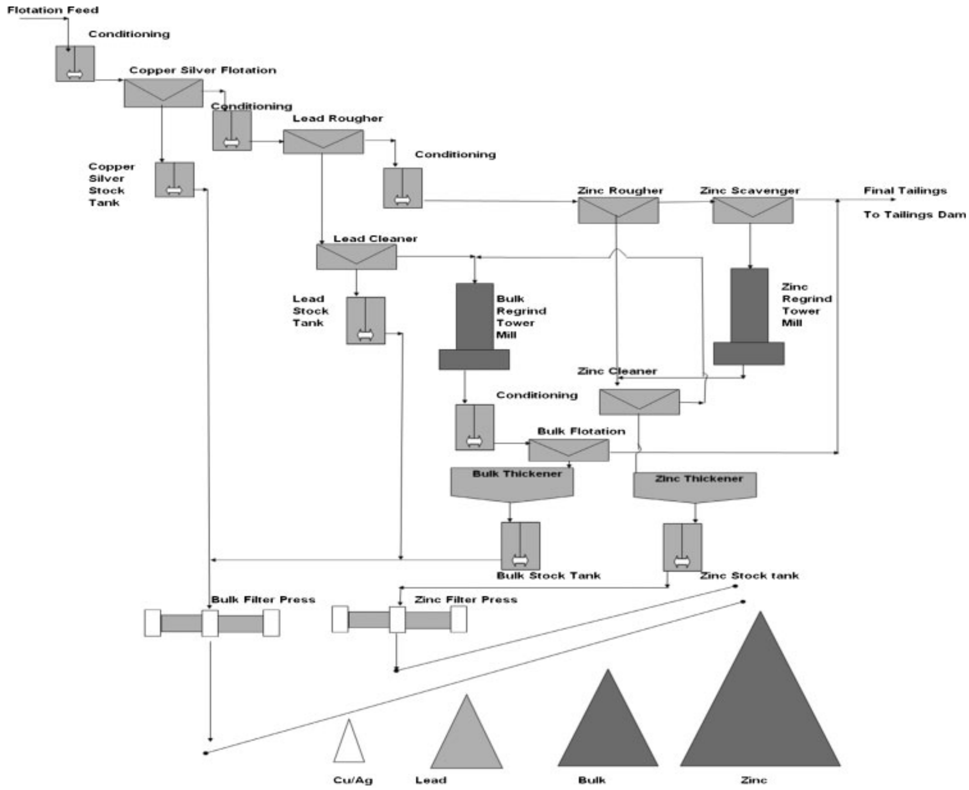
During the operation of the Hellyer concentrator the variability of grade–recovery relationship with ore type became apparent. Metal ratios, particularly zinc/iron, were a strong determinant on recoveries. It was then recognised that the initial trialling of textures in the pilot plant was flawed because by coincidence they had all been tested at a near identical metal ratio.[8] Correlations were then developed initially between plant performance and QEM*SEM phase specific surface areas measurements and later with metal ratios. This then allowed the prediction of metallurgical performance for each mining block in the orebody.

The significant impact of mineral grain size and associations on metallurgical performance was clearly demonstrated in the latter years of Hellyer. Statistical analysis of optical mineralogical data showed a close correlation between mineral associations and zinc recovery. This is illustrated in Fig. 4.

It is also important to realise the significance of simpler measurements such as metal ratios. These are more readily available than mineralogical data and can be very useful. At Hellyer the average zinc/iron ratio for the orebody was 0·6, but the initial pilot plant work was performed on ores with zinc/iron ratios of 0·7–0·8. These ores floated well and led to the erroneous conclusion that there was little difference between ore types. The later treatment in the Hellyer concentrator of ores of zinc/iron ratio equal to 0·4 showed zinc recoveries could be as much as 20% lower due to the increased sphalerite/pyrite associations.

Subsequent investigations[9] found that flotation residence times were strongly affected by mineral oxidation. Galvanic actions occurred between the galena and the pyrite producing lead 'hydroxides' that coated all minerals. These lead 'hydroxides' reduced the flotation rate of galena and increased the flotation rate of sphalerite and pyrite. Selectivity was adversely affected and differential separation only possible at very low flotation rates. This effect was exacerbated by increased fines, so it was more evident in the continuous grinding circuits than at benchscale.

The mineral oxidation occurred steadily over the first 30–60 min flotation before reaching a plateau. The Cleveland 30 t h^{-1} operation which floated lead immediately after grinding experienced less oxidation than the Hellyer circuit which used 30 min copper flotation before the lead circuit. This partly explains the increased residence time needed in the Hellyer circuit over the 30 t h^{-1} Cleveland plant.



3 Hellyer concentrator flowsheet flotation circuit (circa 1990)

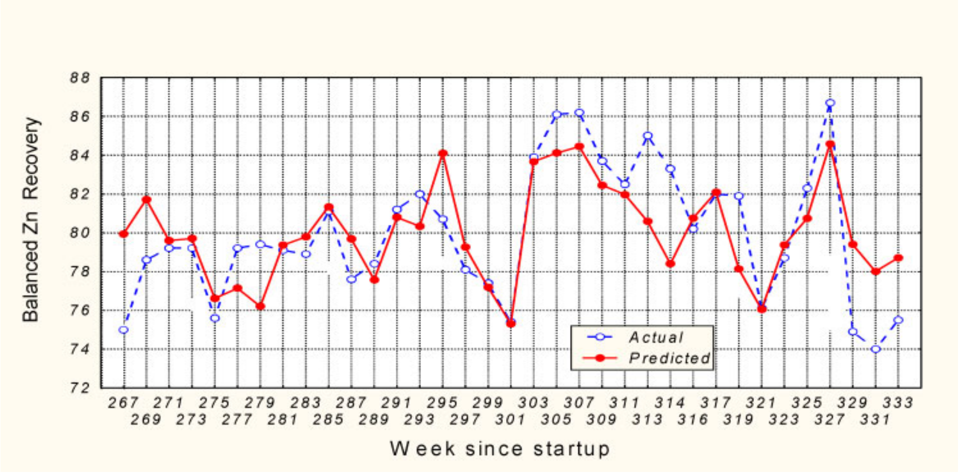
Flotation selectivity and recovery were enhanced in the laboratory and plant through the use of surface cleaning agents (e.g. ethylenediaminetetraacetic acid) or through the use of high shear rates in high intensity conditioning in flotation conditioning. Improved plant performance was achieved through a combination of adding lime to the grinding circuit and limiting oxidation, attritioning with silica sand in lead conditioning and using high power intensities in cleaner circuit feed conditioning.

In retrospect, the selection of low power intensity flotation cells may also have been detrimental to flotation performance, particularly if flotation conditions in grinding and pre-flotation are optimised.

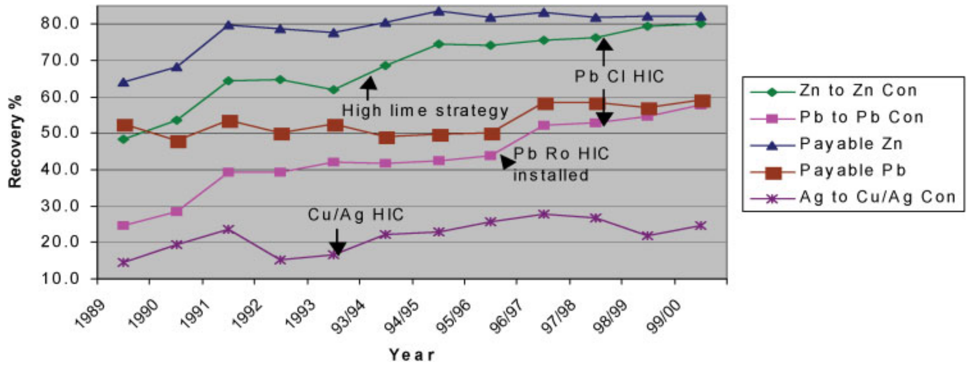
The impact of some of the key process changes, as discussed in the previous section, on metallurgical performance are shown in Fig. 5. The combination of the use of high intensity conditioning in the copper and lead circuits and increased lime addition to the grinding circuit to inhibit oxidation greatly improved selectivity

Table 3 Hellyer flotation plant operating data

Item	Feed	Rougher	Scavengers	Cleaner	Recleaner	3rd cleaner
Copper float						
Feedrate, t h^{-1}	120–156					
Concentrate, % solid	30–40	10–30		35–45		
Residence time, min		32		55		
Grade Cu, %	0·4	3		12		
Lead float						
Feedrate, t h^{-1}	120–156					
Concentrate, % solid	30–40	30–40	30–40	40–50	40–50	40–50
Residence time, min		74	74	52	29	28
Grade Pb, %	6·5	25	12	35	50	60
Zinc circuit						
Feedrate, t h^{-1}	75–115					
Concentrate, % solid	30–40	30–40		30–40	30–40	30–40
Residence time, min		84		57	40	34
Grade Zn, %	14	30–35		45	48	50
Bulk float						
Feed rate, t h^{-1}	40–60					
Concentrate, % solid	10–20	40–50		40–50	40–50	
Residence time min		69		75	76	
Grade Pb, %	10	14		18	30	
Grade Zn, %	16	29		25	76	



4 Hellyer concentrator: modelling Zn recovery from optical mineralogy parameters



5 Hellyer concentrator metal recoveries (1989–2000)

and flotation kinetics due to modification of the surface chemistry of the sulphide minerals.

Case study two: a simple copper–gold flotation circuit (the Cadia Hill Project)

The Cadia Mine is located 25 km southwest of Orange in the central west of New South Wales. Cadia has a history of mining and exploration dating back to 1851 and shortly after the first Australian gold rush at Ophir, north of Orange.[10]

Exploration commenced at Cadia Hill in 1992 and this identified a substantial zone of copper and gold mineralisation. Subsequent resource definition drilling defined an orebody with a mineable reserve of 200×10^6 t ore with an average gold grade of 0·74 g t^{-1} and 0·17% copper. A full feasibility study commenced in 1994, approvals to proceed with development were granted in September 1996 and the processing plant was commissioned in July 1998.

Mineralogy

The mineralisation of economic interest consists of native gold and chalcopyrite with lesser bornite located within or disseminated near quartz veins mostly within the quartz porphyry. Most veins are 1 to 20 mm thick with a vein density of 2 to 5 m^{-1}. These rarely occupy more than 5% of the rock mass.

Initial diamond core drill spacing was on 100×150 m grid. This was followed by both 100×100 m and 50×50 m patterns to obtain a resource estimate classifiable as Inferred and Indicated.

Drill core was cut in half and one metre portions submitted for gold and copper assays. Composited assay pulp samples, contained within the mineralised zone, were analysed for sulphur, cyanide soluble copper and certain smelter penalty elements (mainly mercury).

A mineral assemblage classification was developed for the Cadia ore. The classification is based on Cu/S ratios and the amount of cyanide solution copper and provides a classification of the sulphides as shown in Table 4.

Applying this classification to the geological block models and cumulating the tonnage of each ore type provides the department of the different mineral types within the deposit.

Benchscale flotation test work

The flotation programme involved reagent scoping studies, ore variability test work, a pilot plant campaign and orebody mineral (metal) recovery modelling.

The flotation reagent scoping study was undertaken in three phases. Initial 'in-house' flotation test work by

Table 4 Mineral assemblage classification system adapted for Cadia

Cu/S ratio	S, %	CNSolCu, %	Interpreted sulphides
<0·5	>0·01	<20	Pyrite+chalcopyrite
<0·5	>0·01	>20	Pyrite+bornite
0·5–1·5	>0·01	<20	Chalcopyrite+pyrite
0·5–1·5	>0·01	>20	Chalcopyrite+bornite
<1·5	>0·01	All	Bornite
All	<0·01	All	Oxide

Newcrest indicated that a grind size of ~150 μm was near optimal. The 'in-house' procedure, similar to that used at Newcrest's Telfer gold mine, incorporated a selective flotation regime to produce a copper–gold concentrate and a pyrite–gold concentrate. It was found also that regrinding of the rougher concentrate had benefit in improving copper concentrate grade by ~2%.

Drill core samples crushed to −2 mm and characterised mineralogically were sent to four vendors of flotation reagents for collector evaluation. Data from the 'standard' flotation test were provided in the data package sent to the vendors. The vendors were requested to evaluate collectors for the copper–gold float with a preference for the float to be conducted at a pH below 10 because of pulp viscosity problems and the potential to reduce lime costs. For the pyrite float a combination of the collectors PAX and A208 were selected.

The vendors were allowed two months to carry out in-house reagent testing before nominating their preferred flotation collector for evaluation. The appropriate flotation procedure and collector were then sent to the Lakefield Laboratory, in Canada, who on the behalf of Newcrest carried out four flotation tests for each vendor with the purpose of optimising flotation performance (grade, recovery, reagent quantities). At the completion of each test, the vendor was supplied with metallurgical results and after reviewing these nominated the next changes to the flotation procedure. Interestingly and to some degree not unexpectedly, all vendors were able to produce similar copper–gold flotation results. However, the operating conditions, especially pH, varied considerably.

After reviewing the Lakefield test results, it was decided, on the basis of reagent costs and flowsheet simplicity, to carry out further evaluations of the copper float on only two of the collectors tested, i.e. S701, an ethylthiooctane base collector, and RTD11A, a thionocarbamate base collector. The next phase of the flotation programme focused on the 'robustness' of these two collectors. For this work, 34 different drill core samples were selected having both varying mineralogy and head grades of copper and gold. During the variability test programme, the collector addition was kept constant while some latitude, on the part of the flotation operator, was allowed to vary the frother addition. The results of this programme clearly showed S701 to be more 'robust' in providing acceptable copper grades and recoveries of gold and copper for a wide range of mineralogy and head grades. Consequently, S701 was selected as the collector for all further laboratory and pilot plant test work.

Locked cycle test work

The aim of the locked cycle test work was to evaluate and reinforce the metallurgical recovery models recovery for developed for Cadia Hill. The models were based on the correlation of assay head grade and other physical factors with laboratory batch and pilot plant concentrate grades and recoveries. The head samples used for the flotation test work were composited from the core samples used to conduct the laboratory batch tests for model construction. A batch test was also conducted on the composite sample for comparison.

The locked cycle test results compared favourably with the batch rougher test result. Most locked cycle tests showed some instability, with gold building up in the copper circuit circulating loads. The bulk flotation test conducted on the bornite ore was very stable in terms of copper flotation, but still retained gold in the recycle streams. A typical result is shown in Fig. 6.

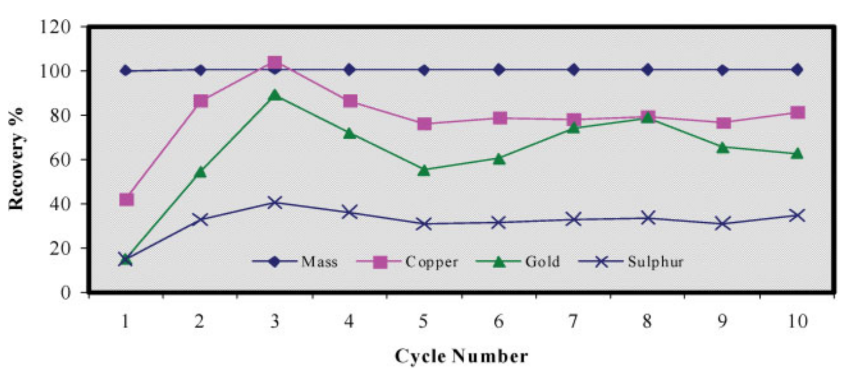
The locked cycle test programme for Cadia was simpler than that for Hellyer. The total time involved in completing a single cycle was 50 min. Only 20 rougher floats and 20 recleaner floats were involved in a complete test, while there were a total of 60 reagent addition steps. The predicted copper, gold and sulphur recoveries recorded for the test were 84, 75 and 31% respectively.

The poor equilibrium conditions were predominantly associated with the occurrence of bornite in the ore. The bornite floated slowly, particularly in cleaning. The gold in these ore also responds particularly poorly to flotation. The magnitude of the difference between the two calculated recoveries for a given test is another indication of the quality of the data.

The bornite ore responded well to bulk flotation of the sulphides and the stronger flotation collector regime. The other ores contained high levels of pyrite, negating bulk flotation options. Copper recoveries in the locked cycle tests were close to those indicated by the rougher batch tests conducted on the heads. Gold recoveries varied from those indicated in the batch test work, with two ores achieving better results and two ores worse results in the locked cycle tests. The two poor results were obtained with bornite and chalcopyrite–bornite ores in differential flotation.

Pilot plant operation

The flotation pilot plant test work commenced soon after the completion of the flotation variability test work. Four bulk samples with varying mineralogy, selected from ore mined from a decline, were treated in the pilot plant. Survey data from the pilot plant tests were used to provide design numbers for the final



6 Typical Cadia batch locked cycle flotation test showing recovery data

flotation plant design (flotation parameters and liquid–solid separation). Products from the different flotation streams (i.e. copper and pyrite concentrates) were collected to either send to smelters, or for further laboratory test work. For the pyrite concentrates, test work was undertaken to establish if gold contained in the pyrite could be recovered by gravity, after fine grinding, or by cyanide leaching. The gravity test work was not successful and although cyanide leaching provided acceptable gold recoveries the economics were not favourable.

Variability test work

An extensive laboratory flotation programme on 297 composite drill core samples, equally spaced throughout the deposit, was undertaken soon after the completion of the pilot plant campaign. The purpose of this test work was to provide data to determine both the copper and gold recovery–grade profiles for the deposit. Early in the batch flotation programme it because evident that 'gold drop-out' occurred during the batch cleaning tests. A typical example of this is shown in Table 5.

During the flotation pilot plant trials at Amdel, it was found that gold 'losses' in the cleaner tailings formed only a small portion of the total gold in the circuit. From this observation, it appeared that the gold drop-out was only a problem in the batch flotation test and would mostly be recovered in continuous flotation operation. However, there was still the problem of evaluating the results from batch rougher flotation test work. This was overcome by a procedure, incorporating both a batch flotation test and a locked cycle flotation test, on individually composited samples representing specific mineral assemblages. From the batch and locked cycle test, a factor was obtained to take account of losses of copper and gold during the cleaning cycle. These factors compared favourably to results obtained during the Amdel pilot flotation surveys. The factors varied between 0·93 to 0·99 for copper and 0·86 to 0·95 for gold (i.e. copper rougher recovery × factor=final copper cleaner recovery).

Metallurgical recovery model

After the completion of the 297 batch flotation tests, the results were compiled for each mineral assemblage and a linear regression model was fitted to the data. Input data comprised head assay data and sample location while the outputs were copper and gold recoveries. Copper concentrate grades were fixed as there was little observed variation in concentrate grade from batch tests on samples of similar mineralogy. The regressed mineral assemblage recovery models were given to the mining engineers to incorporate then with the geological 'block' model to allow for pit optimisation and ore scheduling. In the final analysis, an overall copper and gold recovery–grade profile was produced for the deposit and the data is summarised in Table 6.

Flotation plant design and description

Flotation feed from two-ball mill circuits are sent to two parallel banks of rougher–scavenger flotation cells, each bank having seven Outokumpu OK150 Tank cells. The design residence time per bank is 20 min which is double the residence time for the batchscale test work (scale-up factor of 2). The scavenger concentrate is rougher in a Svedala VTM400 VertiMill. The reground concentrate is cleaned in six Outokumpu OK30 Tank cells followed by four cleaner scavenger cells of similar dimensions. The cleaner scavenger tailings are either recycled to the rougher circuit or open circuited to the tailings thickener. The cleaner scavenger concentrate is recycled back to the front of the cleaner (regrinding with scavenger concentrate), whilst the cleaner concentrate is pumped to a recleaner circuit consisting of four Outokumpu OK8 trough flotation cells. All flotation reagents are supplied by bulk tankers and reagents are stored in large holding tanks on site.

The final flotation concentrate is screened to remove coarse particles and reduce abrasion in the concentrate pipeline. The screened concentrate is dewatered in a 12 m diameter Outokumpu high rate thickener. The thickened concentrate is pumped to a storage tank before being pumped 35 km in an underground pipeline

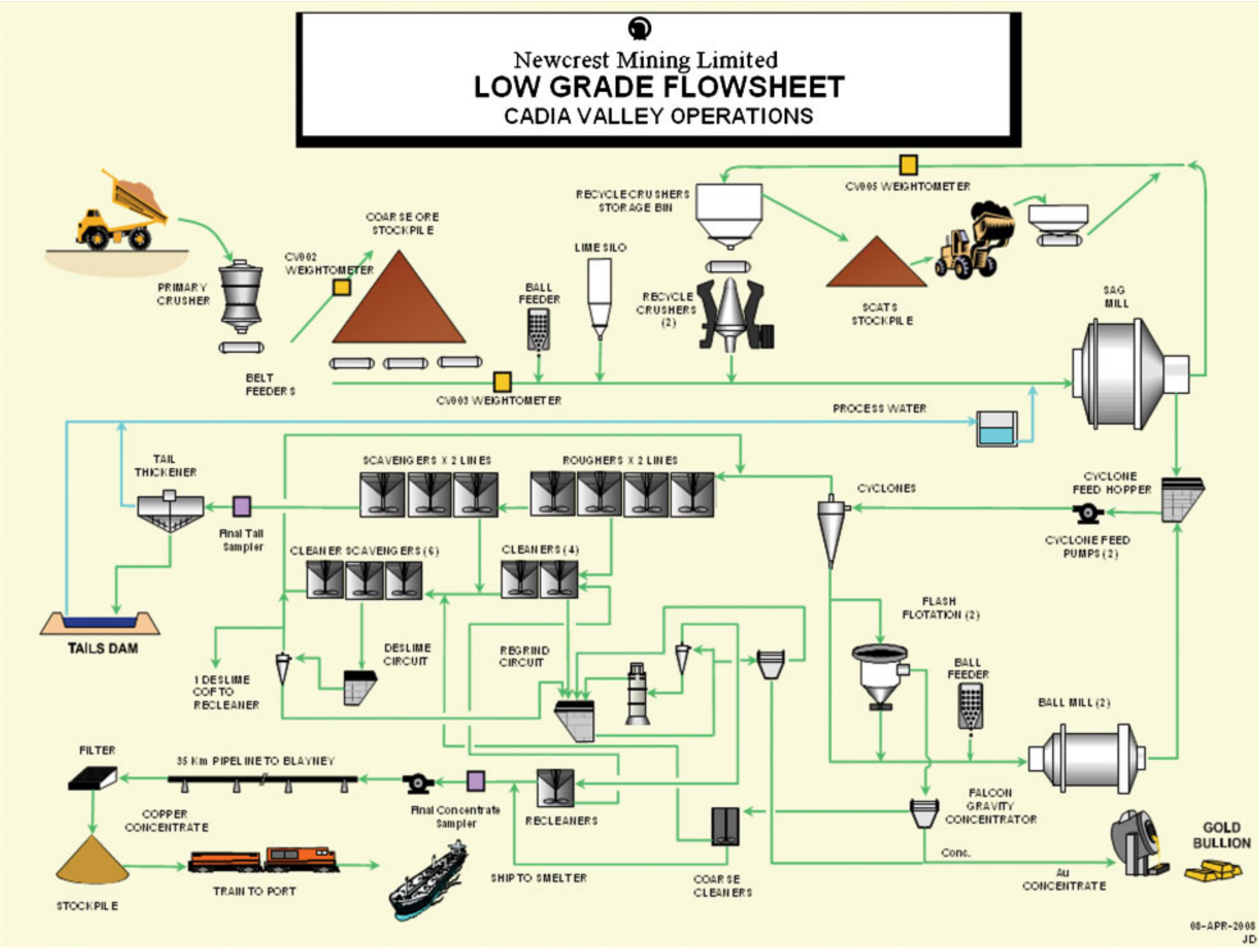
Table 5 Batch flotation results on Cadia ore showing gold drop-out during cleaning

		Grade			Recovery		
Flotation products	Mass, %	Cu, %	Au, g t^{-1}	S, %	Cu, %	Au, %	S, %
Copper final concentrate	0·53	25·9	49·4	30·6	68·0	39·4	26·6
Copper recleaner tailing	0·30	8·6	51·2	10·8	12·6	22·9	5·2
Copper cleaner tailing	1·58	0·7	3·0	1·9	5·7	7·1	4·9
Pyrite concentrate tailing	1·98	0·5	1·6	19·4	4·8	4·9	62·6
Tailings	95·6	0·19	0·18	0·01	8·9	25·7	0·8
Calculated head	100	0·20	0·67	0·62	100	100	100

Table 6 Metallurgical performance for Cadia ores

		Overall float recovery		Overall process recovery*	
Ore type	Copper concentrate grade, %	Au, %	Cu, %	Au, %	Cu, %
Bn	34	64·1	79·4	62·5	77·1
Py/Cpy	25	77·3	84·0	75·4	80·6
Cpy/Py	25	75·1	89·1	73·2	85·6
Cpy/Bn	32	57·4	77·0	56·0	74·6
Py/Bn	25	61·5	78·4	60·0	75·3

*Includes gravity recovery.



7 Schematic Cadia flowsheet

to the filtration plant at the railhead in Blayney. A single Svedala VPA 1540/24 plate and frame filter, treats up to 24 t h^{-1} of flotation concentrate and produces a filter cake of about 9 to 10% moisture.

The rougher scavenger flotation tailings gravitate to a 53 m diameter Weir Envirotech high rate thickener. The thickened tailings at 55% solids are pumped 4 km to the tailings dam. The schematic Cadia flowsheet is shown in Fig. 7.

Pertinent flotation design criteria are shown in Table 7.

Plant operation and metallurgical development

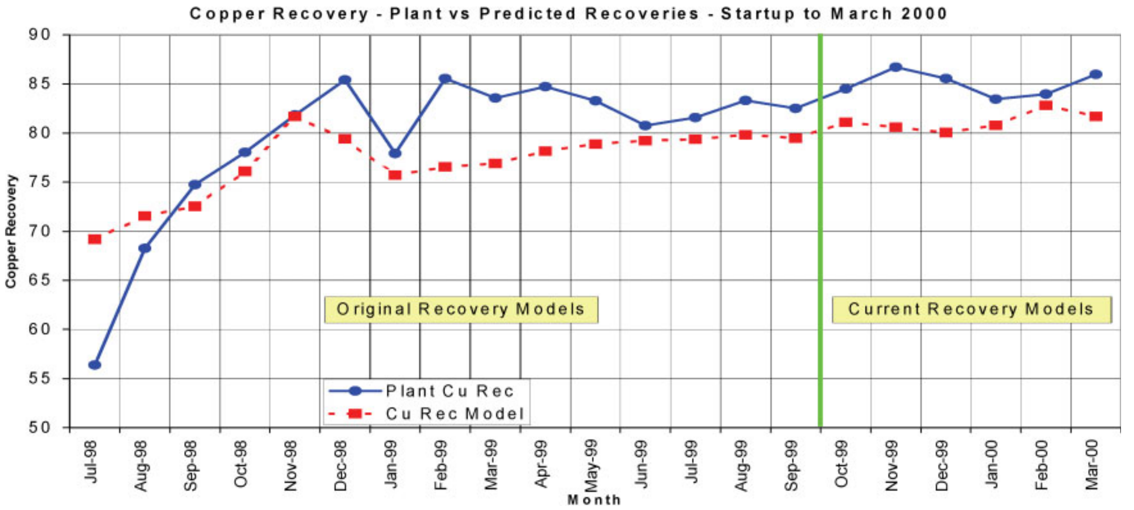
Although general performance statistics for Cadia were good, the start-up at Cadia was not without its own share of operating and mechanical issues.[11]

The presence of sericite/kaolinite in the Cadia ore was expected, from pilot plant and batch laboratory flotation test work, to create froth stability problems in the concentrator. However, the magnitude of the problem was greater than anticipated. At start-up, it became apparent that the installed flotation concentrate pipe launders, were inadequate, leading to concentrate spillage, reduced pulling rates and consequently lower than predicted flotation recoveries. Froth factors of up to 10 were measured on the plant compared to a design of 3. All the rougher, cleaner and cleaner–scavenger launders have since been replaced with open trough launders.

Another and more difficult problem relates to the clarity of the concentrate thickener overflow solution. Air entrainment, the main reason for the voluminous froth, is exacerbated before pumping by the requirement to screen the final concentrate. Numerous remedies were attempted to overcome this problem including installation of water sprays around the circumference of the thickener, use of de-foamers, coagulants, flocculants and deaeration of the concentrate to name but a few. The

Table 7 Cadia flotation design criteria

Items	Rougher	Scavenger	Cleaner	Cleaner–scavenger	Recleaner
Feed rate per line, t h^{-1}	1024				
Residence time, min	12	8	7	10	10
Pulp density, %	34	…	…	…	…
Concentrate Cu, %	5	2	15·5	7	25
Concentrate, % solids	24	16	25	15	30
Froth factor	2	2	2·5	2	3



8 Comparative data for copper recovery

best solution thus far is to deaerate the concentrate using a novel deaeration cyclone developed by Outokumpu.

During commissioning it soon became apparent that the OK150 flotation cell discharge dart valves were undersized. Larger darts were installed and modifications were made to the shaft of the dart to make it more robust.

Three flotation agitator rotors have failed because of damage to the tips and blades. This was caused by the presence of oversize pebbles pumped from the SAG mill hopper and occurred when a number of SAG mill trommel screen panels failed.

The predicted concentrate regrind feed rate, has been considerably lower than predicted. As a consequence smaller cyclones have been installed and open vane impellers placed in pumps. The design particle size for the regrind circuit was a P80 of 38 μm. However, size by size analysis of the final copper concentrate still shows the presence of coarse unliberated particles that dilute the overall copper content. Work has since been conducted to modify the regrind circuit and direct rougher concentrate to the regrind circuit to increase copper mineral liberation.

Flotation reagent consumptions and in particular frother consumption are lower than those predicted. Frother consumption (MIBC) is currently around 10 g t^{-1} compared to the design value of 40 g t^{-1}. The predicted consumption was based on de-rated laboratory and pilot plant addition rates. Collector addition is slightly lower at 6 g t^{-1}, compared to the design of 11 g t^{-1}, and is the total for the two collectors added. Originally it was intended to add only S701. However, a few months before start-up it was found that addition of a Cytec collector S8761 (a monothiophosphate) had both recovery and cost saving benefits. In addition, it was felt that the S8761 would contribute to a reduction in frother addition and possibly improved froth properties.

For the first 4 months of operation, when lower grade bornite type ores dominated in the feed, actual concentrator copper and gold recoveries matched those from the predicted recovery models. As the proportion of chalcopyrite–pyrite, pyrite–chalcopyrite and the gold content in the feed increased a progressive positive divergence between actual and predicted recoveries became apparent, the difference being around 3% for copper and 8% for gold. The differences between actual plant recoveries and those predicted from laboratory and batch flotation test work were due to:

- under estimation of recovery by the linear regression models at the high end of gold and copper in the feed
- preferential liberation of gold from pyrite in the grinding circuit because of the recycle of higher density solids in the cyclones
- lower grades of flotation concentrate
- additional gold recovery due to the use of the gold selective collector S8761
- additional gold recovery due to the inclusion of the flash flotation–gravity circuit.

New metallurgical recovery models for the different ore types have been developed and adopted. These predict the gold and copper content in the final flotation tailings rather than the overall recovery. Comparative data for copper recovery is shown in Fig. 8.

Lessons and useful hints from the case studies

The following dot points highlight a few of the more relevant lessons and outcomes from the two case studies.

Hellyer Mine: complex copper, lead, zinc flotation circuit

- The significant impact of mineral grain size and mineral association on metallurgical performance.
- Close correlation between mineral associations and zinc recovery
- Metal ratios as an indicator of mineralogical content and ore variation. Also a useful measure to select samples for variability test work.
- Preferential overgrinding of the heavier sulphide minerals in the plant and pilot plant situation compared to batch grinding leading to poorer flotation performance (slower rates) between plant and laboratory.
- Vastly different chemical environment in the plant compared to laboratory resulting in different flotation response. Oxidation, mineral galvanic interaction, presence of large amounts of very fine sulphides and large circulating loads all contributing to solution

chemical changes and poorer flotation performance in the plant situation.
- Energy intensity in conditioning and flotation impacts on metallurgical performance when surface active species are present.

Cadia Hill Gold Mine: simple copper–gold flotation circuit

- Inadequate flotation launder design that was partly due to the application of lower 'froth factor' values for design. Open trough launders are preferred to pipe launders for froth removal.
- Estimates of the amount of scavenger concentrate were lower that predicted leading to an over design and poor grinding performance in the regrind circuit. As a general observation it has been found that regrind circuits are difficult to design for low grade ores. Therefore, a more appropriate approach would be to install the correctly sized equipment after the flotation circuit has been in operation for some time and design parameters are well establish. An alternative is to install a smaller unit and increase regrind capacity as required.
- Preferential grinding of the sulphides leading to more gold liberation from pyrite in the plant situation compared to the laboratory.
- Flotation reagent consumption had been much lower than design. This is partly due to better reagent control when treating large tonnages in large flotation cells.

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