This paper discusses the methods used in the design of flotation plants, including benchscale batch and locked cycle tests and pilot plant trials. Two case studies are presented, one for the Cadia Hill Gold Mine and the other for the Hellyer copper, zinc and lead plant. These case studies present different problems with widely different target grind sizes, liberation characteristics, Hellyer’s sequential flotation circuit versus Cadia’s “simple” copper/gold flotation circuit. They have a common theme in 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 testwork (including locked cycle tests) on drill core and many months of operation of a 30 tonne per hour “pilot plant” using the old Cleveland Tin Mine process plant, modified for the duty.

The Cadia concentrator was designed on an extensive benchscale variability testwork program, approximately a dozen locked cycle tests conducted on drill core and two weeks of continuous pilot plant trial of three separate samples obtain from an adit into the orebody. The orebody is a monzonite porphyry with disseminated chalcopyrite/bornite/pyrite of very low grade (0.17 percent Cu, 0.73 g/t 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 testwork data are discussed in this paper. Plant performance is reviewed in the context of predicted versus operational performance.
INTRODUCTION

Irrespective of the project, the development of the optimum flotation process is reliant on three key factors:

- understanding of the geology and associate mineralogical variations that impact on metallurgical performance;
- definition of the mining method and the relationship between the ore reserve and mine production;
- evaluation of mineral and water chemistry which can affect the flotation process; and
- completion of adequate metallurgical and process testwork to allow the design of flowsheet, reagent scheme and process control requirements.

In order to illustrate the generic factors involved both in evaluating flotation flowsheets and designing the flotation circuits the development of two sulfide flotation projects, Hellyer and Cadia Mines, is compared and contrasted in the Case Studies provided later in the paper.

Both process plants utilised elements of novel technology and had elements of risk that required appropriate management. Both plants installed the largest flotation cells available at the time and included other aspects of novel technology. Both projects have been successfully operated and exceeded projected metallurgical flotation targets.

STEPS IN THE DEVELOPMENT AND DESIGN OF A FLOTATION CIRCUIT

Sample Selection

The first stage of process design is a thorough review of the geology and mineralogy of the deposit. Mining methods and mine plans/schedules need to be assessed when selecting samples for the various phases of the flotation testwork program. Ore selection must take account of the fact that ore bodies are not homogenous and consequently flotation response will vary within the deposit. Both changes in head grade and mineralogy will lead to changes in flotation response. The basis of sample selection include the following:

- rock type;
- mineralogical characteristics;
- ore depth and redox profile (oxide, transitional, supergene hypergene);
- geological model and ore deposition theory;
- chemical assay, or grade, and
- mine schedule.

Three major sample types are typically chosen.

Samples are selected for initial reagent definition to achieve the desired mineral separation. Separation may be a simple bulk float (i.e. pyritic gold ores, some copper ore with negligible pyrite) or as is the case for polymetallic ores, a complex three stage sequential separation (i.e copper, lead, zinc).
Variability samples are selected to assess orebody variability on the basis of known physical and chemical characteristics. As an absolute minimum, samples should be selected on the basis of spatial distribution in the ore body if no other characteristics are obvious.

Bulk samples are usually selected for pilot plant trials or large scale flotation tests. The representatively of the sample requires careful consideration. The ‘representative sample’ is useful for development of a general flotation procedure, but additional testing may be desirable on individual samples from various areas or depths of the deposit to establish the optimum conditions in each case and to obtain mill design data valid over the expected range of ore variation. Obtaining only a single bulk sample from a large drill hole, winze, trial pit or development face is an option, however unless the relationship between the metallurgy, mineralogy and geology of the bulk sample can be related to the variability testwork then it is not recommended. This is particularly important with massive sulfide orebodies where deleterious penalty elements may only be found in particular structures/areas and not observed in bulk samples used to provide concentrate samples for market assessment. Furthermore ores that perform poorly on their own may be ‘masked’ if combined with ores having good flotation response.

Fresh diamond core or mined rock (occasionally available in “mature” orebodies) are preferred for laboratory flotation testwork. The use of reverse circulation (RC) drill chips should be avoided due to the oxidation of minerals, stratification of heavy particles, loss of softer components and fines and overproduction of fines. Generally, RC drill samples provide unrepresentative metallurgy. Ore samples that are prone to oxidation (ie massive sulphides and pyrrhotite ores), and require long term storage for future testworks should be stored at low temperature (< 4°C).

**FLOTATION TESTWORK AND FLOWSHEET DEVELOPMENT**

The purpose of any flotation testwork program is to define the following:

- valuable mineral recovery and grade of the saleable concentrate
- type and quantity of deleterious elements (eg mercury, arsenic) in the concentrate that will incur financial penalties from a smelter or create downstream processing problems
- operating costs (power, reagents etc)
- capital costs (flowsheet, residence time etc), and
- flowsheet flexibility for treating the mine output.

Many factors that affect flotation are largely beyond the control of the investigator. These include characteristics of the ore such as fineness of mineral dissemination, degree of oxidation, and presence of soluble constituents. The quality and quantity of water are also in this category although these factors sometimes can be controlled by treatment and by recirculation.

Variables that are subject to control include grinding factors such as fineness of grind, pulp density, type of grinding medium, and chemical additives; conditioning factors such as collectors, depressants, pH, time, intensity, and temperature; and flotation factors such as type of cell, collectors, frothers depressants, pH intensity of agitation, and time. Other variables may include desliming, in continuous or cycle tests, circulating loads in grinding and flotation, and recirculation of water.
Laboratory Batch Flotation Testwork

The open circuit batch flotation test work is generally the first step in an investigation. While the conditions employed in batch testing should be readily transferable to plant-scale operations, this is generally less important than the use of a procedure that can be closely duplicated.

An important rule in flotation process development is to aim for simplicity. A process with unnecessary steps or reagents is difficult to investigate and analyse. There is a tendency to add new reagents or steps without proper evidence that such changes really provide an improvement.

Many different makes and types of laboratory flotation machines have been used over the years. In recent times the preferred makes are those supplied by Agitair, Denver or Wemco. For a testwork laboratory treating a wide variety of ores, the most useful laboratory flotation machine is probably one with cells of 500, 1000 and 2000g capacities, with a different diameter impeller for each cell, with stainless steel or rubber-covered impellers, glass or plastic cell bodies, and requiring an outside source of low-pressure air. A rotameter installed in the air line allows for control and duplication of the airflow from test to test.

The initial assessment usually involves grinding the ore sample (drill core samples) to various particle sizes and applying a reagent regime that is known to give the desired mineral/s separation. Following a review of the initial flotation testwork results alterations are made to grind size (coarser or finer), reagent system and flotation times to further enhance mineral recovery. Should recovery problems persist then a detailed mineralogical examination is recommended. Optimization of the cleaning stage or stages, that may include further reagent addition and regrinding, follows once acceptable recoveries are attained in the rougher float.

Several ingredients go into planning a good test program. Characteristics of the ore, empirical knowledge of related flotation separations, and economic considerations are all important factors in the selection of the reagents and conditions that should be studied. An understanding of experimental methods, that is, how to conduct the laboratory work efficiently, is a basic requirement. The evaluation of results as the work proceeds is fed back to the planning phase. Most experimental programs in floatation necessarily consist of a series of sequences of planning, experimenting, and evaluation.

The relationship between laboratory and plant metallurgy is often complex. The flotation feed preparation process (grinding and conditioning) produces different size distributions and chemical environments. These relationships must be understood by the metallurgist (Lane, Fleay, 1994, Lane, el al, 1999).

Laboratory Locked Cycle Testwork

A locked cycle test is conducted when the open circuit batch test procedure is established. Analysis of the locked cycle test stability is critical in assessing the metallurgical recoveries. It is important to ensure that the product mass and metal flow is equivalent to the feed to each cycle thus ensuring that equilibrium is reached and recoveries and concentrate are not overstated.
A locked cycle test is a repetitive batch used to simulate a continuous circuit. The basic procedure has a complete batch test performed in the first cycle which is then followed by similar batch tests which have “intermediate” material from the previous cycle added to the appropriate location in the current cycle. These batch tests, or cycles, are continued in this manner for an arbitrary number of cycles. The final products from each cycle, i.e., final concentrate and final tailings, are filtered and removed from further processing. At the end of the test, all the products, final and intermediate, are tried, weighted and subjected to chemical analysis. The test is balanced and a metallurgical projection is made.

The main objectives of the locked cycle test are to simulate plant operation with regard to the build up of fines (typically gangue) or composites; recirculating loads, water quality, reagents and soluble metal species. Even where separation by batch test is good, locked cycle tests may be used to determine the impact of recycled reagents in process water for multi-stage flotation.

Locked cycle analysis is time consuming but provides the best simulation plant conditions at benchscale. However, ores that yield very low middlings deportment of values (very clean separation of value minerals from gangue) may not benefit significantly from locked cycle testwork for determination of the metallurgical balance or flowsheet design. The number of stages required to achieve optimum separation of the middlings defines the number of cycles required in the lock cycle test.

Six cycles is generally considered the minimum, with the tailings of each subsequent stage of processing recycled to the feed of the next cycle. Problematic ores may not reach equilibrium even after 10 stages, and the assessment of water quality may take significant more effort.

The assessment of equilibrium conditions and the prediction of metallurgy from locked cycle tests is a matter of much discussion among practitioners. Simple methods, such as averaging the recovery and grade of the last two or three cycles may be adequate for simple process routes. For complex separations, a detailed assessment of the equilibrium conditions for mass, values and gangue is required by determining a detailed balance for each cycle and considering the build-up or reduction in middlings mass and metal deportment. These methods vary from a simple check that concentrate and tailings deportment equate to feed inputs, to complex statistical assessments based on in-house databases.

**Mini Pilot Plant**

The availability in recent years of mini flotation pilot plants (Anreade et al., 2001) provides an alternative method for predicting the effects of circulating loads and changes in the solution chemistry using drill core samples. The mini pilot allows great operational flexibility and therefore permits optimization of operating conditions during its execution. Recent work has shown the results obtained from the mini pilot plant are equivalent to that of a conventional scale pilot plant.

**Pilot Plant Operation**

Pilot plants should generally only be undertaken to validate the outcomes of the benchscale testwork program and not to develop or conduct preliminary evaluations of flowsheets. Some exceptions occur when only a continuous process will allow flowsheet definition and even in these cases, parallel benchscale testwork should be undertaken as a control.
The level of control over the process diminishes from bench scale to pilot scale (even when on-stream analysis is included in the pilot plant) and the amount of effort, resources and cost increases dramatically. However, pilot plants offer the opportunity to produce concentrates for dewatering, marketing and downstream processing testwork and tailings streams for dewatering, rheology, and downstream testwork.

**Variability Assessment**

The selected flowsheet should be assessed and optimised for the variability samples. Failure of the base flowsheet may occur due to a number of reasons, including the oxidised nature of the sample, changes in mineralogy or head grade, and changes in liberation requirements.

Locked cycle tests may be conducted on variability samples if the outcomes of the batch testwork are significantly different to those observed in the composite testwork program.

An outcome of the variability testwork program may be the need to campaign or blend particular ore types to maximise revenue from the plant. This may impact on the mine plan and the optimum project.

**FLOTATION CIRCUIT DESIGN**

A flotation circuit is a combination of flotation cells and auxiliary equipment arranged to delivery the optimum results from an ore following grinding and reagent treatment on a continuous basis. The circuit is designed from the results of laboratory results and pilot plant testing of ore samples. The arrangement of flotation cells is usually for both series and parallel flow. Some designers prefer parallel lines for flexibility and operating availability, other, maintain that operating availability is a function of the design of the processing designs lines and of the ease and type of their maintenance and not of their number. Single line designs can have significant advantages in conserving floor space and building volume, in reducing operating labour and in reducing the hardware for measurement and control.

Flotation circuit details and complexity depends on a number of factors, of which the numbers of commodities to be recovered, their values, and the ore texture are the most important. The simplest flotation circuits are found with a single, low value commodity (ie coal). Slightly more complex circuits having a rougher and two stages of cleaning. Porphyry copper ores represent a further degree of complexity that involves roughing, with or without scavenging, regrinding of the rougher concentrate followed by two stage cleaning. More complexity is involved with ores that either require very high grade concentrates because of rigorous specifications due to downstream processing requirements, or with very finely disseminated complex sulphide ores contain copper, lead and zinc. Fineness of dissemination of minerals, and the required concentrate purity all contribute to the degree of elaboration of the flotation flowsheet. The flotatability (measure of how fast or slow minerals float) of each mineral component will dictate where these particles report in the flotation circuit. For example on cleaning, the finer, coarser and partially liberated particles are less likely to refloat and thus report to the recycle stream. Excessive buildup of particles in recycle streams create instability in flotation circuits.
Flotation Time and Number of Flotation Cells

Design of flotation circuitry are extremely variable and are only determined from the result of laboratory and pilot plant testing, guided by experiences gained from other operations with similar types of ores and ore textures. Estimation of flotation cell requirements as to size (volume) and numbers, once the tonnages, time requirements, and safety factors have been decided is more routine.

An example of the estimation of a rougher circuit’s flotation cell requirements is given in the Denver Bulletin (M75125). This recommends ratios of plant circuit retention times to laboratory batch times in the range 1.6 to 2.6 with an average of 2.1. While there is considerable variations in the magnitude of the safety factors, the following considerations enter into their determination. One option is to provide a greater rougher circuit volume than required by the design criteria to allow for feed rate variances originating in the grinding circuit, as well as from spillages and recycle streams. A 25 percent increase has been common, but where grindability variance is unusually large, this may increased to 50 percent.

Another uncertainty requiring some thought is scale-up problems arising from small to large flotation cells. A correction factor of 80 percent is sometimes used to convert actual to effective volumes in the transfer from smaller to commercial sizes. Insufficient operating information is as yet available for the very large cells, which are uniformly provided with less power per unit volume than the smaller sizes. Unless these have an unexplained design aspects which increases efficiencies, if power intensity is any guide.

Wemco’s scale-up flotation time factor are based on scaled down from continuous cells to a 7-liter batch cell and considering all aspects of hydrodynamics and including mechanical froth removal. Plant installation surveys determined that froth removal rate was the key to equating batch and continuous test results. In a batch cell test, pulp level is at a relatively high level for good froth removal. In a plant, the pulp level can be varied. For those installations where the pulp level in the plant was relatively high, the scale-up factor was 2.5. For plants with low pulp level, the scale-up factor was substantially greater. This scale-up factor is related to the “froth factor,” measured as the percentage of the froth over the lip of the weir, and then related the “froth factor” to the ratio of rate constant of laboratory batch cell to that of the plant. For cells operating with froth factors of 55 to 60 percent, the scale-up factor is about 2.5. For lower pulp levels in the cells, the difference between the batch and continuous cells is even greater.

Regarding the question of the appropriate number of flotation cells in series, the ultimate performance will be a function of the flotation kinetics, the number of cells in series, and the total installed volume. The more cells-in-series the better the plug flow characteristics of the installation and the less short circuiting which will occur; a condition which must be considered favourable for most flotation systems. A minimum of six cells in series is usually recommended.

Up until the early nineties, the air flow number was the most commonly used approach for scale-up criterion for similar metallurgical performance between cells of different sizes (Arbiter et al, 1976). Recent work at the University of Queensland (Julius Krutschnitt Mineral Research Centre) has shown the bubble surface area flux, a parameter which is determined by bubble size and superficial gas velocity, has been identified as an alternative
criterion for flotation scale-up from a metallurgical point of view. Investigations have demonstrated a strong correlation between the flotation rate constant and bubble-surface area flux ($S_b$). Scale-up testwork have showed that $S_b$ could be used to scale-up from a 250 litre cell to a 3 cubic metres at similar froth residence times. Work conducted at the Pasminco Broken Hill concentrator indicated the $S_b$ was able to scale-up from a 60 litre pilot cell to a 100 cubic metres Outokumpu tank cell. Their results have further demonstrated the importance of froth residence time in determining the overall kinetics of flotation.

**Froth Transport in a Flotation Circuit**

The transport of froth (synonymous with concentrate transport) has its own unique set of problems. Froth is a three phase system or air, water and solids. The ‘froth factor’ is a measure of the air contained in the froth and quantified by filling either a measuring cylinder or bucket, of known volume, with froth and measuring the froth column. After air dissipation the remaining water and solids volume is measured. The original volume of froth divided by the combined volume of water and solids is the ‘froth factor’. Measured ‘froth factor’ values are not employed by the flotation cell or pump designers. They are modified based on experience and application.

Froth transfer from flotation cell to cell requires either gravity flow or pumping. The characteristics of the froth depend to a large degree on the type of ore being treated, the fineness of the particles, the solids concentration, the amount of air in the froth slurry and the type of reagents used.

Froth factors for launders and sumps are dependant on reagent addition rates and reagent type, the ability to add spray water, the nature of the sprays, the design of the launder or sump and the fineness and nature of the particles. Froth factors for up to eight to ten have been measured/reported ores with stable froth structure. These froths require high volume launders, energy to release air and effective addition of spray water (to avoid excessive dilution). Open launders assist in effective froth transport and air/slurry separation. A minimum width of 300mm to 400mm is recommended.

Froth can vary from brittle (easily broken down and the bubbles are generally large) to tenacious (the air is tightly bound in the froth and will remain a froth for many hours – bubbles tend to be very fine). Froth characteristics will vary from day to day and even hour to hour depending on these many parameters, so it is important to select froth pumps for the worst pumping scenario. The main requirement is to select the correct size pump and impeller type for the type of froth and then the correct sump speed (Warman Bulletin, No 28). Overspeeding is one of the parameters that will affect pump performance dramatically and deleteriously when handling froths. As a guide, vertical pumps are good for pumping brittle type froths. For medium type froths to very tenacious froths, horizontal froth pumps are generally superior. Recommended froth factors are shown in Table 1.

<table>
<thead>
<tr>
<th>Type</th>
<th>Froth Factor</th>
</tr>
</thead>
<tbody>
<tr>
<td>Brittle Froth</td>
<td>1.1 to 1.25</td>
</tr>
<tr>
<td>Medium Froth</td>
<td>1.25 to 1.5</td>
</tr>
<tr>
<td>Tenacious Froth</td>
<td>1.5 to 1.6</td>
</tr>
</tbody>
</table>
FLotation equipment

Conventional Flotation Cells

Conventional flotation equipment has changed radically in the last decade and very large cells up to 200 cubic metres are now available. Cell volumes range from a couple of cubic metres up to 200 cubic metres. Flotation cells come in various shapes, from square and rectangular (<30m³) to U-shape and circular (tank cells) for the very large flotation cells. Smaller cells have external or peripheral froth launders whilst the larger cells have both internal and external launders. Internal launder promote froth removal and thus improve single cell mineral recovery and promoting coarse particle (>100um) removal. Improper design of internal launders will lead to bubble coalescence, below the launder, and hence froth collapse. Furthermore, insufficient launder width and slope will result in poor froth transport and lead to decreased cell performance.

Impeller speed and design are important factors in ensuring proper slurry mixing, air dispersion and a quiescent pulp-froth interface, to prevent froth collapse.

Column Flotation Cells

Column flotation underwent a revival in the late 1980’s and early 1990’s due to its capability to replace multistage cleaning and low capital cost. In Australia, there was a trend to installing columns in rougher flotation applications as well as cleaner flotation driven in the main by the cheaper installation costs of column circuits when compared with conventional cells.

The general consensus is that these columns worked well where the value minerals are fast floating but failed to yield optimum metallurgy where the fast float components had been removed in preceding flash flotation, or the flotation kinetics were affected significantly by power input and pre-flotation conditioning was inadequate.

Column cells are no longer cheaper (capex/opex basis) on a capacity basis, although they do offer reduced footprint and short circuiting per stage when compared with conventional cells. The selection of columns, including Jameson Cells, is appropriate when the following apply:

- replacement of multistage cleaning applications where froth wash water is advantageous;
- non-sulfide and non-floating gangue is the principal component to be removed;
- the value component is fast floating, and/or
- the flotation kinetics are not significantly impacted by power input.

The acceptance of column cells in Australia has diminished to such a degree that they are rarely considered in mineral flotation.

The evaluation of column circuits is typically conducted at pilot scale (>5 m high x 0.1 m diameter) after initial benchscale testwork to establish appropriate kinetics or the need for multistage cleaning.
CASE STUDIES

Case Study One: – A Complex Copper, Lead and Zinc Flotation Circuit – The Hellyer Project

The Hellyer Project was developed by Aberfoyle Limited (now Western Metals) and proceeded from discovery in 1983 to production in 6 years (Eager 1990). The 15 Mt orebody consisted of fine grained massive sulfide containing copper, lead, zinc and silver minerals with gold in a predominantly pyrite matrix.

Sulphide mineralisation was intersected in 1983 preliminary metallurgical testwork commenced mid 1984 and an adit to provide access to the ore body was initiated in 1985. Laboratory batch testwork providing two options for further evaluation was completed by the end of 1985. Pilot plant testwork 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 1.3 Mt/a concentrator was produced in March 1989.

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 sulfide mineralisation was unaltered leading to the very fine grain size and high level of sulfide 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. Chalcopryite was strongly associated with sphalerite and often included on 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 2.

Table 2
Hellyer Mineralogy

<table>
<thead>
<tr>
<th>Mineral</th>
<th>Occurrence (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Pyrite</td>
<td>40 – 70</td>
</tr>
<tr>
<td>Sphalerite</td>
<td>15 – 25</td>
</tr>
<tr>
<td>Galena</td>
<td>6 – 12</td>
</tr>
<tr>
<td>Arsenopyrite</td>
<td>1 – 3</td>
</tr>
<tr>
<td>Chalcopyrite</td>
<td>0.8 – 2</td>
</tr>
<tr>
<td>Tetrahedrite</td>
<td>0.1</td>
</tr>
<tr>
<td>Non sulfides</td>
<td>5 – 25</td>
</tr>
</tbody>
</table>

Principal conclusions from the early mineralogy (Richmond and Lai 1988) were that a fine grind was required for sulfide 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 Testwork**

Benchscale batch flotation testwork 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 microns, 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 testwork program followed at the Aberfoyle laboratory in Tasmania. At the completion of this program two flowsheets producing a bulk or four separate concentrates, were proposed for flotation locked cycle testwork.

The Bulk flotation flowsheet was aimed at generating a combined or bulk lead, zinc, silver, gold and copper product for processing in Imperial Smelther Furnace (ISF) smelters. The four product flowsheet produced individual products of copper-silver, lead-zinc and bulk concentrate for treating in their respective smelters. 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 adjacent to the mine. However mine development and concentrator design and construction would take a number of years. It was decide to convert the nearby Lunia 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 Lunia 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.

The key conclusions from the benchscale testwork were that the minerals were slow flotating and long residence times were required to maximise selectivity and recovery and a regrind of the Pb and Zn rougher concentrates to 20 microns 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 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 Testwork**

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

High circulating loads in locked cycle testwork for lead and zinc cleaning resulted 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. A typical locked cycle test result showing recoveries and grades is shown in figures 1 and 2.

**Figure 1**

![Figure 1: HELLYER LOCKED CYCLE BATCH FLOTATION DATA](image)

**Figure 2**

![Figure 2: HELLYER LOCKED CYCLE BATCH FLOTATION DATA](image)

Even though there are six cycles only, the flowsheet for the processes 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 minutes to complete and there are 312
reagent addition steps. The documented predicted grades and recoveries from the locked test are shown below in table 3.

<table>
<thead>
<tr>
<th></th>
<th>Copper</th>
<th>Zinc</th>
<th>Lead</th>
</tr>
</thead>
<tbody>
<tr>
<td>Grade (%)</td>
<td>10.3</td>
<td>63.6</td>
<td>49</td>
</tr>
<tr>
<td>Recovery (%)</td>
<td>48</td>
<td>31</td>
<td>62</td>
</tr>
</tbody>
</table>

**Pilot Plant**

Aberfoyle operated the Cleveland Tin mine and processing plant at Luina, 50 km North West of Hellyer. In the mid 1980’s the dramatic fall in the tin price resulted in the 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 tonne per hour pilot trials at Cleveland over the typical 0.25 to 1 tonne per hour pilot plant at facilities like Amdel, was taken as:

The principal difference between the benchscale testwork and the pilot plant operation was the increased quantity of less than 7 micron fines produced in the pilot plant. At benchscale only 22 percent of the mass was less than 7 microns for a gruel of 40 um. At pilot scale this increased to as much as 50 percent less than 7 microns and was typically 30 to 35 percent less then 7 microns 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 maximize the selectivity of the valuable minerals over pyrite. Pilot plant flotation rates were typically double that observed in the benchscale testwork. A total of 15 000 tonnes of ore was treated through the pilot plant prior to 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 tonne per hour and operated producing a bulk concentrate whilst the Hellyer plant was designed and constructed. During this period, flotation cells and impeller types were evaluated focusing on the particular 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. 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.

**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 recognized that performance would vary with head grade. As the head grade was reasonably constant for the first 5 years no allowance was made for head grade in the predicted grade-recovery relationship. It was anticipate 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 is a very complex flotation flowsheet with four concentrates being produced and multiple cleaning and regrinding stages.

Flotation feed is 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 cubic metres) roughers is fed to a 10 metre high by 0.9 metre diameter column cell to produce a final concentrate containing 12-16 percent copper. Column cell tailings are returned to the conditioning tank ahead of the copper-silver rougher cells.

The tailing from the copper-silver roughers is conditioned with lime to increase before adding cyanide prior to lead roughing circuit. Flotation collector is stage-added to each of the seven Maxwell MX14 lead rougher cells. The lead rougher concentrate is further cleaned in a three stage cleaning circuit using Maxwell MX12 (32 cubic metre) cells. The first cleaner tailing reports to the conditioning tank ahead of the copper-silver rougher cells.

The tailing from the lead roughers is conditioned with copper sulphate and lime, prior to zinc rougher flotation. Collector is again stage-added to each of the eight MX14 Maxwell zinc rougher and scavenger cells. Additional copper sulphate and lime is added mid-way through scavenging. The zinc scavenger tailing, containing less that one percent zinc, is the final tailings. The zinc scavenger concentrate reports to the zinc regrind circuit. This stream is first deslimed before reporting to a Kubota Tower Mill in closed circuit with a second set of cyclones. The regrinding improves liberation of sphalerite from the pyrites gangue. The regrind material is reconditioned with copper sulphate, lime and collector before being combined with the zinc rougher concentrate. These combined streams report to the first of three zinc cleaning stages. Lime is added to the first and third zinc cleaning stages to raise the pH. The zinc concentrate produced from the third cleaner contains 50 percent zinc with a recovery of 63 to 68 percent.

Prior to bulk rougher the combined lead and zinc first cleaner tailings are regrind in the bulk regrind circuit consisting of cyclones enclosed circuit with a third 335kW Kubota Tower Mill. This regrind step is necessary to reduce composite sphalerite-pyrite and galena pyrite to enable more selective flotation. The product size is about 20-25 microns and is reconditioned with copper sulphate. Lime is added to the bulk roughers and second added to each of the six bulk rougher cells. The bulk rougher concentrate reports to the first of two bulk cleaning stages. The bulk scavenger tailing reports with the zinc scavenger tailing to the final tailings sump and is then pumped to the tailings dam. The final bulk concentrate contains 35 percent zinc and 15 percent 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 benchscale. The slow flotation in the plant was linked to high levels of surface passivation (typically metal
This resulted in particles that were slow to further oxidise as noted in the early benchscale testwork. Some relevant flotation design criteria are shown in Table 4.

<table>
<thead>
<tr>
<th>Item</th>
<th>Feed</th>
<th>Rougher</th>
<th>Scavengers</th>
<th>Cleaner</th>
<th>Re Cleaner</th>
<th>3rd Cleaner</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Copper Float</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>tph</td>
<td>130 - 156</td>
<td>30 - 40</td>
<td>30 - 10</td>
<td>40 - 48</td>
<td></td>
<td></td>
</tr>
<tr>
<td>% Solid</td>
<td>30 - 40</td>
<td>32</td>
<td>55</td>
<td>12</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Residence time (mins)</td>
<td>0.4</td>
<td>3</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Grade Cu (%)</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td><strong>Lead Float</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>tph</td>
<td>130 - 156</td>
<td>30 - 40</td>
<td>30 - 40</td>
<td>40 - 50</td>
<td>40 - 50</td>
<td>40 - 50</td>
</tr>
<tr>
<td>% Solid</td>
<td>30 - 40</td>
<td>74</td>
<td>74</td>
<td>52</td>
<td>29</td>
<td>28</td>
</tr>
<tr>
<td>Residence time (mins)</td>
<td>6.5</td>
<td>25</td>
<td>12</td>
<td>35</td>
<td>50</td>
<td>60</td>
</tr>
<tr>
<td>Grade Cu (%)</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td><strong>Zinc Circuit</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>tph</td>
<td>75 - 115</td>
<td>30 - 40</td>
<td>30 - 40</td>
<td>40 - 50</td>
<td>40 - 50</td>
<td>30 - 40</td>
</tr>
<tr>
<td>% Solid</td>
<td>30 - 40</td>
<td>84</td>
<td>57</td>
<td>40</td>
<td>34</td>
<td></td>
</tr>
<tr>
<td>Residence time (mins)</td>
<td></td>
<td>14</td>
<td>30 - 35</td>
<td>45</td>
<td>48</td>
<td>50</td>
</tr>
<tr>
<td>Grade Cu (%)</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td><strong>Bulk Float</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>tph</td>
<td>40 - 60</td>
<td>40 - 50</td>
<td>40 - 50</td>
<td>40 - 50</td>
<td></td>
<td></td>
</tr>
<tr>
<td>% Solid</td>
<td>10 - 20</td>
<td>69</td>
<td>75</td>
<td>76</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Residence time (mins)</td>
<td>10</td>
<td>14</td>
<td>18</td>
<td>30</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Grade Pb</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Grade Zn</td>
<td></td>
<td>16</td>
<td>25</td>
<td>76</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

**Plant Operation and Metallurgical Development**

During the operations of the Hellyer concentrator the variability of grade-recovery with ore type become apparent. Metal ratios, particularly zinc to iron, were a strong determinant on recoveries. It was then recognized that the initial trailing of textures in the pilot plant was flawed because by coincidence they had all been tested at a near identical metal ratio. Correlations were then developed initially between plant performance and Qem*Sem Phase Specific Surface Areas (PSSA) 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 figure 3.
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 to iron ratio for the orebody was 0.6 but the initial pilot plant work was performed on ores of zinc to iron of 0.7 to 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 to iron ratio equal to 0.4 showed zinc recoveries could be as much as 20 percent lower due to the increased sphalerite/pyrite associations.

Subsequent investigations (Rumball, Richmond, 1996) 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 affect 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 minutes of flotation before reaching a plateau. The Cleveland 30 tonnes per hour operation which floated lead immediately after grinding experienced less oxidation than the Hellyer circuit which used 30 minutes of copper flotation before the lead circuit. This explains the increased residence time needed in the Hellyer circuit over the 30 tonnes per hour Cleveland plant.

Flotation selectivity and recovery were enhanced in the laboratory and plant through the use of surface cleaning agents (eg EDTA) or through the use of high shear rates (HIC) in flotation conditioning. Improved plant performance was achieved by 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 have been detrimental to flotation performance, particularly if flotation conditions in grinding and pre-flotation are optimised.
Hellyer Concentrator – Key Process Changes 1992 to 2000

The impact of some of the key process changes, as discussed in the previous section, on metallurgical performance are shown in figure 4.

Figure 4

Hellyer Concentrator Metal Recoveries
1989 - 2000

Case Study Two: A Simple Copper – Gold Flotation Circuit – The Cadia Hill Project

The Cadia Mine is located 25kms South West of Orange in the central west of New South Wales (NSW). 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 (Wood et al., 1995). 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 million tonnes of ore with an average gold grade of 0.74g/t and 0.17 percent 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 20mm thick with a vein density of 2 to 5 per metre. These rarely occupy more than 5 percent of the rock mass.

Initial diamond core drill spacing was on 100 metre by 150 metre grid. This was followed by both 100 metre by 100 metre and 50 metre by 50 metre 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 to S ratios and the amount of cyanide solution copper and provides a classification of the sulphides as shown in table 5.

<table>
<thead>
<tr>
<th>Cu:S Ratio</th>
<th>S %</th>
<th>CNSolCu (%)</th>
<th>Interpreted Sulphides</th>
</tr>
</thead>
<tbody>
<tr>
<td>&lt; 0.5</td>
<td>&gt; 0.01</td>
<td>&lt; 20</td>
<td>Pyrite + Chalcopyrite</td>
</tr>
<tr>
<td>&lt; 0.5</td>
<td>&gt; 0.01</td>
<td>&gt; 20</td>
<td>Pyrite + Bornite</td>
</tr>
<tr>
<td>0.5 – 1.5</td>
<td>&gt; 0.01</td>
<td>&lt; 20</td>
<td>Chalcopyrite + Pyrite</td>
</tr>
<tr>
<td>0.5 – 1.5</td>
<td>&gt; 0.01</td>
<td>&gt; 20</td>
<td>Chalcopyrite + Bornite</td>
</tr>
<tr>
<td>&lt; 1.5</td>
<td>&gt; 0.01</td>
<td>All</td>
<td>Bornite</td>
</tr>
<tr>
<td>All</td>
<td>&lt; 0.01</td>
<td>All</td>
<td>Oxide</td>
</tr>
</tbody>
</table>

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

**Benchscale Flotation Testwork**

The flotation program involved reagent scoping studies, ore variability testwork, a pilot plant campaign and ore body mineral (metal) recovery modelling.

The flotation reagent scoping study was undertaken in three phases. Initial “in-house” flotation testwork by Newcrest indicated that a grind size of about 150 microns 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 about 2 percent.

Drill core samples crushed to minus 2mm 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 program 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 program, the collector addition was kept constant whilst some latitude, on the part of the flotation operator, was allowed to vary the frother addition. The results of this program 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 testwork.

**Locked Cycle Testwork**

The aim of the locked cycle testwork was to evaluate and reinforce the metallurgical recovery models recovery for developed for Cadia Hill. The models are 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 testwork 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 figure 5.

![Figure 5](image)

The locked cycle test program for Cadia was simpler than that for Hellyer. The total time involved in completing a single cycle was 50 minutes. 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 percent, 75 percent and 31 percent 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 ores 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 testwork, 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 testwork commenced soon after the completion of the flotation variability testwork. 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 flotation plant design (flotation parameters and liquid – solid separation). Products from the different flotation streams (ie. copper and pyrite concentrates) were collected to either send to smelters, or for further laboratory testwork. For the pyrite concentrates, testwork was undertaken to establish if gold contained in the pyrite could be recovered by gravity, after fine grinding, or by cyanide leaching. The gravity testwork was not successful and although cyanide leaching provided acceptable gold recoveries the economics were not favourable.

**Variability Testwork**

An extensive laboratory flotation program 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 testwork was to provide data to determine both the copper and gold recovery - grade profiles for the deposit. Early in the batch flotation program it became evident that “gold drop out” occurred during the batch cleaning tests. A typical example of this is shown in table 6.

<table>
<thead>
<tr>
<th>Flotation Products</th>
<th>Mass (%)</th>
<th>Grade</th>
<th>Recovery</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>Cu (%)</td>
<td>Au (g/t)</td>
</tr>
<tr>
<td>Copper Final Concentrate</td>
<td>0.53</td>
<td>25.9</td>
<td>49.4</td>
</tr>
<tr>
<td>Copper Re-Cleaner Tailing</td>
<td>0.30</td>
<td>8.6</td>
<td>51.2</td>
</tr>
<tr>
<td>Copper Cleaner Tailing</td>
<td>1.58</td>
<td>0.7</td>
<td>3.0</td>
</tr>
<tr>
<td>Pyrite Concentrate Tailing</td>
<td>1.98</td>
<td>0.5</td>
<td>1.6</td>
</tr>
<tr>
<td>Tailings</td>
<td>95.6</td>
<td>0.19</td>
<td>0.18</td>
</tr>
<tr>
<td>Calculated Head</td>
<td>100</td>
<td>0.20</td>
<td>0.67</td>
</tr>
</tbody>
</table>

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 dropout 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 flotation testwork. 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 (ie. copper rougher recovery x 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 whilst 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 them 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 7.

<table>
<thead>
<tr>
<th>Ore Type</th>
<th>Copper Concentrate Grade %</th>
<th>Overall Float Recovery (Au %)</th>
<th>Overall Process Recovery (Au %)</th>
<th>Overall Process Recovery (Cu %)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Bn</td>
<td>34</td>
<td>64.1</td>
<td>62.5</td>
<td>77.1</td>
</tr>
<tr>
<td>Py/Cpy</td>
<td>25</td>
<td>77.3</td>
<td>75.4</td>
<td>80.6</td>
</tr>
<tr>
<td>Cpy/Py</td>
<td>25</td>
<td>75.1</td>
<td>73.2</td>
<td>85.6</td>
</tr>
<tr>
<td>Cpy/Bn</td>
<td>32</td>
<td>57.4</td>
<td>56.0</td>
<td>74.6</td>
</tr>
<tr>
<td>Py/Bn</td>
<td>25</td>
<td>61.5</td>
<td>60.0</td>
<td>75.3</td>
</tr>
</tbody>
</table>

**Table 7**

Metallurgical Performance for Cadia Ores

**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 minutes which is double the residence time for the batch scale testwork (Scale up factor of 2). The scavenger concentrate is rougher in a Svedala VTM400 Verti Mill. 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 re-cleaner circuit consisting of four Outokumpu OK8 flotation cells. All flotation reagents are supplied by bulk tankers and reagents are stored in large holding tanks on site.

1 Includes gravity recovery
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 metre diameter Outokumpu high rate thickener. The thickened concentrate is pumped to a storage tank prior to being pumped 35km in an underground pipeline to the filtration plant at the railhead in Blayney. A single Svedala VPA 1540/24 plate and frame filter, treats up to 24 tph of flotation concentrate and produces a filter cake of about 9 to 10 percent moisture.

The rougher scavenger flotation tailings gravitate to a 53 metre diameter Weir Envirotech high rate thickener. The thickened tailings at 55 percent solids are pumped 4kms to the tailings dam.

Pertinent flotation design criteria are shown in table 8.

<table>
<thead>
<tr>
<th>Items</th>
<th>Rougher</th>
<th>Scavenger</th>
<th>Cleaner</th>
<th>Cleaner Scavenger</th>
<th>Re Cleaner</th>
</tr>
</thead>
<tbody>
<tr>
<td>Feed Rate (tph)</td>
<td>1024</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Residence Time (mins)</td>
<td>12</td>
<td>8</td>
<td>7</td>
<td>10</td>
<td>10</td>
</tr>
<tr>
<td>Pulp Density (%)</td>
<td>34</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Concentrate Cu (%)</td>
<td>5</td>
<td>2</td>
<td>15.5</td>
<td>7</td>
<td>25</td>
</tr>
<tr>
<td>Pulp Density %</td>
<td>24</td>
<td>16</td>
<td>25</td>
<td>15</td>
<td>30</td>
</tr>
<tr>
<td>Froth Factor</td>
<td>2</td>
<td>2</td>
<td>2.5</td>
<td>2</td>
<td>3</td>
</tr>
</tbody>
</table>

**Plant Operation and Metallurgical Development**

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

The presence of sericite/kaolinite in the Cadia ore was expected, from pilot plant and batch laboratory flotation testwork, to create froth stability problems in the concentrator. However the magnitude of the problem was indefinable. In practice and at start up it soon 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 being 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 that still persists relates to the clarity of the concentrate thickener overflow solution. Air entrainment, the main reason for the voluminous froth, is exacerbated prior to 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 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 undersize. Larger darts were installed and modifications were made to the shaft of the dart to make it more robust.
To date 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 microns. 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 is underway to assess the regrinding of other streams in the flotation circuit.

Two extra concentrate filter chambers (8 percent extra capacity) have been installed on the Svedala plate and frame filter in an effort to decrease the moisture content of the cake which at the time was around 10 percent moisture compared to the design of 9.5 percent. Filter cloth wear is a priority and filter cloth evaluation is an ongoing process.

Flotation reagent consumptions and in particular frother consumption are lower than those predicted. Frother consumption (MIBC) is currently around 10g/t compared to the design figure of 40g/t. The predicted consumption was based on de-rated laboratory and pilot plant addition rates. Collector addition is slightly lower at 6g/t, compared to the design of 11g/t, and is the total for the two collectors added. Originally it was intended to add only S701 however a few months prior to 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 four months of operation, when lower grade bornite type ores predominated 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 percent for copper and 8 percent for gold. A study underway to ascertain reasons for the differences between actual plant recoveries and those predicted from laboratory and batch flotation testwork has thus far highlighted probable and possible reasons for this as follows:

- 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 figure 6.
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 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 testwork.
- 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.

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 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. Therefore a more appropriate approach would be to installed the correctly sizes 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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REFERENCES

Aberfoyle Limited Hellyer Operation, 1989, Information bulletin

Agar, GE, and Kipkie, WB, 1978, “Predicting Locked Cycle Flotation Test Results from Batch Data” CIM Bulletin Vol 71, No 824, pg 140-147


Andrade, VL, Santos, NA, Goncalves, KLC, Wyslousil, H, 2002, Obtaining Metallurgical Data from Drill Core Samples Using Mini Pilot Plant, 34th Annual Meeting of Canadian Mineral Processors, Ottawa

Arbiter, N, 2000, Development and Scale Up of Large Flotation Cells, Mining Engineering, March 2000


Eager M, 1990, The Hellyer Mine – Feasibility to Production with Confidence, Mincost ’90, Sydney, Australian Institute of Mining and Metallurgy, pg 133


Francis RS, Richmond, GD, Campbell, JJ, *Operations at the Hellyer Concentrator*, XXXXX


Lane G, 1992, *Development Metallurgy – Symptom or Cause*, AMIRA/ JKMRC Flotation Workshop, Brisbane


Lane GS, Richmond GD, 1993, *Improving Fine Particle Flotation Selectivity at Hellyer*, XVIII International Mineral Processing Congress, Sydney, pg 897


MacDonald, RD, and Brison, RJ, 1962, “*Applied Research in Flotation*” Chapter 12 in *Froth Flotation*, Society of Mining Engineers NY Fuerstnau, DW Editor


Mular, AL, Richardson, JM, 1986, “Metallurgical Balance” Chapter 39 in *Design and Installation of Concentration and Dewatering Circuits*, Society of Mining Engineers N.Y. Mular and Anderson Editors


Ounpuu, M, Proceeding, 2001, 33rd Annual Meeting of Canadian Mineral Processors, Ottawa


Richmond GD, XXXX, *Beneficiation of Fine Ores – What Can You Do?*, XXXXXXX
