

Recent Process Developments at the Phu Kham Copper-Gold Concentrator, Laos

D Bennett¹, I Crnkovic² and P Walker³

ABSTRACT

The Phu Kham deposit represents a copper-gold porphyry system, with mineralisation present in skarn, stockwork and disseminated styles. Significant folding and alteration events have created a complex heterogeneous mineralogy horizon. Weathering and water table contact have created a leached zone, overlying transition zones with supergene chalcocite- dominant secondary copper mineralisation and clay-rich gangue. Primary ore copper mineralisation is mainly chalcopyrite with minor bornite. The major challenges to the copper-gold flotation process are a wide size distribution of chalcopyrite mineralisation and poor primary grind liberation, a high pyrite content in skarn ore requiring aggressive pyrite depression conditions, clay-rich gangue and non-sulfide copper mineralisation in weathered zones, and a significant association of gold with pyrite.

The Phu Kham concentrator has been developed as a conventional semi-autogenous grinding (SAG) and ball milling circuit followed by selective rougher flotation, regrinding and cleaner flotation to produce a copper concentrate containing payable gold and silver values. The concentrator flow sheet design offered a capital efficient compromise between high copper recovery bulk sulfide flotation with large cleaning capacity, and lower recovery copper selective rougher flotation to ensure concentrate specification of 24 per cent copper grade could be achieved. This paper will examine and discuss concentrator flow sheet development, including projects implemented since commissioning to improve copper recovery, and future projects designed to maintain and enhance copper in concentrate production with decreasing copper grade and increasing pyrite content of ore feed, and increasing hardness of primary ore.

INTRODUCTION

The Phu Kham operation consists of a copper-gold mine using conventional shovel mining and truck haulage to a 12 Mt/a concentrator. The project is owned and operated by Phu Bia Mining Limited. PanAust Ltd based in Brisbane Australia holds a 90 per cent interest in Phu Bia Mining through its wholly owned subsidiary Pan Mekong Exploration Pty Ltd, with the remaining ten per cent held by the Government of Laos PDR.

The Phu Kham Copper-Gold deposit is located in Xaisoumboun province as shown in Figure 1, approximately 120 km north of the Lao capital Vientiane. Access to the mine is approximately four hours by road from Vientiane.

The Phu Kham 12 Mt/a concentrator was designed and built to treat a high pyrite copper-gold skarn ore with significant clay content, as described by Meka and Lane (2010). The plant was commissioned in 2008 for a capital cost of approximately \$150 M, placing it in the lowest quartile for capital intensity for copper mineral processing projects.

The installed plant was a compromise between a high recovery but high capital intensity design, and a lower

recovery but technically lower risk and low capital intensity design. The selective rougher flotation design was driven by the complex and variable mineralogy and high pyrite content, with over 90 per cent of pyrite required to be rejected in order to produce a final concentrate of over 23 per cent copper. With increasing depth of the pit since the commencement of operations, the weathering profile of the feed has changed such that the ore became primary dominant in 2010, with chalcopyrite the main copper sulfide mineral. The complex folding and alteration of the ore zones has meant continued mining of supergene and oxidised areas within the pit, with the copper mineralogy remaining diverse and varying from native, oxide, secondary, and primary copper species within short time periods.

The development of the Phu Kham flow sheet was driven by the poor recovery in comparison to other low-grade copper-gold ores, and a need to counter decreasing ore grades from 2013. Major projects implemented up until 2011 included increasing rougher capacity by 25 per cent and increasing first cleaner capacity by 16 per cent, and the installation of a

1. MAusIMM, Principal Metallurgist, PanAust Limited, Level 1, 15 James Street, Fortitude Valley Qld 4006. Email: duncan.bennett@panaust.com.au

2. MAusIMM, Process Technical Services Superintendent, Phu Bia Mining Limited, 7th Floor, Capital Tower, 23 Singha Road, Vientiane, Laos People's Democratic Republic. Email: ivan.crnkovic@pbm.panaust.com

3. MAusIMM, General Manager – Technical Services, PanAust Limited, Level 1, 15 James Street, Fortitude Valley Qld 4006. Email: peter.walker@panaust.com.au

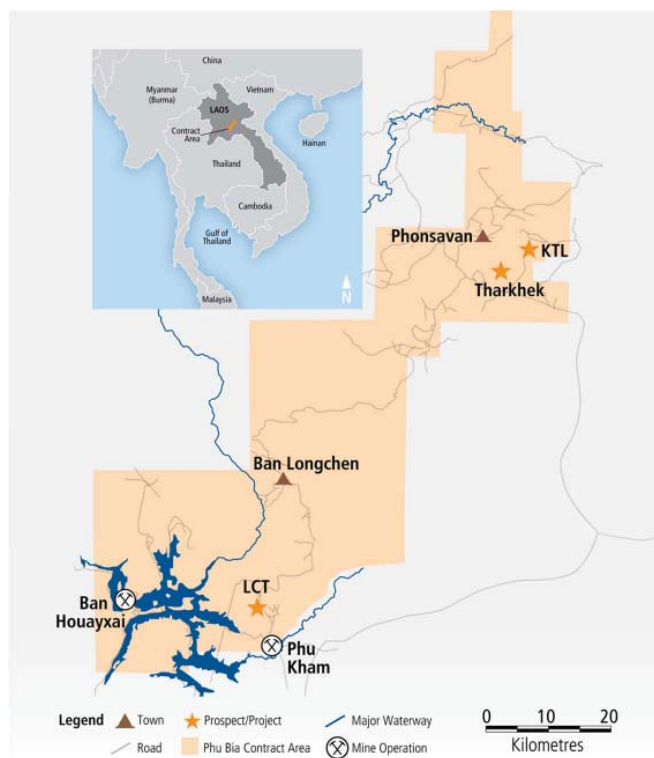


FIG 1 - Location of the Phu Kham copper-gold mine.

Jameson Cell in a cleaner scalper duty. In 2012, the operation is being upgraded to a nominal throughput of 16 Mt/a with installation of a second 13 MW ball mill, a further 33 per cent increase in rougher capacity, 40 per cent increase in second cleaner capacity, and 33 per cent increase in third cleaner capacity.

In 2009, a project to achieve step-change in copper and gold recovery from Phu Kham was initiated. A process development study was completed in 2011, which showed that it was technically feasible to generate a low-grade copper and gold mineral concentrate by bulk sulfide flotation of concentrator tailings suitable for leaching for recovery of copper and gold into high-grade products. During the study, opportunities for increasing copper and gold recovery in the existing concentrator using standard processing methods became apparent, and detailed mineralogical work and metallurgical test work was undertaken to determine the causes of copper and gold loss to tailings. The mineralogical work revealed that up to 60 per cent of copper sulfide mineral lost was in coarse non-sulfide gangue composites, and over 50 per cent of gold loss was in gold-pyrite composites. The work presented an opportunity to recover these composites by less selective rougher flotation, before upgrade and additional recovery for both copper and gold into a 23 per cent copper concentrate by regrinding of rougher concentrate to 20 μm and additional cleaning flotation capacity.

In 2013, the operation will increase total recovery of both copper and gold by six per cent into final concentrate. The recovery increase will be achieved through increasing mass recovery in 'less selective' roughing, with additional regrind capacity to reduce the rougher concentrate particle size to 20 μm , before upgrade to final concentrate in an expanded cleaning circuit. A second filter will be installed to dewater the additional concentrate produced.

GEOLOGY AND MINERALOGY

The Phu Kham copper-gold deposit is a complex copper-gold mineralised dyke system, which has undergone a number

of faulting, folding and alteration events. Mineralisation is present in iron-rich skarns, silica-rich stockwork, and altered disseminated styles. Chalcopyrite and bornite are the dominant primary copper minerals in skarn, stockwork, and disseminated mineralisation. Gangue mineralogy is mainly quartz, mica and pyrite, with significant kaolinite clay and talc-related magnesium silicate content within the weathered zones.

A gold-enriched oxide zone on the Phu Kham orebody was the resource for the heap leach gold mine which was built and operated by Phu Bia Mining during the 2005 to 2010 period. Below the oxide zone, there is a zone of supergene weathering, with copper leached from the oxide zone re-precipitated in contact with pyrite grains as particles and coatings of chalcocite and covellite, with minor enargite and tennantite copper arsenic sulfides. Significant copper enrichment in the oxide and supergene zones is also present as oxide and native copper species.

Skarns are present as replacement of carbonate minerals, with disseminated grains of chalcopyrite and bornite in banded to massive pyrite skarns and veinlets containing pyrite, chalcopyrite and bornite in garnet, magnetite, and hematite-chlorite skarns. Pyrite skarns are common throughout the mineralised system.

Stockwork mineralisation is present as fine fractures in quartz veins. The fractures host pyrite, chalcopyrite and bornite sulfide minerals. Minor chalcopyrite mineralisation is also present in quartz-carbonate veins. Disseminated mineralisation consists of scattered grains of bornite and chalcopyrite in sericite altered host rock.

Gold occurs as small grains associated with pyrite and copper sulfides throughout the mineralised system.

CIRCUIT DESCRIPTION

The original 12 Mt/a concentrator design and commissioning in 2008 has been described in detail by Crnkovic *et al* (2009).

The crushing plant consists of a primary 55 in \times 77 in gyratory crusher, with single truck dump point above a pocket designed to hold 200 t capacity equivalent to two 777D haul trucks. Crusher discharge drops to a crushed ore bin of 200 t capacity. The crushed ore bin is emptied by a variable speed apron feeder onto a crushed ore transfer conveyor belt (CV-001). The CV-001 conveyor transfers the ore to an 890 m long overland conveyor CV-002, which moves ore to the coarse ore stockpile with a live capacity of approximately 24 000 t. There is additional dead capacity for storage of up to 300 000 t of ore.

Ore is reclaimed from the crushed ore stockpile by two variable speed apron feeders onto a SAG mill feed conveyor. SAG mill grinding media is added to the ore feed conveyor via a spillage return hopper. Primary grinding is achieved in a dual pinion 13 MW variable speed slip energy recovery/hyper-synchronous drive 34 ft \times 20 ft SAG mill in closed circuit with scats return conveying including a high-lift conveyor to overcome topography constraints. SAG mill discharge is classified using an integral mill trommel, with minus 12 mm product reporting to a 1.85 MW cyclone feed pump. Cyclone feed is classified in a cluster of 18 650 mm diameter cyclones, with cyclone underflow reporting to a dual pinion 13 MW drive 40 ft \times 24 ft ball mill. Quicklime is added to the ball mill for flotation pH control to depress pyrite. Ball mill product returns to the cyclone feed pump.

Cyclone overflow reports to a multiple stage feed sampler before a bank of 200 m³ tank cell roughers. The original plant had a single 200 m³ rougher feed conditioning tank

before eight 200 m³ rougher cells, with the conditioning tank converted to a ninth flotation cell in 2009. In early 2011, a tenth 200 m³ rougher cell was installed and commissioned. Aero 9810 collector is used to recover copper sulfide minerals while maintaining selectivity against pyrite.

Rougher bank tailings passes through a static dual fin pipe sampler before reporting to final tailings mixing box where it is combined with cleaner scavenger tailings. The mixing box discharges to a metallurgical multiple stage sampler and a final tailings sump. The final tailings sump discharges slurry by gravity through two 750 mm diameter tailings lines, which transport tailings approximately 1.5 km to a cross-valley subaqueous tailings storage facility (TSF).

Rougher concentrate is classified in a cluster of six 400 mm diameter cyclones, with cyclone overflow reporting to a Jameson Cell feed hopper. Cyclone underflow reports to an open-circuit M10000 IsaMill™ regrind mill. IsaMill™ discharge reports to the Jameson Cell feed hopper. The 24 downcomer, 6500 mm diameter Jameson Cell was commissioned in March 2011 in a cleaner feed scalping duty, with concentrate passing through a pipe sampler for process control, before reporting to the final concentrate thickener feed sampler, and tailings reporting to the conventional cleaning circuit. A simplified flow diagram of the 12 Mt/a concentrator following installation of the Jameson Cell is shown in Figure 2.

The conventional cleaning circuit consists of three stages, with the first stage in open circuit. The original first stage of cleaning had a 70 m³ conditioning cell, followed by three 70 m³ first cleaners and three 70 m³ cleaner scavengers. In 2009, the conditioning tank was converted into an additional first cleaner cell. Cleaner scavenger tailings pass through a static dual fin sampler before reporting to the final tailings mixing box. First cleaner and cleaner scavenger concentrates report to the second cleaner, which consists of four 20 m³ cells. Second cleaner concentrate advances to the third cleaner of three 20 m³ cells, while second cleaner tailings returns to the first cleaners. Third cleaner concentrate passes through a pipe sampler, before

reporting to the final concentrate thickener feed sampler. Third cleaner tailings return to the second cleaners.

Final concentrate (combined Jameson Cell and third cleaner concentrates) is sampled in a multiple stage metallurgical sampler, before gravitating to a 15 m diameter high-rate thickener. Thickener supernatant flows to a thickener overflow process water tank, while thickener surface froth is captured and discharged to a floor sump for return to thickener feed. Thickener underflow at a nominal density of 65 to 70 per cent solids is pumped to a mechanically agitated filter feed tank of approximately 24-hour surge capacity. The thickened concentrate slurry is dewatered using a 64-plate horizontal filter, with filter discharging into a covered storage shed. Concentrate is loaded into 20 t containers for transport by truck to the Sriracha port in Thailand.

Concentrator raw water is harvested from the Nam Mo River before being pumped to a crusher process water tank and mill header tank. The raw water is mainly used for cooling, pump glands, flotation froth wash showers, and fire water. Process water is recovered from the TSF supernatant, and transferred to a process water tank via two transfer stations.

A photograph of the 12 Mt/a concentrator in June 2008 is shown in Figure 3.

PROCESS IMPROVEMENTS AND FLOW SHEET DEVELOPMENT

Flotation cell conversions

In September 2009, the existing rougher conditioner and cleaner conditioner tanks were retrofitted with flotation mechanisms, thereby increasing the roughing capacity from eight to nine 200 m³ cells, and increasing the cleaner capacity from six to seven 70 m³ cells. The benefits arising from these changes amounted to increased copper recovery in the rougher flotation circuit by 3.5 per cent, and increased copper recovery in the cleaner circuit by 2.5 per cent. The increased residence time in each of the circuits, resulted in higher recovery of slow

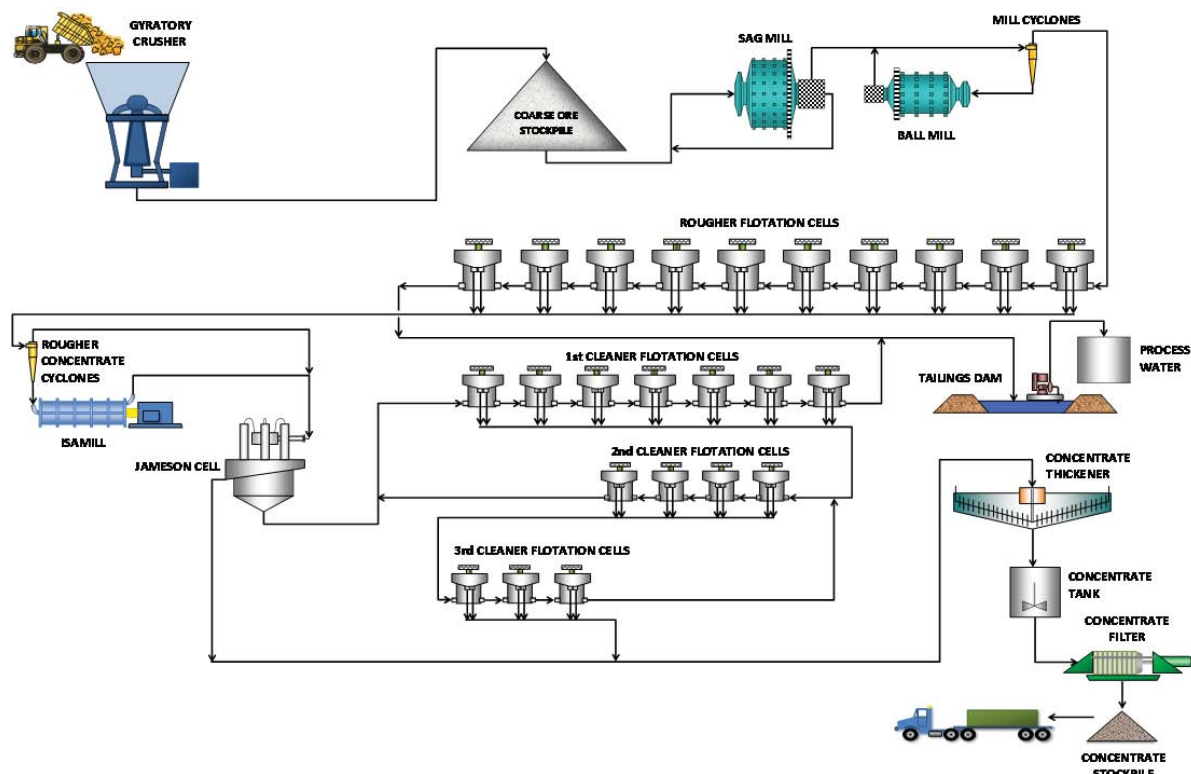


FIG 2 - 2011 Phu Kham 12 Mt/a concentrator simplified flow diagram.



FIG 3 - Southern view of the Phu Kham concentrator, June 2008.

floating secondary copper minerals, and composite particles particularly while transitional ore types were dominant at this time. The improved recovery performance was validated from using the database of daily rougher tail and cleaner scavenger tail refloatation tests.

FloatForce™ mechanisms

An investigation into using Outotec FloatForce™ rotor-stator mechanisms in the rougher flotation circuit commenced in 2009, starting with the first rougher cell 1B. The design of the new mechanism was to deliver improved recovery from increased mixing efficiency, by not allowing any air in the central mixing area of the impellor thereby improving mixing efficiency without affecting slurry pumping. The installation of the FloatForce™ mechanism was simple, and was commissioned without any issues. The conclusions, drawn from extensive survey data around rougher cell 1B (before and after installation), demonstrated an improvement in rougher cell 1B copper recovery of 0.2 per cent. On the basis of the survey data, the total predicted rougher copper recovery increase for the nine rougher cells with FloatForce™ mechanisms was 0.3 per cent. The installation period for the remaining eight mechanisms was completed in June 2011.

Tenth rougher cell installation

The ongoing rougher tail re-flotation tests continued to highlight that there was a further copper recovery benefit of approximately 0.6 per cent to be gained with the addition of another 200 m³ tank cell, which would increase rougher residence time by 2.4 minutes at maximum design throughput of 1750 t/h. In late January 2011, the tenth 200m³ rougher tank cell was installed at the head of the rougher circuit. At the time that the tenth rougher cell was installed, the flotation feed rate was increased by about four per cent, which effectively reduced the overall rougher residence time increase from an expected 11 per cent to seven per cent. The overall surveyed copper recovery improvement was between 0.4 per cent and 0.6 per cent, depending on throughput rate, which met the project criteria.

Cleaner circuit debottlenecking

One of the major limitations in the original plant design had been a lack of cleaning capacity, particularly with respect to the second cleaner bank of four 20 m³ cells. The limitation was that of carrying capacity, rather than residence time. The cleaner circuit performance would generally deteriorate when the cleaner feed copper metal units exceeded 9.1 t/h copper, limiting copper metal production to a sustainable maximum of approximately 8.5 t/h.

To further investigate the cleaning circuit capacity, a cleaner circuit optimisation study was completed in February 2010. From plant data, a mineral based floatability component model was developed which allowed different cleaner circuit configurations to be simulated. The option which gave the optimum copper grade and recovery result was to install additional cleaning capacity ahead of the existing cleaner circuit, so effectively cleaner feed scalping. Different flotation cell technologies were considered for this application, with the Xstrata Jameson Cell meeting design criteria. The simulations indicated that a 0.6 per cent improvement in cleaner recovery could be achieved. The Jameson Cell was chosen because of low installed cost, simulated performance, low performance risk, moderate installation risk, and low production continuity risk during installation.

The circuit simulations, including the Jameson Cell as a cleaner feed scalper, indicated substantial recovery improvement over the existing circuit at circuit feed rates greater than 150 t/h. This was due to the elimination of the carrying capacity limitation as shown in Figure 4.

On this basis it was decided to proceed with this design, the Jameson Cell in a cleaner feed scalping simulation recovering approximately 60 per cent of the copper present in the cleaner circuit feed across a cleaner circuit feed rate range of 100t/h to 300 t/h. The Jameson Cell concentrate grade from the simulation was 27 per cent copper, against a target of 25 per cent copper. The simulations showed that substantial unloading of the remainder of the cleaner circuit would occur. The result of this was that the third cleaner concentrate grade was low at less than 22 per cent copper; however, the net effect was to produce an overall circuit final concentrate grade of 24 per cent copper.

Although there was a small cleaner recovery improvement shown from the simulations performed using a cleaner scalper, the real benefit is in maintaining cleaner recovery when the cleaner feed rate is greater than 150 t/h.

The cleaner scalper cell was commissioned in March 2011. Commissioning was carried out over a period of one week, and no significant problems were encountered. The performance evaluation of the Jameson Cell was remarkably consistent with the expected performance from the equipment vendor, the simulation data, and from Phu Kham Metallurgical Laboratory flotation tests simulating performance of the Jameson Cell prior to commissioning. From surveys carried out in February 2012, with the Jameson Cell online and off-line, the benefit of having the Jameson Cell in circuit was determined to be 0.8 per cent increase in copper recovery.

In terms of overall cleaner circuit debottlenecking, the objectives have been achieved. The cleaner circuit with cleaner feed scalping capacity is 10.1 t/h copper metal at 24 per cent concentrate grade, which is the current limit of the concentrate filtration circuit, for a total 16 per cent copper metal production increase.

Phu Kham Upgrade Project

The Phu Kham Upgrade Project commenced in March 2010 with a study to develop designs to ensure copper in concentrate production is maintained over 60 kt/a after 2013 when plant copper feed grade is expected to decrease. In order to maintain copper metal production, plant nominal design throughput will increase from 12 to 16 Mt/a (2000 t/h). Maximum instantaneous design throughput for the upgraded plant is 2250 t/h. The plant upgrade concept was not original, and had been studied in 2008 as part of a copper production expansion project. Key aspects of the upgrade designs for Phu Kham were the limitation of available space for additional

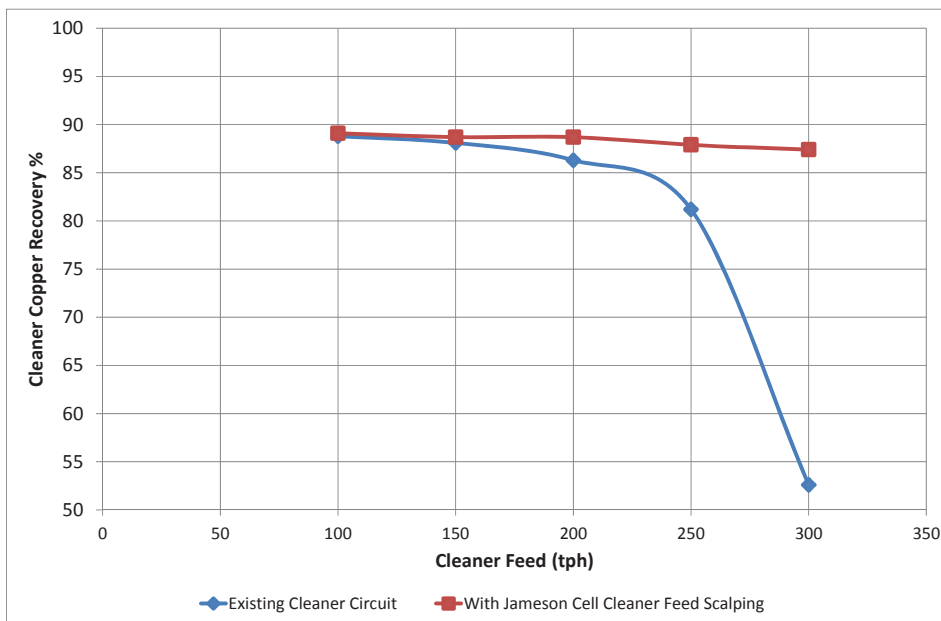


FIG 4 - Effect of cleaner circuit feed tonnage on copper recovery for Jameson Cell Cleaner feed scalping simulations.

equipment, as the original 12 Mt/a plant design had not specifically made allowance for any expandability.

The initial phase of the upgrade included a plant debottlenecking study, which consisted of analysis of the actual plant performance and capacity data from 2008 to March 2010 against the original plant process design criteria. The purpose of the bottleneck study was to determine aspects of the original plant that either were, or would become bottlenecks with the 25 per cent increase in mill throughput. The key findings from the plant bottleneck study are shown in Figure 5, which shows that rougher copper recovery was 16 per cent below design, cleaner copper recovery was seven per cent below design, and mill throughput was three per cent below design at 12 Mt/a. The mill throughput variance was a function of rougher copper recovery and cleaning circuit capacity rather than limitations in the grinding circuit.

The crushing and concentrate dewatering plant capacities were also considered during the upgrade bottleneck study. The bottleneck study indicated that additional crushing capacity would be required with capability for handling wet and sticky

ore, which is a common feature of the transition zones of the orebody. A mineral sizer in parallel to the existing crushing plant, with product reporting directly to the coarse ore stockpile was included in the upgrade designs. Although the concentrate thickener and filter performance had not indicated that future concentrate production rate would exceed capacity, limited data was available to confirm the capacity against upgraded plant design criteria. Test work was conducted to determine settling rates and filtration rates for concentrate during the upgrade project to obtain the required data.

The basis of design for the grinding circuit upgrade has been previously described by Hadaway and Bennett (2011). Two options for increasing grinding circuit throughput after the SAG mill to a nominal design of 16 Mt/a primary grind at 80 per cent passing 106 µm or 75 µm were reviewed. The first option was based on the original 2008 plant upgrade design incorporating an additional 6.5 MW single pinion ball mill, and the second for another 13 MW ball mill.

Data from JKTech grinding circuit modelling in 2009 was extrapolated using the Phu Kham mine schedule to

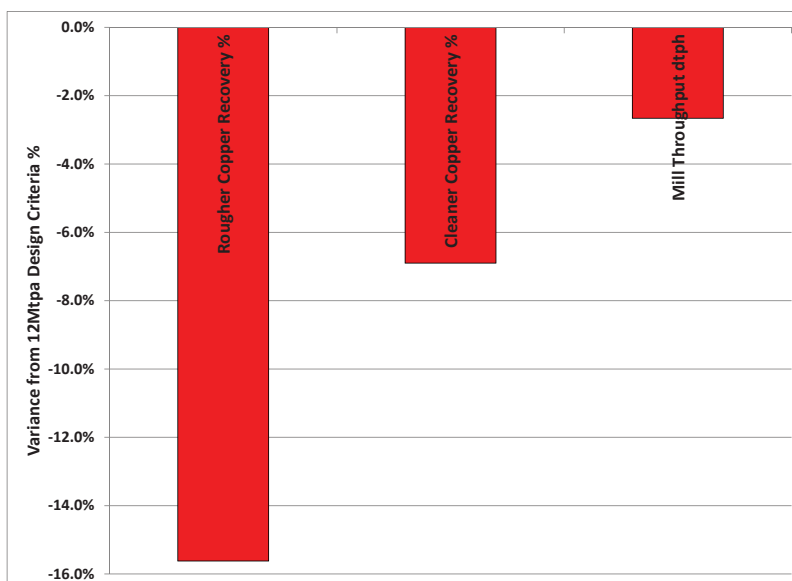


FIG 5 - Phu Kham 12 Mt/a concentrator actual performance variances against design, 2008 - 2010.

determine throughput estimates at 106 µm and 75 µm for the two mill options. Pebble crushing was not considered in the evaluation. The 6.5 MW mill had previously been shown to be able to increase throughput at the 106 µm primary grind to above the 16 Mt/a nominal design, however, was unable to meet the 16Mt/a target at a significantly finer primary grind. The 13 MW mill is able to achieve above 18 Mt/a for the 106µm primary grind, and will achieve above nominal design throughput at a 75 µm primary grind.

The effect of primary grind on flotation recovery was reviewed based on feasibility study work from bench scale batch tests in 2005. The study work indicated that the major primary ore sources, in particular stockwork primary, were relatively insensitive to primary grind size. Plant operations mineralogy data from 2008 to 2011 monthly composites indicates that minor sensitivity exists, with increases of over five per cent in copper sulfide liberation with a primary grind size decrease from 80 per cent passing 106 µm to 75 µm.

An economic analysis was conducted based on differences in capital and operating costs for the two options at 16 Mt/a throughput and 106 µm and 75 µm primary grind. The increase in operating cost for finer grinding versus revenue benefits in copper recovery showed that above \$2.50/lb copper price the finer primary grind increased gross margin. Capital cost per installed megawatt was 26 per cent less for the 13 MW mill option, and the capital payback period for the 13 MW option was significantly shorter.

A risk assessment was conducted for the 6.5 MW option and the 13 MW option. The risk of the 13 MW option was considerably lower than for the 6.5 MW option, mainly due to the operating flexibility for periods of low-grade ore and ore types with higher sensitivity of recovery to primary grind. The 6.5 MW option was not able to take advantage of economies of scale gained by increased throughput, or the estimated one per cent increase in copper recovery at the finer grinds, and would not reach the nominal 16 Mt/a throughput at 106 µm primary grind after 2014. The throughput at 106 µm and 75 µm primary grinds for the two mill options is shown in Figure 6.

Based on the results of the risk assessment, the recommendation for installation of an additional 13 MW ball was accepted. Procurement of a second dual pinion 13 MW drive 40 ft × 24 ft ball mill commenced in November 2010.

The dominant cause of the 15 per cent rougher copper recovery shortfall shown in Figure 5 was a combination of lower than design rougher residence time due to five per cent lower rougher feed density, and a cleaner circuit capacity constraint which limited rougher mass recovery. The dominance of transition ores with significant slow-floating secondary copper mineral content milled during the March 2009 to February 2010 period and the under-representation of these ore types in the feasibility study test work provide explanation for some of the copper recovery shortfall in cleaning stages against design. The debottlenecking study was developed for the flotation circuit to determine increased capacity requirement at the design 16Mt/a upgrade throughput.

Rougher flotation feed density design for the 12 Mt/a plant was 35 per cent solids. Actual operation rougher feed density averaged 30 per cent solids due to the higher slurry viscosity from kaolinite clay content not quantified during the feasibility study. An extra 200 m³ rougher cell was required to achieve the same residence time as at 35 per cent solids, which was achieved by conversion of the rougher conditioning tank to a cell in 2009. The reduced residence time from operating at the lower rougher density at design tonnage throughput resulted in a three per cent decrease in copper recovery, based upon plant residence time – recovery data from July 2009 to February 2010. The upgrade design therefore allowed for reduced rougher feed pulp density, and a residence time calculation confirmed that a 33 per cent increase in rougher capacity was required for the 25 per cent increase in mill throughput at 16 Mt/a, which would also provide an additional one per cent copper recovery. A total of five 200 m³ rougher cells in addition to the existing ten cells were included in the design, for a total of 15 cells.

The cleaning circuit was not expected to require significant expansion as a result of the 16 Mt/a upgrade, as the lower-grade mill feed would result in equivalent concentrate production to the 12 Mt/a design throughput rate. The upgrade design for the cleaner flotation circuit also included the Jameson Cell cleaner scalper although this had not been installed at this time. However, cleaner circuit mass balance simulation data including the Jameson Cell indicated that 40 per cent increase in the existing second cleaner residence time and lip length was required at 16 Mt/a. To gain this increase in second cleaner capacity, the existing three 20 m³

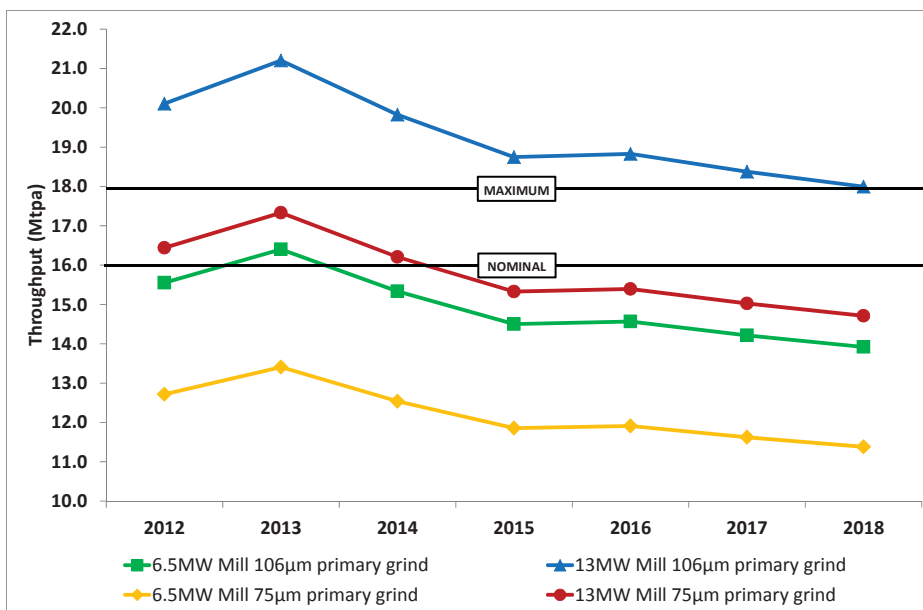


FIG 6 - Throughput at 106 and 75 µm primary grind for additional 6.5 MW and 13 MW ball mill options.

third cleaner cells have been combined with the four 20 m³ second cleaner cells, and four new 20 m³ third cleaner cells added to for the upgrade. The simulation data for the cleaners also demonstrated that upgrade of the first cleaner capacity was not warranted, as the fine low-grade middlings recovered in the final cells were not able to be upgraded to near final concentrate specification.

Prior to commencement of the detailed design phase of the upgrade study, maximum sustainable production rate (MSPR) analyses were undertaken for the crushing and concentrate dewatering plants to determine whether capacity expansion was required for these areas of plant based on 16 Mt/a production schedules. The MSPR was defined as the best consecutive five days of performance, normalised using plant-specific industry standards for annual availability to allow for major scheduled maintenance. The MSPR for the crushing plant also included seasonal variation due to the tropical environment and the wet season impacts on crusher productivity.

The primary conclusions from the performance review of the Phu Kham crushing plant were that it had demonstrated the target upgrade production rate of 16 Mt/a over the June 2010 period, and approximately one third of total crushing plant downtime had been caused by events up and downstream of the crushing plant while the plant was available to crush. The low crusher utilisation of 64 per cent was equivalent to over 2.4 Mt/a of crushing capacity at the target throughput of 2400 t/h and target utilisation of 75 per cent of total time. With increases in haul fleet numbers for the upgrade, improvements in run-of-mine stockpile inventory, and a stand-by loader available when there were delays in truck presentation to the crusher, the MSPR demonstrated that increasing crushing capacity was not required.

The design specifications for the 64-plate and frame filter were for a filtration rate of 225 kg/m²/h, with an annual design production rate of 311 000 t of concentrate.

Actual filter plant operating data was analysed to check filter performance against the design capacity. MSPR for the filter was determined to be 18 per cent above the life-of-mine maximum concentrate production schedule, leading to deferral of capital expenditure for the filtration plant. The main reasons for the higher than design performance were; optimisation of filter cycle settings following an improvement program including operations, maintenance, and vendor support input, and change to filter cloth media type.

Following the review of the upgrade design, engineering and procurement services commenced for the Phu Kham 16 Mt/a Upgrade project in January 2011, with commissioning scheduled for the third quarter of 2012. A simplified flow diagram for the upgraded plant is shown in Figure 7, with new equipment highlighted in mauve.

Increased recovery project

The Phu Kham feasibility studies between 2004 and 2006 identified two options for flotation processing of Phu Kham ore. The first option involved bulk flotation of the rougher feed targeting a 25 per cent mass recovery into rougher concentrate using non-selective amyl xanthate sulfide mineral collector. The rougher concentrate was then regrinded to 80 per cent passing 38 µm and subjected to cleaner flotation at a pH of 12 for pyrite depression. This process produced high copper recovery results, however, there was difficulty achieving final concentrate grade of greater than 22 per cent copper across all ore types, particularly transition chalcocite-covellite secondary copper mineral dominant ores. A rougher feed photomicrograph (Shouldice and Mehrfert, 2009) is shown in Figure 8 with chalcocite-covellite intergrowth with pyrite and rimming of pyrite. There was also indication of copper activation of pyrite from soluble copper species in weathered and transition ores.

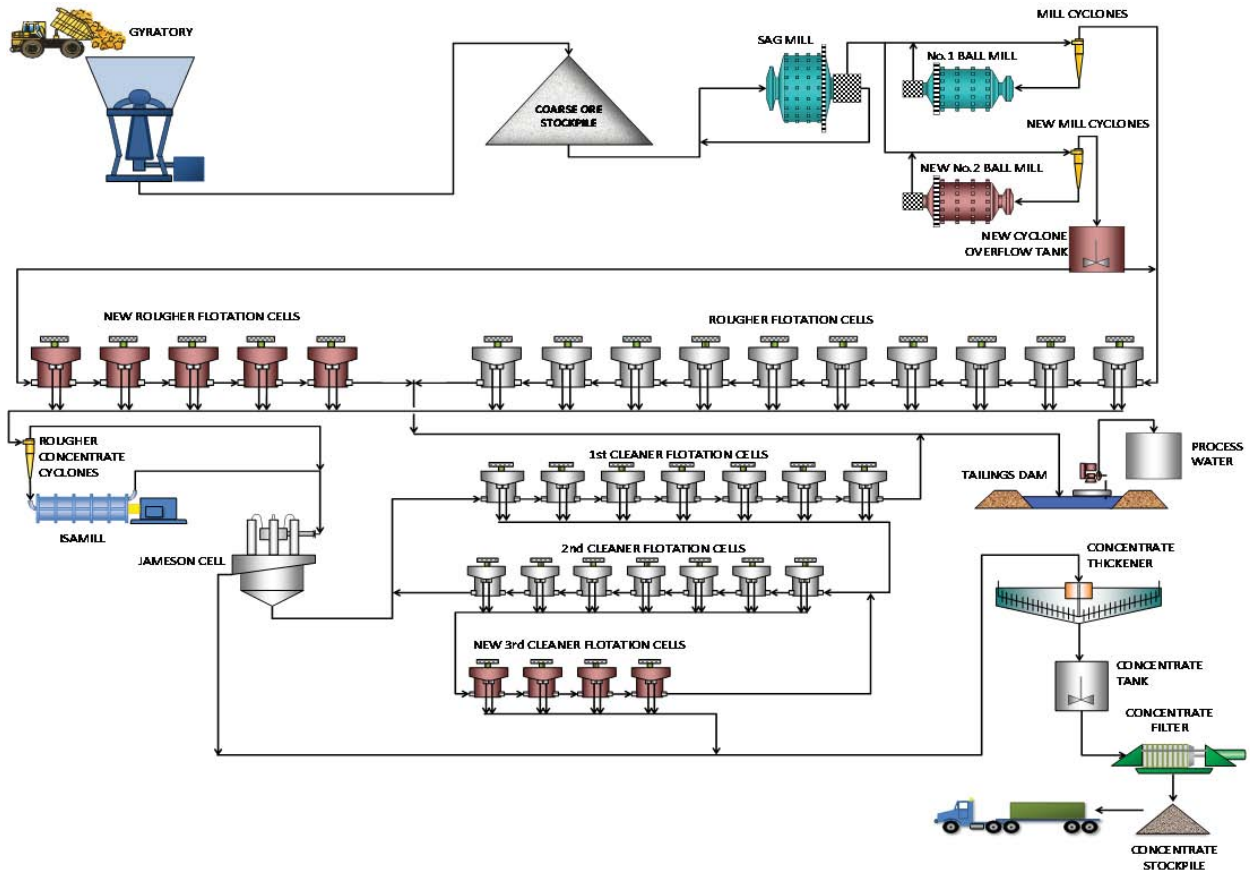


FIG 7 - Phu Kham 16 Mt/a upgrade simplified flow diagram.

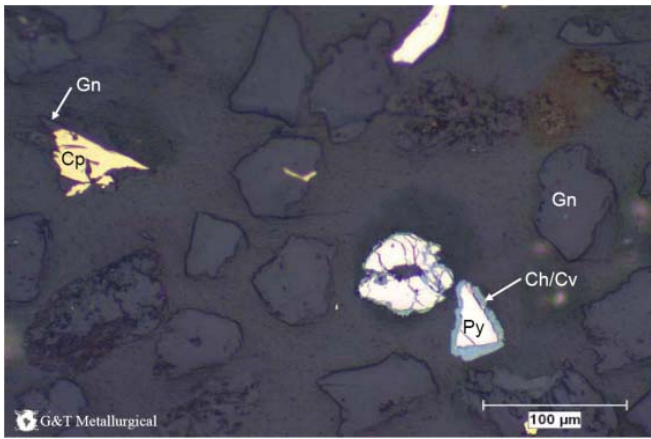


FIG 8 - Phu Kham transition ore rougher feed photomicrograph.

The second process option involved selective flotation in roughing at pH 11 - 12, using a copper sulfide selective collector. The rougher concentrates were again reground to 80 per cent passing 38 μm and lime to pH 12 and sodium cyanide was added to the cleaning stages to depress pyrite. The selective flotation option consistently achieved over 22 per cent copper final concentrate grade; however, ultimate copper recoveries to final concentrate were lower than the bulk flotation option.

The design of the original 12 Mt/a Phu Kham concentrator was a compromise between the two process options, with partially selective roughing being applied to minimise pyrite gangue recovery into cleaner flotation feed. Sodium cyanide addition to the cleaners was included in the design, however, has never been used with concentrate grade over 22 per cent copper consistently achieved since commissioning. This partially selective flotation process had significant capital cost advantages over bulk flotation at a time when the long-term copper price estimate was much lower than 2012 prices, due to the lower rougher concentrate regrind and cleaning capacity required, and provided the best cost-benefit process alternative while reducing risk of being unable to achieve concentrate specifications using bulk rougher flotation.

Minimal work was performed during the feasibility studies to test the sensitivity of final copper grade and recovery on rougher concentrate regrind product particle size. Grind size analysis was limited to two mineralogical examinations which concluded that reasonable copper and

gold recoveries to rougher concentrate would result from a primary grind of 80 per cent passing 106 μm, and that a rougher concentrate regrind to less than 45 μm was required to achieve an acceptable final concentrate grade. Mineral and liberation analysis showed that associations between copper sulfide minerals and pyrite did not indicate complex or fine intergrowths that would adversely impact on the metallurgy.

The Phu Kham increased recovery project (IRP) commenced in 2009, as part of a concept study to develop a process to increase copper and gold recovery from Phu Kham ore. Since commencement of operations in 2008, copper recovery had increased with increasing proportion of chalcocypirite-dominant primary ores replacing the chalcocite-covellite secondary copper mineral dominant high clay and talc transition ores. The increasing plant throughput and poor primary liberation with increasing pyrite content has caused copper and gold recovery to 'flat-line' as shown in Figure 9.

Bulk sulfide flotation bench tests in 2009 using isopropyl xanthate collector on Phu Kham rougher tailings indicated that up to ten per cent additional copper recovery and 70 per cent additional gold recovery could be achieved into a scavenger concentrate of approximately 0.8 per cent copper. The initial test program was designed to determine whether a low-grade copper-gold concentrate suitable for downstream hydrometallurgical processing to saleable products could be recovered from the concentrator tailings.

The preliminary test program showed that a bulk sulfide concentrate from plant tailings flotation could be upgraded to over ten per cent copper concentrate grade, depending upon copper sulfide mineral liberation, using a roughing, regrind, and two stage cleaning process similar to the Phu Kham concentrator process. Figure 10 shows the copper grade - recovery relationships with varying rougher concentrate regrind power input of 10 kWh/t, 20 kWh/t and 40 kWh/t.

The test results presented in Figure 10 clearly demonstrated that finer regrinding of rougher concentrates from plant tailings flotation would improve both copper grade and recovery. Further flotation tests on plant final tailings samples using roughing at pH 9 with amyl xanthate, followed by regrinding of concentrate at 10 kWh/t power input, and two stages of cleaning consistently produced a low-grade flotation concentrate of approximately three per cent copper and 2 g/t gold. Average copper recovery was 69.6 per cent and average gold recovery was 54.1 per cent from 24 flotation tests as shown in Table 1.

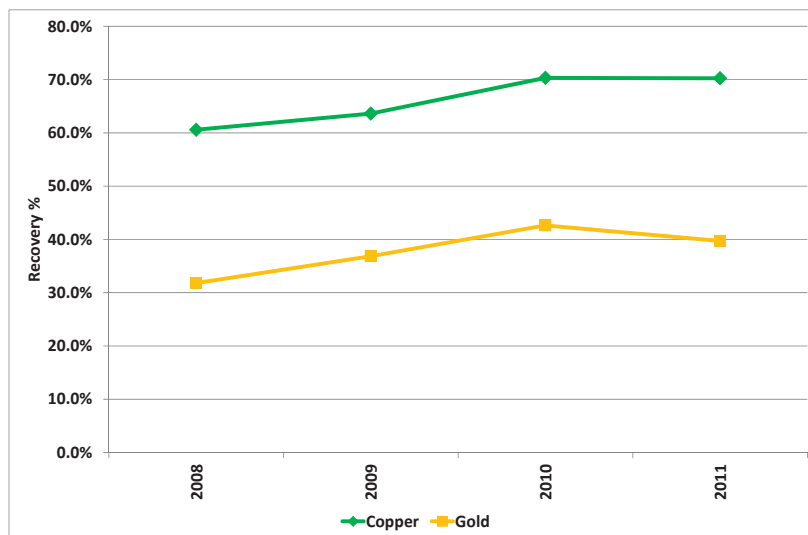


FIG 9 - Phu Kham concentrator copper and gold recovery by year, 2008 - 2011.

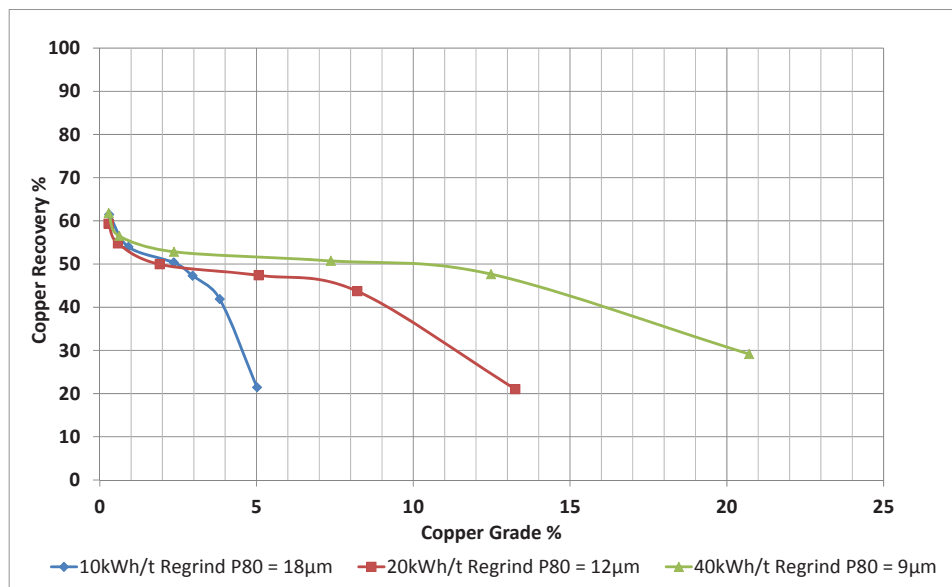


FIG 10 - Phu Kham tailings flotation rougher concentrate regrind power-grade-recovery relationships.

TABLE 1
Plant tailings flotation test results summary.

Stream	Copper		Gold	
	Grade % Cu	Distribution %	Grade Au ppm	Distribution %
Plant tailings (feed)	0.17	100.0	0.15	100.0
Concentrate	2.81	69.6	1.87	54.1
Final tailings	0.05	30.4	0.07	45.9

The downstream extraction of copper and gold from the low-grade pyrite-rich concentrate produced by the bulk tailings flotation was developed to prefeasibility level during 2010 and 2011, using Xstrata’s Albion atmospheric leaching technology. The Albion test work showed that copper extraction of over 95 per cent using acid Albion leaching, and gold extraction of over 85 per cent using a standard carbon-in-leach process was achievable from a flotation concentrate of three per cent copper and 2 g/t gold, providing an overall 15 per cent copper recovery and 25 per cent gold recovery increase for Phu Kham operations.

The prefeasibility study flotation results also indicated that there was potential for increasing copper and gold recovery by mineral processing methods alone, with reduced technical risk and capital intensity compared to hydrometallurgical processing. A detailed mineralogical examination of Phu Kham tailings commenced during the prefeasibility study to improve understanding of the mechanisms of copper and gold loss and develop a simplified process for improving recovery.

Since 2008, monthly plant composites have been submitted to G&T Metallurgical Services for quantitative mineralogical analysis, and this data has provided the critical information used for recovery improvement process development. The mineralogy data demonstrated that since primary chalcopyrite ores had become the dominant source of plant feed, the major cause of loss of copper in plant tailings had changed from slow floating fine liberated copper minerals (Crnkovic *et al*, 2009) to chalcopyrite locked in poor quality coarse binary particles with non-sulfide gangue. The copper loss in plant tailings from November 2011, representing a typical month, is shown in Figure 11.

A digital photomicrograph of flotation tailings is shown in Figure 12 (Shouldice and Ma, 2009). The wide size range of the chalcopyrite particles in non-sulfide gangue is evident.

Copper sulfide mineral grain size data for rougher tailings is presented in Table 2. The data shows that under selective rougher flotation conditions, recovery of coarse low-quality binary copper sulfide and gangue composite particles is poor due to the fine copper sulfide grain size.

Gold recovery by flotation at Phu Kham has been poor since commissioning, averaging approximately 40 per cent to final copper concentrate product. Prediction of gold recovery had also been demonstrated to be inaccurate during plant operation, due to a lack of understanding of the key mineralogical characteristics of gold occurrence and the variability of gold occurrence across different Phu Kham mineral assemblages. As part of the increased recovery project, the mineralogical reasons for gold loss into Phu Kham tailings were examined to ensure potential opportunities to increase gold recovery into final concentrate were included in IRP process design.

Diagnostic leach tests were conducted on Phu Kham plant tailings samples. Results of the diagnostic leaching are presented in Table 3.

The diagnostic leaching results in Table 3 demonstrated that over 60 per cent of the gold in tailings was available for cyanide leaching, either as free gold or partially liberated gold. This was supported by the Albion gold leach test work on acid Albion copper leach residues, which demonstrated that extraction of gold to solution did not significantly increase with increasing sulfide sulfur oxidation, as shown in Figure 13.

Laser ablation testing was conducted on the rougher tailings to determine the proportion of gold locked with pyrite and other sulfide mineral species. The laser ablation test results were combined with the results from the diagnostic leaching to provide a total gold association for the rougher tailings as presented in Table 4.

Diagnostic leaching provided a measure of the unlocked (cyanide soluble) and locked gold department, but the total unlocked gold could not be split between fully liberated gold particles, and partially liberated (exposed in composites) gold particles. Therefore the diagnostic leach measure of 57 per cent cyanide soluble gold in Table 4 could not provide definitive information for the cause of gold loss to tailings.

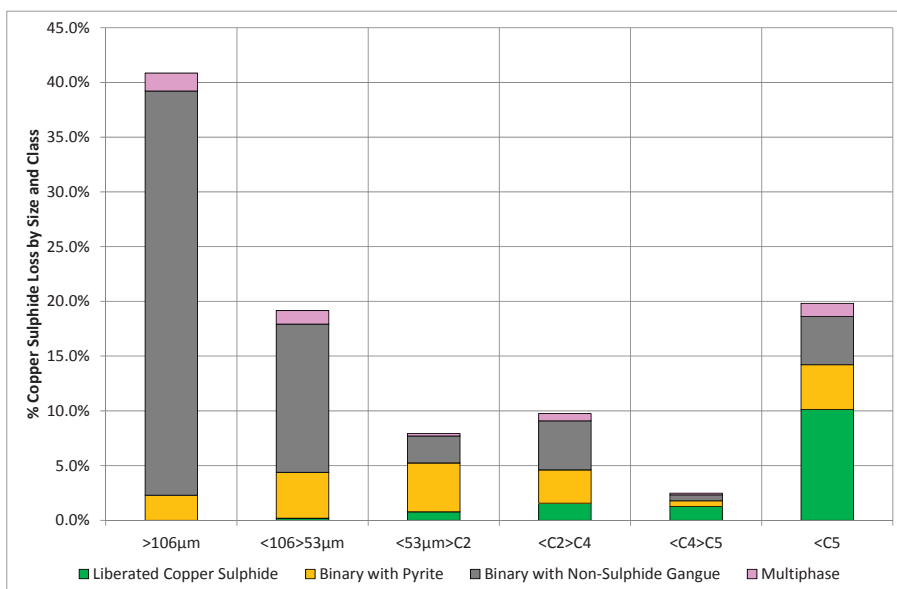


FIG 11 - Copper sulphide loss in Phu Kham flotation tailings by size and mineral association.

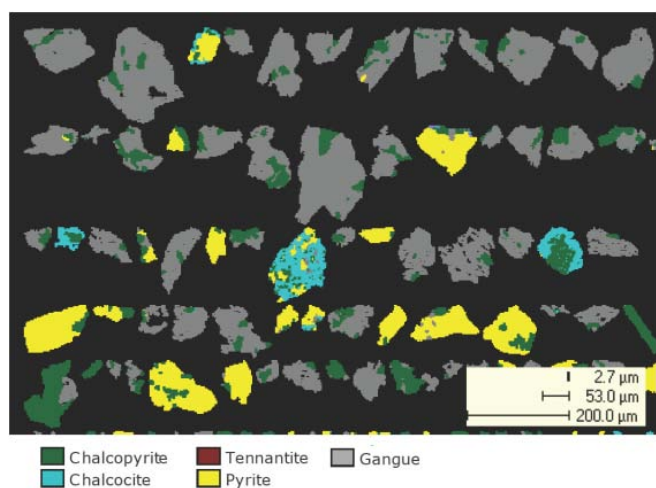


FIG 12 - Phu Kham flotation tailings photomicrograph.

TABLE 2

Rougher tailings copper sulphide grain size by size fraction.

Size fraction	Copper sulphide grain size (µm)
>106 µm	19.6
<106>75 µm	13.0
<75>53 µm	11.4
<53>38 µm	9.1
<38 µm>C2	9.1
<C2>C4	9.8
<C4>C5	8.1
<C5	3.7

Final Phu Kham copper concentrate monthly plant composites were analysed by automated digital image scanning (ADIS) at G&T Metallurgical Services to determine the characteristics of gold and gold composite particles recovered in flotation (Shouldice and Johnston, 2012). Laboratory work on rougher tailings sample from the IRP laboratory test work was also conducted to produce a gold-rich concentrate by gravity concentration suitable for ADIS and photomicrograph analysis.

TABLE 3

Results of diagnostic gold leaching of Phu Kham flotation tailings.

Plant tailings stream	% Gold cyanide soluble	% Gold locked in sulfide minerals	% Gold locked in non-sulfide gangue
Cleaner tailings	72	21	7
Rougher tailings	57	42	1
Final tailings	64	35	1

The ADIS work on the final concentrate showed an average gold particle size of 7.9 µm. The class and mass distribution summary and area as a percentage of the observed gold particles are presented in Table 5.

Binary particles of gold locked with pyrite in concentrate were of particular interest. Only eight per cent of the observed particles were gold-pyrite binary composite particles; however, 71 per cent of the total mass of observed gold was in these particles. The average surface area of the gold in the gold-pyrite binary particles was 87 per cent of the total particle surface area. An interpretation of this data indicated that for a gold-pyrite particle to float into concentrate, the gold:pyrite surface area ratio must be sufficiently large to overcome the depression of the attached pyrite particle under high pH flotation cleaning conditions. The liberated gold recovered was typically fine, with an average particle size of 7 µm.

Photomicrographs of the gold showing some typical particles in concentrate are presented in Figure 14.

The ADIS work found that the average gold particle size was 19 µm in rougher tailing, approximately 13 times larger than the average gold particle in final concentrate. No binary particles were observed, only liberated gold and gold-chalcopyrite-pyrite-gangue multiphase particles as summarised by mass distribution in Table 6.

The liberated gold particles in tailings had a mean size of 35µm, with the data indicating that liberated gold particles above 20 µm in size are unlikely to be recovered to final flotation concentrate, with a 13 µm particle the largest observed in the concentrate. Cyanide leaching would have extracted 69 per cent of the gold particles observed.

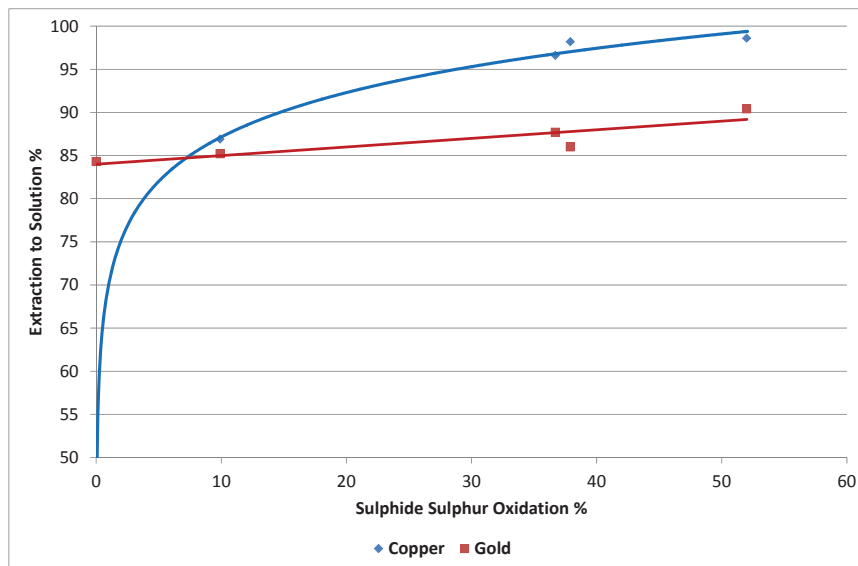


FIG 13 - Copper and gold extraction in Albion leaching as a function of sulfide sulfur oxidation.

TABLE 4

Phu Kham rougher tailings gold department.

Gold mineral association	% of total gold
Cyanide soluble	57
Pyrite locked	25
Other sulfide locked	17
Non-sulfide gangue locked	1
TOTAL	100

The significant characteristic of the multiphase particles observed in tailings was that greater than 95 per cent of the total mass of the particles were gangue mass. The gold in the multiphase particles had an average diameter of 9 μm, while the pyrite gangue had an average diameter of 121 μm.

Photomicrographs of the gold showing some typical particles in rougher tailings are presented in Figure 15.

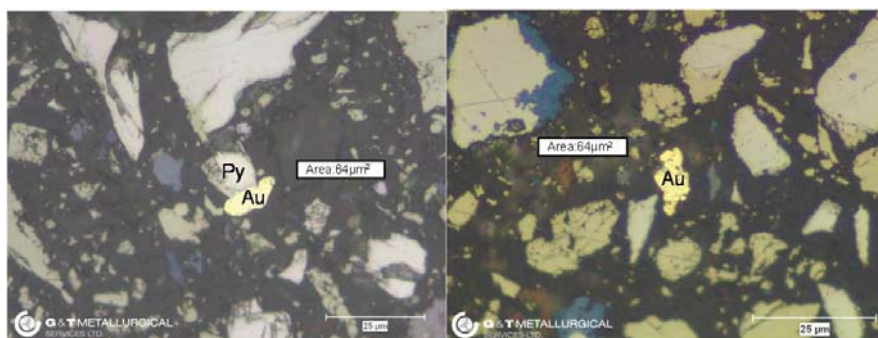


FIG 14 - Phu Kham August 2011 final concentrate gold photomicrographs.

TABLE 5

Phu Kham final concentrate gold distribution data.

Data	Liberated gold	Gold locked in binary particles						Multiphase
		Cp	Ch/Cv	Te	Py	Goe	Gn	
Association class	38%	30%	8%	5%	8%	3%	3%	5%
Gold mass by association class	11%	16%	<1%	1%	71%	<1%	1%	<1%
Particle area percentage gold	100%	73%	52%	51%	87%	89%	98%	9%

Notes: Cp = chalcopyrite, Ch = chalcocite, Cv = covellite, Te = tennantite, Py = pyrite, Goe = goethite, Gn = gangue, MP = multiphase.

Based on the concentrate and rougher tailings ADIS data, a summary of the estimated recoveries of the main Phu Kham ore gold association classes as presented in Table 7.

Two mineral process options for increasing recovery of the low quality copper sulfide – non-sulfide gangue binary particles were tested; mainstream inert grinding, and bulk sulfide flotation. The mainstream inert grinding process is classification of rougher tailings for recovery of the plus 53 μm fraction, regrinding to approximately 80 per cent passing 53 μm, and scavenger flotation. Bench scale test work on the mainstream inert grinding showed that an additional seven per cent recovery of copper and four per cent recovery of gold to final concentrate was achievable, however, the mass recovery of 45 per cent of the plant tailings into the plus 53 μm fraction required approximately 16 MW of additional installed power for regrinding, most of which would be wasted regrinding liberated gangue.

The bulk sulfide flotation process laboratory test work followed the original Phu Kham prefeasibility study process

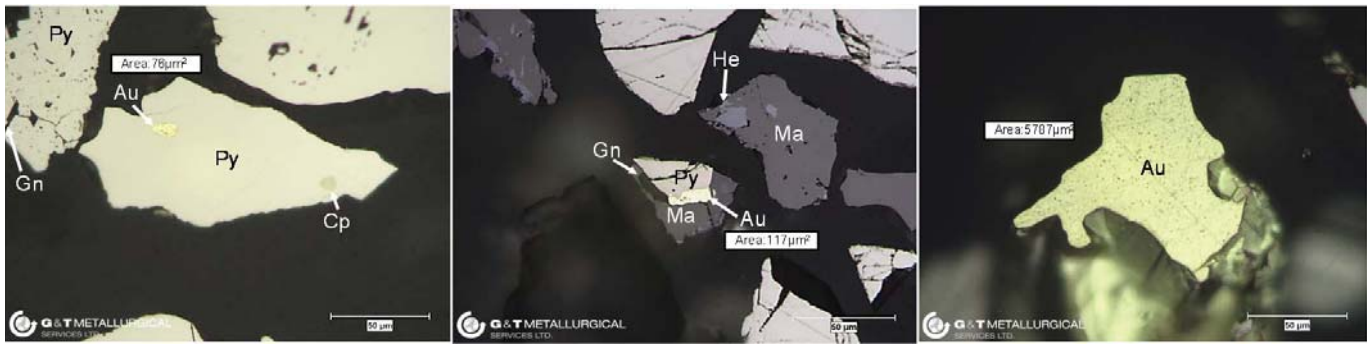


FIG 15 - Phu Kham rougher tailings gold photomicrographs (notes: Au = gold, Cp = chalcopyrite, Py = pyrite, He = hematite, Ma = magnetite, Gn = gangue).

TABLE 6
Flotation rougher tailing gold mode of occurrence, size and mass data.

Gold mode of occurrence	Average particle diameter (µm)				Average % of particle mass			
	Au	Cp	Py	Gn	Au	Cp	Py	Gn
Gold liberated	35				100			
Gold adhesion multiphase	6	17	151	15	<1	<1	99	<1
Gold locked multiphase	8	28	104	27	3	2	80	15
Gold adhesion/locked multiphase	17	14	125	61	2	1	94	3

TABLE 7
Phu Kham indicative gold recovery by mineral association class.

Parameter	Gold association class			
	Liberated Au	Au-copper sulfide	Au-pyrite	Au-gangue
Gold recovery	70%	100%	10%	0%

of maximising copper and gold recovery into a low-grade rougher concentrate using xanthate collector at natural pH. In developing the design for a bulk sulfide rougher, regrind and selective cleaning flotation process, three key parameters were required to be identified; rougher mass recovery to achieve maximum copper recovery across all ore types, rougher concentrate regrind product size, and cleaner capacity.

Rougher mass recovery design was determined by daily rougher flotation testing of plant rougher feed over a three month period in 2011, which allowed variability testing across all major ore types. The rougher tests were conducted using Aero 9810 copper selective collector but at up to 80 percent increase in dose rate to collect poor quality coarse non-sulfide gangue and chalcopyrite composites and maximise copper recovery. Rougher mass recovery and copper recovery relationships were then developed for major ore types to develop nominal and maximum mass recovery designs for IRP regrind and cleaner flotation plant as shown in Figure 16.

The results show that the original design for rougher mass recovery of 11 per cent of rougher feed was suitable for the chalcopyrite dominant primary ore types, but gave lower copper recovery from the high pyrite and non-sulfide copper species in chalcocite-rich transition ores. The results indicated potential for an additional six per cent copper recovery in roughing by increasing mass recovery to a maximum 25 percent of rougher feed for all ore types.

The results for rougher flotation gold recovery from the same test work are shown in Figure 17. The gold recovery increase in roughing with the higher mass recovery is more consistent across primary, high pyrite and transition ore types than for copper, which reflects the association of gold with

copper sulfides and pyrite. The mineralogical analysis of gold deportment and bench-scale test work results concluded that with the design rougher mass recovery increase and finer regrind, gold recovery to final concentrate could be expected to rise by at least six per cent.

Rougher concentrate regrind size optimisation work commenced in 2011 to support the IRP process design. The benefits for copper recovery of finer grinding of concentrates from bulk flotation of Phu Kham tailings had previously been observed as shown in Figure 10. Samples of plant feed were tested at bench scale using roughing, rougher concentrate regrind at increasing power input, and three stages of cleaning to produce the copper and gold grade – recovery response curves in Figure 18. The increase in rougher concentrate regrind power input shows the benefit of the increased liberation for both copper and gold grade and recovery into flotation concentrates.

With the variable mineralogy of Phu Kham ore, the monthly composite mineralogy and laboratory scale test work was used to determine the rougher concentrate regrind size required to achieve maximum copper sulfide liberation in cleaner feed, with a target of 80 per cent copper sulfide liberation considered to be required to maximise recovery and maintain final concentrate specification. Scan data on two samples representing different Phu Kham ore types in Figure 19 showed that maximum copper sulfide liberation was typically achieved at 20 µm particle size, although 80 per cent liberation was not necessarily achieved for all ore types.

Power input to achieve 20 µm regrind product size was calculated from laboratory regrind signature plot data, and daily surveys of the existing M10000 IsaMill™ to be 18 -25 kWh/t, with an additional 3 MW M10000 IsaMill™ included in the IRP design to provide a total of 6 MW power input at the maximum rougher concentrate mass recovery of 25 per cent.

First cleaner design

The increase in rougher concentrate mass recovery to 25 percent of rougher feed, and the reduction in cleaner feed particle size to 20 µm required a corresponding increase in first cleaner flotation capacity. The 12 Mt/a plant had 25 minutes

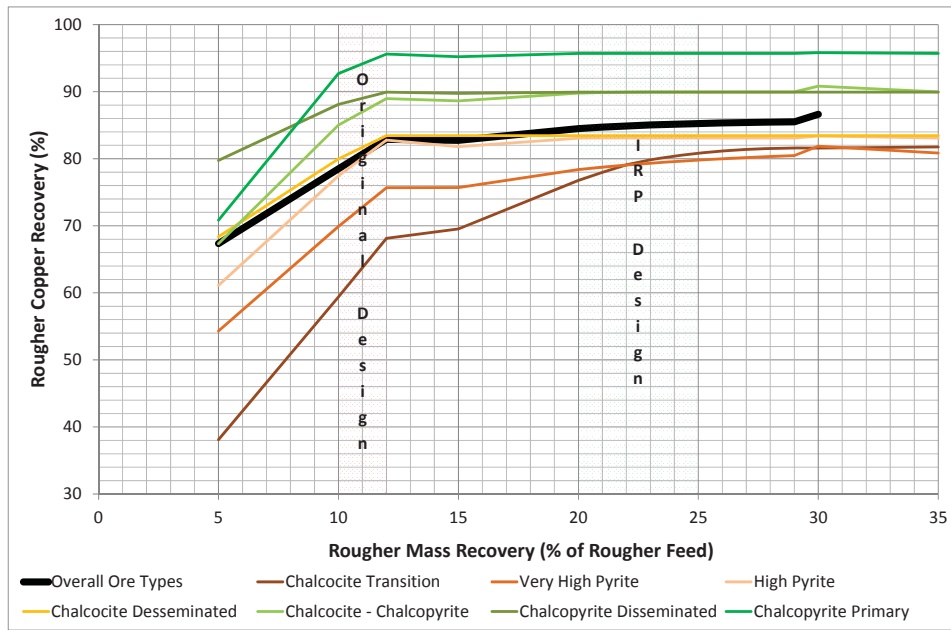


FIG 16 - Rougher copper recovery and mass recovery for major ore types showing original mass recovery design and increased recovery project mass recovery design.

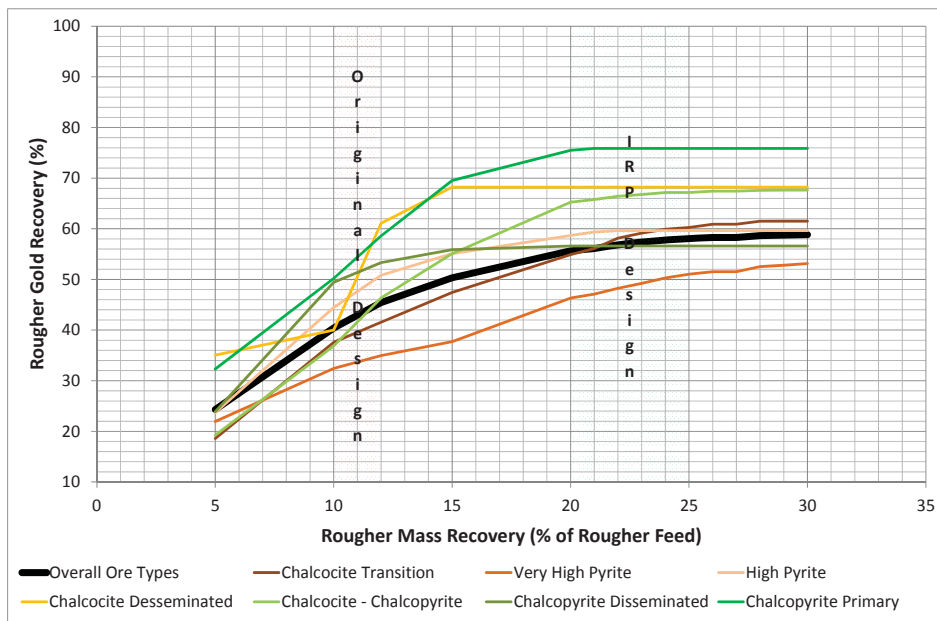


FIG 17 - Rougher gold recovery and mass recovery for major ore types showing original mass recovery design and increased recovery project mass recovery design.

total first cleaner residence time for a 38 µm cleaner feed size, and test work was designed to determine whether this residence time needed to increase at the 20 µm cleaner feed size due to potentially reduced kinetics of the finer particles. Bench scale flotation cleaning tests and cleaner circuit model simulations indicated that flotation kinetics remained similar for copper sulfide minerals due to improved liberation, and that no increase in residence time was required for first cleaning until regrind product size was below 10 µm. Figure 20 shows the plant cleaner circuit chalcopyrite recovery by particle class and size. The recovery of liberated chalcopyrite remains high with decreasing particle size, which confirmed that the increased chalcopyrite liberation from gangue at a 20 µm regrind size will improve copper recovery.

The IRP design includes 100 per cent increase in first cleaner capacity to allow for the increased rougher mass recovery, and lower slurry density to provide improved dilution cleaning to minimise fine gangue particle entrainment.

To validate the extensive test work and mineralogy data results, four IRP process plant trials were conducted at Phu Kham between January 2012 and April 2012. The trial method used was to decrease SAG mill throughput by 50 per cent to avoid overloading the cleaning circuit, reduce rougher cell residence time to the equivalent post-16 Mt/a time of 30 minutes by reducing levels and air in three of the ten cells, and using increased collector and frother addition to increase rougher mass recovery to 25 per cent. The IsaMill™ power draw was increased to 2.8 MW to target a 20 µm regrind product size. Immediately prior to the trial periods, baseline plant surveys were undertaken to obtain comparison data for the same ore type. The key results of the four plant trials are summarised in Table 8.

The first plant trial period was over four hours to ensure critical test criteria could be achieved, in particular rougher mass recovery target of 25 per cent. Although this was a preliminary trial, survey results for the IRP process were

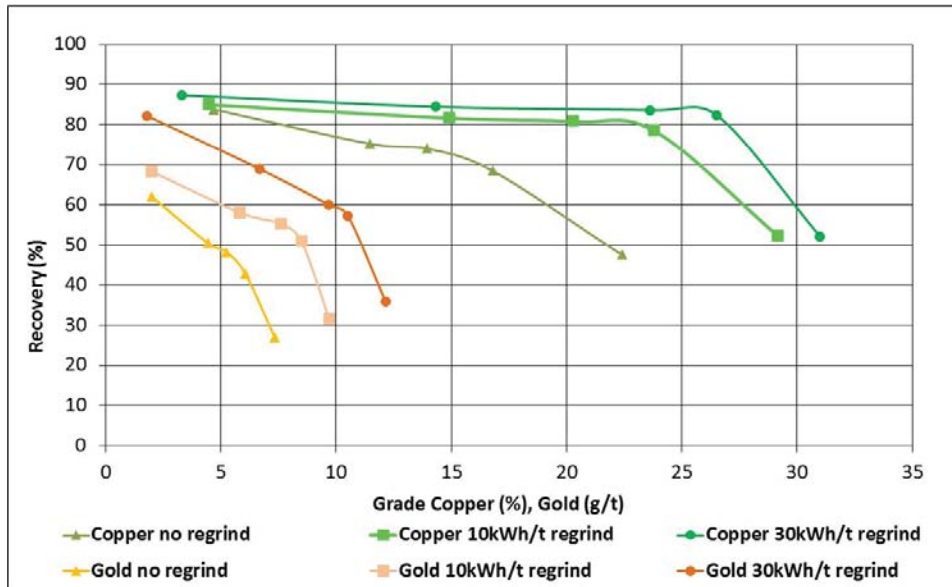


FIG 18 - Copper and gold flotation grade-recovery responses as a function of rougher concentrate regrind power input.

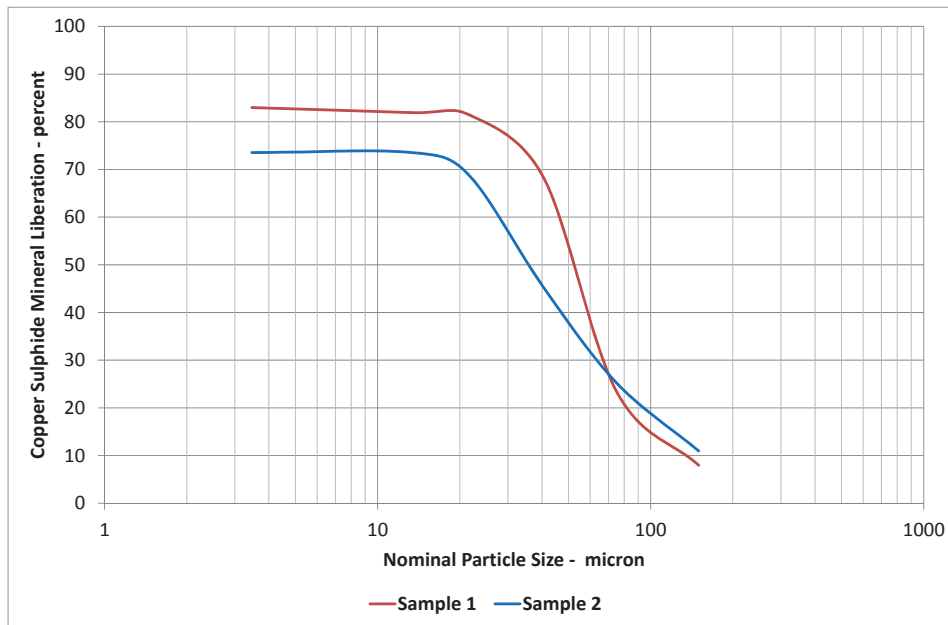


FIG 19 - Copper sulphide mineral liberation with particle size.

positive compared to baseline survey results, with eight per cent overall copper recovery and 19 per cent overall gold recovery achieved into a two per cent higher copper grade final concentrate. The second plant trial was conducted over a full 12-hour shift period, with improvement in both overall copper and gold recovery of 11 per cent at a 0.6 per cent copper grade improvement in final concentrate.

Two short four-hour variability trails were conducted in March and April 2012 on primary ore with a good flotation response and high pyrite ore with a poor flotation response. Copper recovery improvement was consistent at over five per cent for both ores, and gold recovery improvement was 13 per cent and 20 per cent into final concentrate. The short duration of the trials did not allow time for optimisation of cleaner circuit performance. The plant trial results provided significant confidence in the IRP concept and basis of design, which was finalised in May 2012 prior to commencing detailed design. The economic evaluation at six per cent increase in copper and gold recovery after the initial plant

trials provided a compelling investment case, resulting in PanAust Ltd approval in February 2012 for the development of the IRP at Phu Kham.

The simplified process flow diagram for the IRP is shown in Figure 21, with upgrade of the existing rougher concentrate pumping and classification, a second 3 MW M10000 IsaMill™ regrind mill, seven additional 70 m³ first cleaner flotation cells, and upgraded pumping capacity in the cleaner flotation circuit. The increased copper recovery and resultant production of approximately 25 000 t/a additional mineral concentrate at a finer particle size was determined to exceed the existing 64-plate and frame filter MSPR, with an additional 40-plate and frame filter included in the design to meet IRP and future concentrate production output.

CONCLUSIONS

The installed 2008 Phu Kham 12 Mt/a concentrator was a compromise between a high-recovery but high-capital intensity design, and a lower-recovery but technically

TABLE 8
Phu Kham increased recovery project plant trial results.

Preliminary trails	Trial 1: 15 January 2012			Trial 2: 27 January 2012		
	Baseline	Increased recovery project	Difference	Baseline	Increased recovery project	Difference
Rougher recovery (%)	83	92	+9	76	88	+12
Overall copper recovery (%)	71	79	+8	68	79	+11
Overall gold recovery (%)	48	66	+19	45	56	+11
Concentrate grade (%)	22.1	24.1	+2.0	24.2	24.8	+0.6
Variability trails	Trial 3: 21 March 2012			Trial 4: 24 April 2012		
	Baseline	Increased recovery project	Difference	Baseline	Increased recovery project	Difference
Rougher recovery (%)	87	93	+6	81	87	+6
Overall copper recovery (%)	81	87	+6	75	80	+5
Overall gold recovery (%)	40	53	+13	36	56	+20
Concentrate grade (%)	25.7	23.8	-2.0	25.5	21.7	-3.8

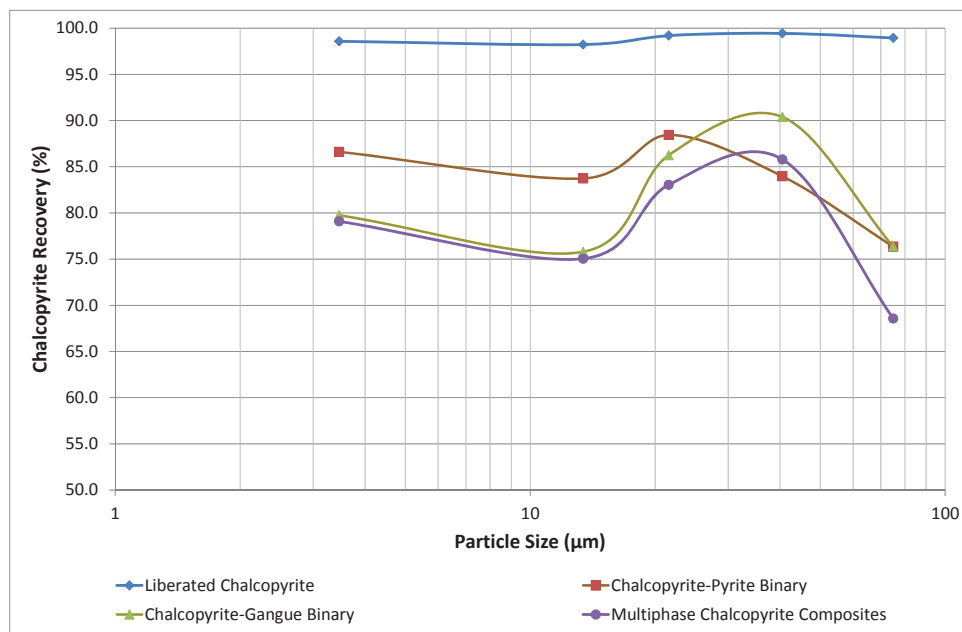


FIG 20 - Phu Kham cleaner circuit chalcopyrite recovery by particle class and size.

low-risk and low-capital intensity design suitable for the prevailing low copper price market. The chosen selective rougher flotation design was driven by the complex and highly variable mineralogy particularly in the transition ore zones, and high pyrite content. Over 90 per cent of pyrite is required to be rejected in order to produce a final concentrate of over 23 per cent copper.

The optimisation of the Phu Kham flow sheet has been driven by low copper and gold recoveries and the requirement to maximise copper production by increasing plant throughput. Flotation circuit improvements and debottlenecking projects including additional roughing capacity and cleaner feed scalping have increased copper metal production by 16 per cent since 2009, while maintaining copper recovery at maximum design throughput of 14 Mt/a.

The plant optimisation and development program and major plant design at Phu Kham has been supported by extensive mineralogy, mineral association, and mineral liberation data from plant monthly composites and bench scale test products.

The collection and analysis of this data has revealed reasons for copper and gold losses and mineral department, and significant opportunities for increasing recovery of copper and gold have been identified.

In 2012, the operation will be upgraded and debottlenecked to process a nominal throughput of 16 Mt/a with installation of a second 13 MW ball mill, a 33 per cent increase in rougher flotation capacity, a 40 per cent increase in second cleaner capacity, and 33 per cent increase in third cleaner capacity.

In 2013, the operation will increase total recovery of both copper and gold by six per cent into final concentrate by increasing mass recovery into rougher concentrate, and debottlenecking of rougher concentrate regrind, cleaning, and final concentrate dewatering plants.

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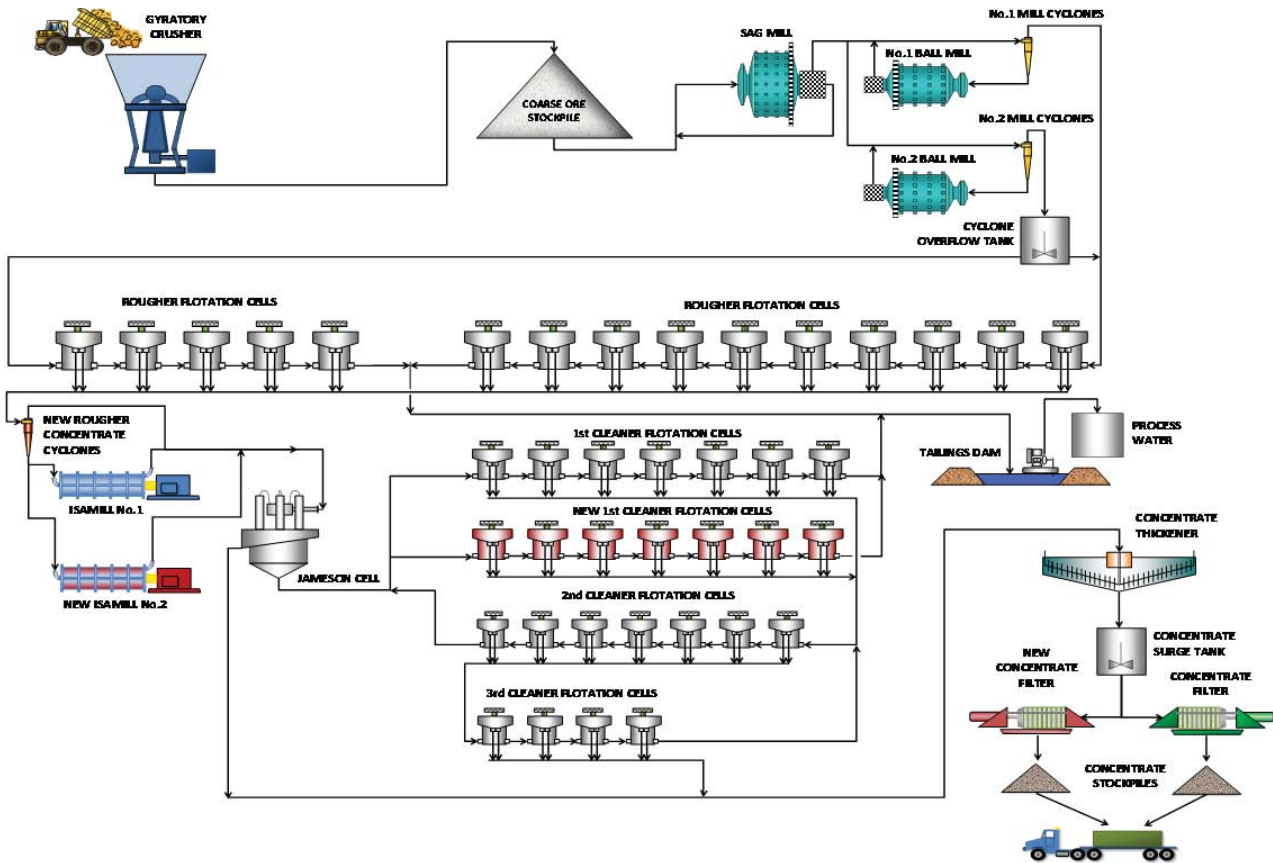


FIG 21 - Simplified Phu Kham process flow diagram including increased recovery project expansion.

work and technical assistance in the numerous projects to increase throughput and recovery at Phu Kham. The authors also gratefully acknowledge the support, technical reality checking, and encouragement of Mr Peter Munro and Dr Bill Johnson of Mineralurgy Pty Ltd, Dr Greg Harbort of AMEC Minproc for his flotation modelling work, and the work of John Glen and Tony Button at Burnie Research Laboratory, and Helen Johnston at G&T Metallurgical Services.

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