Commissioning and Optimisation of the Phu Kham Copper-Gold Concentrator

I Crnkovic1, T Georgiev2, G Harbort3 and M Phillips4

ABSTRACT

Phu Bia Mining Ltd, a subsidiary of Pan Australian Resources Ltd owns and operates the Phu Kham copper concentrator located in the Xaysomboun District, Lao People’s Democratic Republic. Construction and commissioning of the 12 Mt/a copper concentrator and associated facilities was undertaken by Ausenco Ltd and handed over to Phu Bia Mining Ltd in April, 2008. The first copper concentrate was produced in April 2008 after shipping the first shipment of copper concentrate set sail 27 July 2008. Nameplate mill throughput was reached in August, 2008 when 1 020 000 dmt of ore was treated in the milling and flotation circuit.

The Cu-Au mineralisation is contained in skarn and stockwork mineralisation and is interpreted as a distal expression of a conventional porphyry copper-gold system. Copper mineralisation is principally in the form of chalcopyrite with lesser bornite. Gold occurs as small grains, typically associated with copper minerals. In the upper portions of the orebody, oxidation and water table movements have resulted in near surface leaching of the copper and below the leached zone, supergene enrichment resulted in chalcocite mineralisation.

The circuit operates with a semi-autogenous grinding (SAG) – ball mill circuit, feeding a bank of OK200 rougher cells. Rougher concentrate is reground in an IsaMill prior to entering a cleaner circuit consisting of three stages of cleaning and one stage of cleaner scavenging.

The concentrate typically assaying 24 per cent Cu and 6 - 7 g/t Au is hauled for over 1000 km across the border to the Port of Sriracha in Thailand, and stocked in sheds at the wharf from which it is shipped to various Asian smelters.

This paper discusses the successful commissioning and ramp up to design tonnage and metallurgical performance. In addition, a program to optimise both the comminution and flotation circuit is detailed with reference to grinding efficiency, flotation residence time and size by size mineral recovery.

INTRODUCTION

PanAust Limited was founded in 1996. Between 1998 and 2003, during the low point in the metals’ price cycle the company embarked on an initiative to acquire advanced stage exploration assets and early stage development projects. This strategy led the company to evaluate assets in South-East Asia. In 2001, PanAust purchased a 19.9 per cent interest in PanAust, which on its part currently owns a 90 per cent interest in Phu Bia Mining.

The mine was designed to initially produce 12 Mt/a of ore through a conventional flotation process to produce approximately 260 000 dmt of concentrate annually, containing 65 000 t of copper, between 70 000 oz and 80 000 oz of gold, and between 400 000 oz and 600 000 oz of silver.

PanAust is committed to advancing its Southeast Asian projects and to acquiring further assets which will provide growth opportunities in the future. In line with this strategy the company is planning to increase the throughput of ore from 12 Mt/a to 16 Mt/a. Contained metal production from the expanded operation will increase to approximately 75 000 t/a copper, 65 000 oz/a gold and 600 000 oz/a silver. A conceptual study aimed at further lifting Phu Kham copper production to 100 000 t/a is currently underway.

CIRCUIT DESCRIPTION

The Phu Kham copper-gold was designed to nominally process 12 Mt/a of copper-gold bearing ore from the open pit, with a peak throughput capability of 14 Mt/a. At a design availability of 91.3 per cent (8000 hrs per annum) this equates to an hourly throughput of 1500 t/h.

Run-of-mine ore from the open pit is delivered to the primary gyratory crusher dump pocket by 90 t (777 D) capacity rear-dump trucks. The dump pocket is designed to allow truck dumping from one side and has the capacity to hold two truckloads. A second front end loader dump point is included to supplement feed to the crusher.

 Crushed ore is withdrawn from the crushed ore pocket by a variable speed apron feeder. Crushed ore is conveyed from the primary crusher by an overland conveyor and discharged to a conical, open stockpile, which has a live capacity of 16 hours, or 24 000 tonnes. The total stockpile size provided caters for a 48 hour total storage capacity (72 000 tonnes), being recoverable by the use of excavators.

Coarse ore is reclaimed from the stockpile by two apron feeders, each driven by a variable-speed drive. Each feeder has the capacity to provide 100 per cent of the full tonnage rate, but mostly operates at 50 per cent capacity for even draw down of the stockpile. The apron feeder’s discharge onto the mill feed conveyor, which transfers ore to the mill feed hopper. The SAG mill grinding media is also added to the SAG mill feed conveyor via a rotary feeder and hopper arrangement.

The grinding circuit consists of a single semi-autogenous grinding (SAG) mill followed by a single ball mill operating in closed circuit with a cyclone cluster. The design product from the grinding circuit (cyclone overflow) has a size distribution of 80 per cent passing 106 µm.
The 34 ft × 18 ft (13MW) SAG mill operates with pebble lifters and a pebble conveying circuit. SAG mill pebbles are returned to the SAG mill feed conveyor by a high-lift pebble conveyor. The current pebble conveying system will be replaced by a pebble crushing circuit (anticipated in year three) and the SAG mill will operate in closed circuit with pebble crushers. The high-lift conveyor will remain in place and will provide emergency bypass system should the pebble crushers be out of circuit. The SAG mill is fitted with a dual pinion drive (6.5 MW per pinion) and is driven by a variable-speed drive comprised of a SER/Hyper synchronous drive.

The 24 ft × 39 ft (13 MW) ball mill is driven by dual, 6.5 MW wound-rotor motors through reduction boxes and gear and pinion drives. The SAG mill discharge is screened on a rotating trommel screen. Trommel undersize discharges into a sump where it blends with ball mill discharge. The blended slurry is pumped by a single 1.85 MW variable-speed cyclone feed pump to a single cyclone cluster.

The overflow from the cyclone cluster gravitates via a multi-stage multi-fin metallurgical sampler followed by a moving crosscut sampler, prior to discharging into the rougher conditioning tank. The sample collected by the multi-stage slurry sampler is pumped to an On Stream Analyser (OSA) unit.

Slurry gravitates from the conditioner tank to the rougher-scavenger flotation circuit, which consists of four 200 m³ rougher and four 200 m³ scavenger tank type flotation cells. Concentrate from the rougher-scavenger circuit, collected in the concentrate launder gravitates through a static, dual fin launder sampler into the concentrate sump. Concentrate is pumped from the rougher concentrate sump to the regrind underflow to the cleaner number 1 feed hopper.

The regrind cyclones operate in open circuit, with the overflow reporting to the cleaner number 1 feed hopper. Underflow reports to the ISAMill feed hopper. Magotteaux ceramic grinding media are added to the regrind mill feed hopper prior to the slurry being pumped to the ISAMill. The M10000 ISAMill operates in open circuit and discharges the reground underflow to the cleaner number 1 feed hopper, where it recombines with the overflow from the classification circuit.

Concentrate from the regrind circuit, together with cleaner scavenger concentrate and cleaner two tails reports to the 70 m³ cleaner conditioner tank, from which the slurry gravitates to the cleaner-scavenger flotation section. This section consists of three OK70 m³ cleaner one cells and three 70 m³ cleaner scavenger tank type flotation cells.

Concentrate from the cleaner one flotation circuit is pumped, via an inline cleaner one concentrate sampler, to cleaner two. Tails from the cleaner one flotation circuit gravitates to the cleaner scavenger cells. Concentrate from the cleaner scavenger flotation circuit is usually pumped, via an inline cleaner scavenger concentrate sampler, to the cleaner one conditioning tank. This stream could also be diverted either to cleaner two or the regrind cyclone feed hopper, if deemed necessary by the operators. Tails from the cleaner scavenger flotation circuit gravitates via a static cleaner scavenger tails sampler, to the final tails collection hopper.

Cleaner two feed is comprised of cleaner one concentrate and cleaner three tails. Concentrate from the cleaner two flotation circuit is pumped to cleaner three. Tails from the cleaner two flotation circuit gravitates to the cleaner one conditioning tank. Cleaner three concentrate is pumped to the concentrate thickener. Tails from the cleaner three flotation circuit gravitates to the cleaner two feed box.
Low pressure air is supplied to each flotation cell within the Phu Kham flotation circuit. The airflow to each cell is controlled by vendor-supplied instrumentation. Similarly, cell levels are controlled by vendor supplied instrumentation. Dual internal dart valves on the outlet of each cell control pulp level in the cells. The action of the valves is governed by the output from a ball-float, target-plate and ultrasonic level detector combination mounted in each cell. In the event of a power failure, the valves fail closed so as to prevent the cells from draining.

The flotation concentrate, concentrate filter filtrate, thickener floor sump pumps and truck wash sump pumps all report to the concentrate area trash screen. The concentrate area trash screen is required to minimise the ingress of trash into the concentrate thickener. Oversize from the concentrate area trash screen discharges into the concentrate area trash bunker. The undersize from the concentrate area trash screen reports to the concentrate thickener feed box, prior to the concentrate gravitating to the thickener feed well.

A scum removal system is incorporated within the concentrate thickener. Froth scum is recovered from the surface of the concentrate thickener and discharges to the concentrate thickener sump pump. The concentrate thickener sump pump returns the froth scum to the concentrate area trash screen.

Thickener overflow is collected in the concentrate thickener overflow tank, from which it is pumped to the mill cyclone feed hopper, as process dilution and make-up water.

The concentrate thickener underflow is pumped to a mechanically agitated concentrate storage tank, which provides surge capacity between the concentrate thickening and filtration processes. Thickened concentrate is pumped from the concentrate storage tank to the concentrate filter. Filtered concentrate is discharged to the bulk concentrate storage shed and filtrate from the concentrate filter is collected in a filtrate hopper, prior to being returned to the concentrate thickener.

Both the rougher tails and cleaner scavenger tails are individually sampled, prior to being combined in the final tails collection hopper. Tailings from the final tails collection hopper gravitates via a static, multi-fin final tailings sampler prior to discharging into the final tailings discharge hopper. The fall from the plant to the tailings storage facility (TSF) is approximately 150 m. Tailings from the process plant gravitate to the TSF, from which process water is supplied to the plant via a decant water pumping system.

A basic flow diagram, representing the Phu Kham copper gold circuit is represented in Figure 2.

**CONSTRUCTION**

**Process area and infrastructure**

Key milestones for the construction of the Phu Kham project are shown in Figure 3, with a more detailed description of activities given below.

In January 2006, Pan Aust approved the commencement of initial pioneering works for Phu Kham. These activities included earthworks for the camp and process facility sites, together with the construction of a 9 km link road from the Phu Kham site to a recently constructed new access road – funded by the Asian

![Fig 2 - Phu Kham basic process flow.](image-url)
Development Bank – that connects to the main highway to Vietnam. This included construction of a 40 t capacity bridge over the Nam Mo River.

Negotiations with suppliers of long lead items were one of the key activities completed during the early construction phase. By the end of March 2006, tenders for the crusher and both SAG and ball mills had been issued. By June 2006, commitments were made for all critical path and long lead items, including the SAG and ball mills, flotation cells and crusher and concentrate thickener. Orders were placed for the 115 kV substation transformers in September 2006 and construction of the transmission lines and tap-off station was commenced.

Construction of the tailings storage facility started in October 2006. The 115 kV transmission tower assemblies and line route were approved in November 2006 and clearance given to commence construction. Concrete foundations for 25 of the total 28 towers were completed by the end of December 2006. Also by this time process plant earthworks were completed with the exception of the stockpile reclaim area. Concrete form work at the plant site commenced in early December and the first concrete footings were poured in the workshop and concentrate warehouse areas.

Earthworks also started on the mining offices and mobile equipment workshop site in early 2007 and earthworks for the crusher were completed in January 2007. The largest two concrete pours of the development, the SAG mill foundation at 690 m³ and ball mill foundation at 890 m³, were successfully undertaken between late March and early April 2007. By April, all mill and crusher castings were completed. Machining of these components was completed by the end of June 2007. In addition, erection of the first structural steel in the workshop areas commenced during this quarter. Construction of the permanent building had progressed to the stage where the first three blocks were available for occupation. Stage 1 of the tailing storage facility was completed and construction of the Phu Kham high-voltage electrical substation started in March 2007.

The last six months of 2007 represented the major construction phase of the project.

The SAG and ball mill shells arrived at site in Laos in early August 2007 which, with two of the mill motors, was one month ahead of schedule. The SAG mill shell, SAG and ball mill heads, trunnions, ring gears, pinion gears and remaining mill motors arrived progressively during August and early September 2007.

Transport of the major equipment components required a high degree of logistical planning. The mill shells were fabricated in South Africa and most of the ancillary components were fabricated in Australia. The two SAG mill cylindrical sections were each split into three 120° segments and the longer ball mill shells were each transported in six segments. This amounted to a total of 12 × 50 t trailer loads. The primary crusher was fabricated in China. The main castings for the crusher two top shell halves were split into 50 t loads for ease of transportation.

The SAG mill and ball mill segments were preassembled in September and the shells were lifted into position in early October. Assembly of the shell head and trunnion components of the SAG mill and ball mill was completed during this quarter and the ring gears installed, allowing the mills to be turned using their inching drives lining of the SAG mill and ball mill was completed in early January 2008.

The crusher wing wall civil works were completed by mid-November 2007. Installation of the crusher commenced in late November. The final concrete pour for the crusher vault was completed in January 2008 with crusher installation completed by the end of the month.

All flotation cells were lifted into position on support steelwork by November 2007. Installation of their operating mechanisms was completed by year’s end. The concentrate thickener tank was also successfully hydro-tested and the rubber lining installed. Construction of the high-voltage transmission line was completed in late October 2007, and energised the next month. Progressive energising and testing of site transformers, motor control centres and other electrical circuits subsequently commenced.

Processing plant electrical and pipe work installation was commenced near the end of 2007. Construction of the 1 km long overland conveyor from the primary crusher to the processing plant was completed during February 2008.

By mid-February construction was 95 per cent complete, with the SAG and ball mills, stockpile reclaim and plant feed conveyor and flotation cells mechanically complete. Water was pumped from the tailings storage facility via the decant water pumping system and the compressed air and flotation blowers were commissioned in readiness for wet commissioning of the plant.

Total construction cost was within the US$241 M capital budget and completion several months ahead of the original mid-2008 schedule.

Mine development

Mine development was started during 2005 and was accelerated due to integration of Phu Bia gold mining activity with development of the Phu Kham mine. This allowed prestripping of the underlying copper-gold deposit and integration of the open
pit utilisation for both mines, realising substantial cost benefits for the combined operations. Infrastructure such as the accommodation camp, road development, power and the logistical, government liaison and administration support of the established office in the capital Vientiane were also shared.

Pan Aust elected to lease and operate a Caterpillar mining fleet for the Phu Kham Copper-Gold Mine. The owner-operate alternative proved the more cost-effective option compared to contract mining tenders, with unit mining cost estimates consistent with feasibility study assumptions. In addition, owner mining allowed increased flexibility for undertaking project development works, including the construction of the tailings storage facility and the establishment of haul roads.

On 1 January 2007, the first complement of mining equipment was commissioned, comprising eight CAT 777D trucks and two O&K RH40 excavators. Other mining fleet equipment delivered to site and commissioned during the following months included two D10 CAT dozers, two CAT 16H graders, one loader and one service truck.

Waste mining for the Phu Kham copper-gold mine stage 1 pit followed the integration of mining activities with those for the Phu Bia Gold Mine. The first parcel of copper-gold ore was mined and stockpiled by mid-2007. The mining capacity was increased with the commissioning of two RH90 180 t face shovels, adding to the existing two RH40 120 t excavators. By the end of the September, a total of 11 CAT777 100 t trucks were in operation. The expanded fleet achieved a record monthly total material mined of 585 799 t during September 2007. The mining fleet was further expanded by year’s end with the commissioning of eight CAT777D 100 t trucks, bringing the total fleet to 18. The entire mining fleet operates under the automatic control and real time GPS tracking of the Jigsaw production and maintenance management system supplied by Leica Systems.

**COMMISSIONING**

Commissioning with copper-gold ore commenced in late March 2008, with the crusher grinding circuit and rougher flotation circuits commissioned first, followed by the cleaner flotation and concentrate circuits, and finally the balance of the process circuit.

Ore feed commenced using soft transitional (partially oxidised) ore, which did not need crushing and comprised the dominant type of ore to be mined in the first year. Over 750 000 t of transitional ore was stockpiled ready for processing. During the first week of April, the crusher was commissioned and all process plant commissioning trials were completed.

By the end of April 2008, the first copper concentrate was being produced and the operation moved from the commissioning phase into the production ramp up phase.

The successful commissioning with no major incidents is in no small part due to the site’s active training policy. During the 2007 wet season, most of the gold mine operators were retrained into copper-gold mine development and training activities. The company was able to hire and train a workforce and develop operating systems that were adopted by the copper-gold operations.

**RAMP UP**

Phu Kham adopted a conservative ramp up schedule. The ramp up to steady state production at design levels was scheduled to take 12 months with the initial six month focus on meeting design ore throughput levels (12 Mt/a) followed by a focus on optimising metallurgical recoveries for the various ore types.

Throughput tonnage and mill utilisation for the 12 months after commissioning are shown in Figure 4. Mill utilisation during the period May to July 2008, was adversely affected by unexpectedly high clay content of the ore and electrical supply disruptions during the Lao wet season. Although small amounts of clayey material were known to be present in the transition ore, the large amounts of clay mined resulted in crusher blockages, which were time-consuming to remove. Power supply from the national grid was affected by a number of lightning strikes. The most severe occurred in October 2008 when transformer damage occurred at the national power station that supplied electricity to the area. As the major power user in the area Phu Kham operations were mindful of the balance between production and support of the local population and worked together with the national power authority to schedule mill start-ups during non-peak usage periods and to coordinate periods of load shedding.

During the last two weeks of October, the Phu Kham copper-gold operation attained design production levels of copper concentrate equivalent to 260 000 dmt on an annualised basis (ie at forecast levels for 2009).

The safety performance of the Phu Kham Operations from initial commercial production to the end of 2008 was outstanding with a record 188 days free of lost time injury achieved.

**OPTIMISATION**

In August 2008, PanAust contracted GRD Minproc to assist in optimisation of the circuit. The optimisation program was structured to provide multiple benefits including:

- information on operational performance and improvement;
- simulations and determination of requirements for subsequent expansions;
- detailed training of local staff in world’s best practice sampling, mass balancing and flotation modelling/simulation techniques; and
- development of a set of floatability components that could be used for scale up with orebodies of similar mineralogy in the south east Asia region.

Sampling and bubble size measuring equipment, together with a bottom driven, batch flotation cell were hired from JKTech and dispatched to site in September 2008. The initial evaluation commenced in October 2008 under the direction of GRD Minproc, utilising Phu Kham staff. The deliverables from this campaign were:

- Bubble size measurement in all flotation cells. In addition to measurements being taken during normal operating conditions additional analysis was conducted during periods of plant unstable performance.
- Superficial gas velocity measurements to evaluate the air dispersion characteristics of the flotation equipment.
- Multiple cell-by-cell and block surveys to allow mass balancing on a size by mineralogy basis. The site assay laboratory conducted total copper, cyanide soluble copper (CNSC), weak acid soluble (WAS) copper and arsenic assays. This allowed the calculation of chalcopyrite,
chalcocite and tennantite mineral content in the samples. Samples were also assayed for iron and sulfur allowing pyrite and non-sulfide gangue content to be calculated. Screen sizing from 150 µm to 38 µm were conducted in the site’s metallurgical laboratory. Following the downturn in the mineral industry in late 2008, the off-site cyclosising of the minus 38 µm sample fractions was deferred until May 2009.

- Calculation of flotation cell size by mineral froth recoveries. It was originally planned to use JKTech measurement and calculation techniques (Alexander, Franzidis and Manlapig, 2003) to determine froth recoveries. After some difficulties with the hired equipment and questions on the technique accuracy, froth recoveries were calculated using the Metso Minerals Runge method (Runge et al, 2009).

- Evaluation of the IsaMill regrind operation. Surveys were conducted around the regrind cyclones and IsaMill. These samples were used to determine the water and size splits around the equipment and for floatability tests conducted on site. In addition, samples of IsaMill feed were dispatched to AMMTEC to conduct grinding signature plots.

- Batch flotation tests to allow size by mineralogy scale up from reagent trials.

**Aeration**

It is recognised that the bubble surface area flux (Sb) is directly related to flotation kinetics. The calculation of site Sb values across the various blocks of the flotation circuit are summarised in Table 1. In the rougher circuit in particular these values were significantly lower than values contained in the GRD Minproc database. Discussions with Outotec indicated that they were also lower than the design values.

**Bubble size**

Bubble size measurements indicated that a major reason for this discrepancy was large bubble size. Figure 5 presents the average Sauter mean bubble size for all flotation cells within the circuit. There is a definite trend of decreasing bubble size down the rougher line. The bubble size of the first rougher at 3.11 mm indicated that the conditioning tank was providing insufficient conditioning, either due to the modified flotation cell design or incorrect agitator speed. Checks were conducted on the speed of flotation impellers and indicated that they were within design specifications.

![Bubble size measurement in the flotation circuit.](image)

Operations were also concerned that the presence of clay within the ore was adversely affecting the effectiveness of frother. To evaluate this, measurements were taken with similar frother addition at times of either clayey or non-clayey ore treatment. As

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**TABLE 1**

*Flotation circuit aeration details.*

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**6 Adelaide, SA, 12 - 14 October 2009 Tenth Mill Operators’ Conference**
The Phu Kham operations had commenced with methyl iso-butyl carbinol (MIBC) as frother. Consumption levels were higher than specified and it was questioned if this frother was adequate for the ore treated. Trials of various frothers were conducted and the MIBC was replaced in February 2009 with Cytec’s OrePrep® PBM-604, a custom frother for the operation.

Air flow

Air flow rates to the flotation cells were also below design levels. Following discussions with Outotec an audit was conducted in April 2009 (Coleman, 2009). It was found that the setpoint for blower air pressure was set too low. Setpoints were increased prior to 62 kPa for the OK200 roughers, 47 kPa for the OK70 cleaners and cleaner scavengers and 31 kPa for OK20 cleaners. A site maintenance inspection also found the blower mechanical air relief was set below control setpoint and once rectified blower site maintenance inspection also found the blower mechanical air cleaners and cleaner scavengers and 31 kPa for OK20 cleaners. A prior to 62 kPa for the OK200 roughers, 47 kPa for the OK70 cleaners and cleaner scavengers and 31 kPa for OK20 cleaners. A setpoint with the valve operating at 25 - 30 per cent open. Outotec also supplied new 100 mm valves with lower CV values to provide better control.

In cooperation with GRD Minproc and Outotec another site flotation cell audit, including bubble size measurements is scheduled for July 2009.

**Residence time**

As part of the GRD Minproc optimisation visit, a series of surveys were conducted. Residence times were calculated from these surveys and compared with design residence time. The comparison is shown in Table 2.

**Dilution**

At the time of surveying the rougher residence time correlated well with design. The cleaner circuit was operating with residence times well below the design levels. There were a number of reasons for this issue. The excess of frother to the roughers required a high addition rate of launder and pump box water to prevent spillage of froth. Water addition was through high volume, low pressure open pipes. Plans are currently underway to replace these with high pressure/low pressure sprays. The IsaMill during this period was only in intermittent operation and its start up and shutdown resulted in surges through the cleaner circuit. In addition, the return of cleaner scavenger concentrate to the second cleaner reduced the residence time in this section of the cleaners.

**Level control**

Erratic level control and occasional sticking of dart valves also affected flow to the cleaner circuit. Calibration of the froth depth was tested and found to be incorrect. All cells were found to have positive offset ie if cell froth depth was set to 0 mm the cell would be pulping excessively. The ultrasonic level transmitter position for OK200s and OK70s was adjusted down to set 0 mm froth depth. The physical range was reduced from 800 mm to 700 mm. Guide tubes for the float levels were also adjusted to prevent sticking of the floats and the control to dart valves was adjusted to allow more frequent movement of lower amplitude.

**Survey results**

**Flotation performance**

At the time of the survey a rougher copper recovery of 76 per cent was achieved, with a cleaner circuit recovery of 89 per cent. This equated to an overall recovery of 64 per cent at a concentrate grade of 29.6 per cent Cu. Monthly production figures (Figure 8) show this was typical operation at the time.
Cell by cell recoveries

Figure 9 details the rougher cell by cell recoveries of copper bearing minerals (chalcopyrite, chalcocite and tennantite). The anticipated recovery profile was a total copper recovery of approximately 60 per cent in rougher cell 1, progressively decreasing down the bank with lesser degrees of floatability due to particle size and liberation. Although the general trend was observed, there were some significant deviations. The first rougher recovery was substantially lower than expected. This appears due to the low Sb, as discussed above. Recoveries at cells 3, 5 and 7 represented recovery maxima, with lower recoveries preceding them. These represented reagent addition points and indicated that a more frequent addition of reagent would benefit recovery. Chalcopyrite was the most rapidly recovered copper species, while the recovery of chalcocite was seen to increase comparatively after rougher cell 4.

Entrainment

Entrainment measurements were also made down the rougher bank (Figure 10). This showed the expected trend of increasing entrainment with finer particle size. The entrainment/water recovery ratio (ENT) was highest in the first rougher, progressively decreasing down the rougher line. This is inversely related to the froth residence times (air basis) down the bank which progressively increased from rougher cell 1 to rougher cell 8 (Figure 11). The strategy of an increasing froth residence time profile down the bank was to prevent higher volumes of low grade material from roughers 7 and 8 entering the cleaner circuit and further reducing residence time. Another operating strategy under discussion is reversing this profile to allow a higher grade rougher 1 concentrate to bypass the first cleaner bank direct to the second cleaner bank, easing copper loads in the cleaner circuit.

Copper losses to tailing

A breakdown of copper losses in the final tailing is shown in Figure 12. The major area of losses occurred in the minus 38 µm size fraction in the cleaner tailing. These losses were relatively

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**Figure 9 - Down the bank rougher recovery.**

**Figure 8 - Copper recovery and concentrate grade.**

**Figure 10 - Entrainment in the rougher circuit.**

**Figure 11 - Froth residence times in the rougher bank.**

**Figure 12 - Copper mineral distribution in tailing streams.**
evenly split between tennantite, chalcopyrite and chalcocite. Subsequent mineralogical examination by G & T Metallurgical Services (Shouldice and Folinsee, 2009) showed these losses were either liberated or as binaries with pyrite. Losses were also elevated in the plus 106 µm fraction in the rougher tailing. The rougher mineral recovery by size (Figure 13) shows a significant decrease in recovery of all copper minerals above 75 µm.

**Regrind performance**

The regrind survey indicated a problematic water split across the regrind cyclones. At times the cyclone underflow only consisted of 30 per cent solids, compared to an overflow of 20 per cent solids. This compares to a design cyclone underflow of 65 per cent solids. It is recognised that both IsaMill grinding efficiency and wear rates are adversely affected by lower percent solids in the feed. Initial action by Phu Kham operations was to increase the mass recovery from the roughers, allowing a reduction in make up water to the cyclone feed. Further optimisation has focused on cyclone operation with changes to the vortex finder make up water to the cyclone feed. Further optimisation has focused on cyclone operation with changes to the vortex finder size allowing underflow solids to increase to 55 - 60 per cent solids at present.

Samples of IsaMill feed were laboratory ground using the standard Xstrata Technologies procedure at 35 per cent solids to determine whether production grinding efficiency matched the laboratory signature plot. The results indicated that this was the case.

A comparison of floatability tests around the regrinding circuit is shown in Figure 14. Notable is the kinetic drop in the IsaMill discharge. Both the IsaMill discharge and the cyclone over flow have a similar size distribution. This indicates the slower flotation rates caused by over grinding was not an issue. The Phu Kham IsaMill operates with ceramic media to reduce chemical interactions in the mill and chemistry changes were considered unlikely to have a substantial impact on flotation kinetics. It is believed that the increased surface area and insufficient collector coverage were the cause of poor IsaMill discharge floatability.

At the time of the survey campaign in October 2008, the majority of the cleaner circuit collector was added to the cleaner conditioning tank. Occasionally, collector was also added to the head of the cleaner scavenger section at very low rates. The cleaner scavenger tailing responded favourably to additional collector addition, resulting in markedly improved grade/recovery profiles, especially when the grade of this stream exceeded three per cent to 3.5 per cent Cu. Recently, the collector has been added to the first cleaner feed pump hopper. The collector addition points are part of an ongoing review.

The grade recovery curves from the floatability tests (Figure 15) indicate that the IsaMill is a critical component of the circuit. The option of not operating the IsaMill would place concentrate grade at risk, due to the presence of composite particles. The modal monthly composite analysis has shown a change in liberation characteristics over time. As more primary ore has been treated the pyrite has shown an increase in intergrowths and some rimming of copper bearing minerals, requiring a finer regrind for effective liberation.

**Cleaner circuit performance**

Numerous batch tests conducted on cleaner scavenger tailing suggest that the major losses of copper (>60 per cent of the total copper present in this stream) existed in sub 38 µm size fractions. These samples were taken when the on-stream analysis (OSA) readings were showing copper assaying higher than 1.5 per cent Cu. Circa 45 per cent - 50 per cent of the copper misplaced with the cleaner scavenger tailing appeared in the 20 µm particle size. The results from these batch tests indicate the cleaner scavenger tailing losses could by reduced and 35 per cent to 45 per cent of the copper losses could be recovered by increasing the cleaner circuit residence time.

Other options for reducing the cleaner circuit tailing involve redirection of the tailing stream. The potential exists to pump all cleaner scavenger tailing either to the third rougher cell or to a cyclone classifying this stream in such a way that coarse fractions could be returned to ISAMill feed, while fines could be diverted back to the rougher. This return would be on or off depending on the cleaner scavenger tailing grade.

Following improvements in the rougher circuit the regrind and cleaner circuit has become the major area of optimisation. Analysis of sized results from numerous surveys are currently being analysed to allow a strategy for optimisation to be further developed. A significant number of flotation simulations are either in progress, or planned with the GRD MinFloat simulation package developed for Phu Kham by GRD Minproc.
Reagent optimisation

The work on selecting a better reagent scheme suitable for the types of ores being treated is still underway. In this regard, some lab tests showed that oxidised and/or weathered copper minerals could be increased using AM2 hydroxamate flotation reagent as a scavenger or secondary collector alongside the incumbent Aero 9810. Testing of potassium amyl xanthate (PAX) and other collectors supplied by reagent manufacturers such as Cytec, Tall Bennett, FloMin and Orica are currently underway.

CONCLUSIONS

The Phu Kham Copper Gold Operation was successfully designed and built during the recent high commodity prices. The risk of long lead times and accelerating key component prices was mitigated by prudent forward planning. The project was completed within budget and ahead of schedule.

A grade-recovery optimisation program indicates that further improvement is available in the following areas:

- optimisation of bubble size and the air flow rate in the roughers, to increase rougher kinetics;
- optimisation of the regrind circuit, including the cyclone water split, IsaMill operation and collector addition; and
- an increase in the cleaner circuit residence time.

REFERENCES


