Design of a Large-Scale Concentrator for Treatment of a Copper Skarn Orebody

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ABSTRACT

In December 2004 Phu Bia Mining Ltd commissioned Ausenco International Limited to conduct a feasibility study for its Phu Kham copper-gold project located in Laos. The feasibility study for a mining and processing operation treating 12 Mtpa of ore averaging 0.56 per cent copper and 0.25 g/t gold to produce an anticipated annual average 200 000 t of concentrate containing 25 per cent copper was completed in early 2006.

The plant design incorporates a single processing line and includes a gyratory crusher, overland conveyor, coarse ore stockpile, SAG and ball mill grinding circuit, flotation circuit including regrind and three stages of cleaning, concentrate dewatering and concentrate load-out facilities.

The Phu Kham copper-gold mineralisation comprises two principal styles: stockwork-disseminated and banded-massive sulfide within skarn limits. Primary copper in both mineralisation styles occurs as chalcopyrite and secondary copper sulfide minerals. Skarn includes the copper and gold mineralisation contained in pyrite, silicate, magnetite and haematite skarns, and in quartz-pyrite altered sediments.

Both the stockwork-disseminated and banded-massive sulfide ores contain relatively high proportions of pyrite relative to the copper and gold mineralisation. Thus, the flotation metallurgy of the Phu Kham orebody is characterised by the need to reject a high proportion of the pyrite during differential flotation to yield a concentrate grade of approximately 25 per cent copper. The rejection of pyrite is accompanied by the rejection of some copper and gold and can result in modest metallurgical recovery. Consequently, the greater the pyrite content, the lower the expected copper and gold recoveries.

The relative pyrite content is expressed on the basis of sulfur:copper ratio. For Phu Kham ores, the sulfur:copper ratios are in the range eight to 14 on a blended annual basis, but can be greater than 20 in the skarn components. Higher grades of copper and gold are also present in the skarn mineralisation.

The design concepts for the concentrator were influenced by the need to provide a low capital cost approach for the project due to the relatively low grade of the ore and the modest metallurgical recovery.

The concepts used to minimise capital cost whilst retaining operating and maintenance flexibility are discussed, together with the metallurgical challenges imposed by the high pyrite mineralisation.

INTRODUCTION

Ownership and development

The Phu Kham copper-gold project is owned by Pan Australian Resources Limited through a Lao-registered company, Phu Bia Mining Limited (PBM). PBM has a Mineral Exploration and Production Agreement (MEPA) with the Government of Laos. This agreement permits the company to explore for, develop and mine precious and base metals, within specific time frames, within the 2595 km² Phu Bia contract area in Laos. The Phu Kham copper-gold project is located in the south east corner of the Phu Bia contract area in Laos (Figure 1) and will be operated by PBM.

The deposit was discovered by a Normandy-Anglo American joint venture with exploration between 1994 and 1997. During this period the deposit was drill delineated to an approximate depth of 300 m below surface by 64 diamond drill holes.

In 2003, Phu Bia Mining Ltd (PBM) relogged the entire previous drill core and compiled a geological model for the deposit. This work defined the deposit as the distal segment of a copper-gold porphyry system. An initial mineral resource estimate was completed in 2002 using the 1994-97 drill hole database. PBM has completed an in-fill drill program in the central zone of the deposit. A revised mineral resource and mining reserve based on this latest drilling was completed in late 2005.

In December 2004, PBM commissioned Ausenco International Limited (Ausenco) to complete a feasibility study for a mining and processing operation treating between 9 Mtpa and 12.5 Mtpa of ore at a head grade of 0.6 per cent copper and 0.4 g/t gold to produce an anticipated 250 000 tpa of copper-gold concentrate containing 23 - 25 per cent copper and approximately 8 g/t gold. The concentrate was planned to be transported to markets predominantly in Thailand, China, India and Japan via road transport, ports and shipping.

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Ausenco managed the feasibility study with input from third party consultants and engineers in the design of mining, geotechnology, metallurgical test work, tailings storage facility, high voltage power supply, environmental assessment and concentrate marketing.

In 2006, PBM commissioned Ausenco to commence detailed design, procurement and construction of the project. The two 13 MW grinding mills were purchased in June 2006 with the plant scheduled for commissioning in the first quarter of 2008.

**PROCESS DESIGN**

The process design program consisted of metallurgical and mineralogical studies to define the ore processing characteristics followed by a concept definition phase where concepts that would optimise capital cost were developed to suit the location of the project.

The metallurgical performance was a function of the mineralogy. The Phu Kham copper-gold mineralisation comprises two principal styles: stockwork-disseminated and banded-massive skarn. Primary copper in both mineralisation styles occurs as predominantly chalcopyrite in the primary zones and chalcocite, digenite and other secondary sulfides in the transition ore zone. Skarn ores included the copper and gold mineralisation contained in pyrite, silicate, magnetite and haematite skarns, and in quartz-pyrite altered sediments. Both the stockwork-disseminated and banded-massive skarn ores contain relatively high proportions of pyrite relative to the copper and gold mineralisation.

Upper zone transition ores were relatively soft with altered copper mineralisation and complex associations with pyrite. This resulted in higher predicted throughputs, poorer flotation recoveries and relatively low concentrate copper grades considering the dominant chalcocite mineralogy.

The primary ore zones were comprised of more competent ore, resulting in lower grinding circuit throughputs and better copper and gold recoveries in flotation.

The flow sheet selection was typical of a large copper concentrator except for the inclusion of a third stage of cleaning and the use of sand mill based regrind.

The flow sheet is comprised of:
- ore delivery to a primary gyratory crushing;
- primary crushed ore stockpiling;
- apron feeder reclaim;
- a SABC grinding circuit;
- rougher and three stage cleaning flotation circuit;
- rougher concentrate regrind;
- concentrate thickening, filtration and load-out;
- direct flow of unthickened tailings to the tailing storage facility (TSF); and
- recovery of process water from the tailings dam.

The plant layout was constrained by the topography. Plant site selection involved an iterative process with the plant location moving several times due to ground condition, proximity to local communities, proximity to open pit mine and environmental control considerations.

The final location allows gravity flow of tailings and all plant run-off to the tailings dam. Thus, the environmental impact of the project is constrained to the mine and adjacent tailings dams areas.

**OREBODY DESCRIPTION**

Geology and mineralogy pertaining to metallurgy

In a simplistic sense the host rock is an altered porphyry overlaying redbeds. The orebody consists of two zones, thrust and block zones (Figure 2). Within these zones the ore is comprised of stockwork-disseminated sulfide and banded-massive sulfide mineralisation. Overlying the rock types and mineralisation events is secondary alteration that has resulted in:
- a copper deficient oxide layer containing gold that is currently being heap leached (Phu Bia project),
- a mixed ore zone of oxide and transition sulfide mineralisation and host rock (not included as ore),

![Schematic section of Phu Kham Copper-Gold Porphyry System](image-url)
a transition ore zone containing dominant chalcocite (denoted ‘c’ ore),
a lower transition zone with mixed chalcocite and chalcopyrite (denoted ‘t’ ore), and
a primary ore zone containing dominant chalcopyrite (denoted ‘p’ ore).

The supergene zone is relatively small and for the purposes of the study was incorporated in the transitional zone.

Pyrite is the dominant sulfide mineral in both the transition and primary ore zones resulting in typical sulfur:copper grade ratio within the economic ore zone of between ten and 12.

The term ‘skarn’ is used interchangeably to describe particular types of mineralogy within the banded/massive mineralisation category. The skarn ore exists as a mixture of mostly pyrite and silica based skarns with some magnetite skarn and quartz-pyrite altered sediments.

Disseminated stockwork mineralisation was described as stockwork-disseminated (abbreviated to ‘SWD’) and suffixed as either upper transition, lower transition or primary alteration (abbreviated to ‘c’, ‘t’ or ‘p’, respectively).

Banded/massive mineralisation was described by a differentiated ‘skarn’ type, eg sediment (abbreviated as SDM), pyrite (abbreviated as ‘PSK’), magnetite (abbreviated as ‘MSK’) and silicate (abbreviated as ‘SSK’). Likewise, each differentiated skarn type is suffixed according to alteration type.

A number of rock type based and production period based samples were assessed using either optical or automated mineralogical analysis methods. The mineralogical characteristics of these samples suggested that problems would be encountered in the design of a treatment process. The copper sulfide to pyrite mass ratios for these samples, which ranged from 1:5 to 1:14, were appreciably greater than those displayed for most other mineral deposits worldwide.

Approximately 15 per cent of the mineral suite was pyrite. Banded/massive mineralisation was described as porphyry skarn deposits worldwide.

Secondary enriched copper sulfide minerals, chalcocite and covellite, indicated altered mineralisation within the deposit. Since chalcocite and covellite occurred in close proximity to pyrite, often as rims, some activation and subsequent flotation of pyrite is likely. The principal objective of process design was to ensure that adequate flotation differential is maintained between copper sulfides and pyrite in the rougher flotation stage.

The low copper sulfide to pyrite mass ratio in the samples suggested that establishing and maintaining a sufficient differential flotation rate between pyrite and the copper sulfides to achieve high concentrate grades would be difficult at high copper sulfide recovery. This indicated the need for strong suppression of pyrite in the cleaner stages and some selectivity in the rougher flotation stage.

Based on mineralogical investigations, a flotation feed sizing of about P80 106 µm was expected to be sufficient to achieve a 23 per cent Cu concentrate grade and reasonable recovery.

Assuming a rougher flotation feed sizing of about P80 106 µm, the sizing of the rougher concentrate was expected to be approximately P80 75 µm. These rougher concentrates require regrinding. A regrind target of P80 of less than 38 µm is required ahead of cleaner flotation.

Head samples based on mine production and mineralogical classes were ground to a P80 of 106 µm and the MLA extended liberation analysis technique used to assess the liberation of five size fractions:

- >106 µm, 20 per cent of mass;
- <106 >38 µm, 36 per cent of mass;
- <38 >CS, 24 per cent of mass; and
- <CS, 19 per cent of mass.

The size fraction assays indicated a bias of copper deportment to the finer fractions and non-copper sulfides (pyrite) to the <106 >38 µm fraction.

Chalcopyrite was the dominant copper mineral with minor chalcocite, bornite and tennantite in primary ore. Transition ore was dominated by chalcocite, with minor bornite and tennantite. Approximately 15 per cent of the mineral suite was pyrite.

Chalcopyrite liberation in the <38 µm fractions was 83 per cent reducing to 20 per cent in the >106 µm fraction. Bornite, chalcocite and tennantite showed poorer liberation, indicating more difficult metallurgy in treating the transition ores.

Principal non-Cu binary associations of chalcopyrite were with non-sulfides (estimated at 70 to 80 per cent) and pyrite (20 to 30 per cent). Chalcocite showed a higher tendency to be associated with pyrite in composites, again presumably due to copper mineral alteration in the transition zone.

In summary, the MLA assessments indicated that:

- transition zone ore with higher chalcocite may have higher Cu mineral and pyrite association and, thus, result in more difficult flotation separation; and
- supported the optical mineralogy conclusion that a regrind to approximately 38 µm will provide sufficient liberation to achieve a 23 per cent Cu concentrate grade and reasonable recovery.

**Mine schedule**

The mineralogy (based on ore type code) of the feed to the plant as a function of ore schedule is summarised in Table 1.

### PROCESSING REQUIREMENTS

**Key test work outcomes**

The principal challenge with the Phu Kham ore was the high pyrite content and resulting high sulfur:copper grade ratio and consequential difficulties in separating the sulfide minerals. Pyrite must be rejected from concentrate to yield a target concentrate grade of >23 per cent copper. The rejection of pyrite

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**Table 1**

<table>
<thead>
<tr>
<th>Year</th>
<th>1</th>
<th>2</th>
<th>3</th>
<th>4</th>
<th>5</th>
<th>6</th>
<th>7</th>
<th>8</th>
<th>9</th>
<th>10</th>
<th>11</th>
<th>12</th>
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<tbody>
<tr>
<td>SWDc (%)</td>
<td>51</td>
<td>41</td>
<td>39</td>
<td>31</td>
<td>31</td>
<td>22</td>
<td>19</td>
<td>7</td>
<td>1</td>
<td>0</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>SWDt (%)</td>
<td>25</td>
<td>26</td>
<td>35</td>
<td>42</td>
<td>41</td>
<td>39</td>
<td>38</td>
<td>38</td>
<td>21</td>
<td>7</td>
<td>8</td>
<td>2</td>
</tr>
<tr>
<td>SWDp (%)</td>
<td>17</td>
<td>11</td>
<td>10</td>
<td>12</td>
<td>20</td>
<td>28</td>
<td>43</td>
<td>45</td>
<td>59</td>
<td>69</td>
<td>84</td>
<td>62</td>
</tr>
<tr>
<td>SKNc (%)</td>
<td>4</td>
<td>7</td>
<td>6</td>
<td>4</td>
<td>4</td>
<td>2</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>SKNt (%)</td>
<td>2</td>
<td>11</td>
<td>9</td>
<td>7</td>
<td>9</td>
<td>7</td>
<td>5</td>
<td>8</td>
<td>9</td>
<td>2</td>
<td>2</td>
<td>3</td>
</tr>
<tr>
<td>SKNp (%)</td>
<td>1</td>
<td>3</td>
<td>2</td>
<td>3</td>
<td>4</td>
<td>5</td>
<td>7</td>
<td>7</td>
<td>11</td>
<td>22</td>
<td>6</td>
<td>33</td>
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</table>
is typically associated with the rejection of some copper and gold and can result in modest metallurgy with some pyrite skarn ores. The feed sulfur:copper ratio for Phu Kham is in the range eight to 14 on a blended annual basis, but may be >20 in some locations. The greater the pyrite content, the lower the expected copper and gold recoveries.

The metallurgical test work program for the feasibility study was completed in five phases. Samples for the test work program were selected from within the Central Zone of the orebody by PMB geologists based on general criteria agreed by Ausenco and PMB.

Phase 1 included an initial assessment of comminution and flotation characteristics using samples selected in the basis of ore types and preliminary oxidation profiles.

Phase 2 of the test work program was principally comminution test work, but flotation test work was added during the study to allow a preliminary assessment of the impact of core oxidation.

- Phase 2A was comprised of comminution test work on fresh core from selected areas and ore types, thought to be the most competent. These samples were to set the lower throughput limits of comminution circuit capacity and were sourced from new holes that twinned existing holes to enable the use of whole core.

- Phase 2B was comprised of comminution test work on samples selected to represent period composites split into skarn and stockwork ore types. The samples were composited from the general core inventory by PMB geologists to represent the mining periods based on an interim mine schedule.

- Phase 2C test work was conducted to compare the flotation response of fresh core with that of the PMB core inventory (used for the remainder of the test work program). The core drilled for the Phase 2A samples was used to prepare fresh (unoxidised) core samples, and the core from the existing twin hole used to represent ‘oxidised’ core.

Phase 3 of the test work program was undertaken using composite samples representing mining periods defined by the interim mine schedule. The samples were composited based on the volume defined by the pit outlines and the available core by PMB geologists. The Phase 3 program was entirely centred on flotation metallurgy and the preparation of samples for engineering design test work.

Phase 4 included all test work conducted for engineering design on flotation circuit products (ie regind mill specific energy) and the feed material (eg materials handling test work).

During the feasibility study it became apparent that core oxidation was impacting on metallurgy. To evaluate the metallurgy of fresh core, a fifth phase of metallurgy was added to the test work program. The Phase 5 program focused on the metallurgy of the transition zone material that was most impacted by 

Comminution test work outcomes

The ore characteristics for comminution circuit design were defined as a function of ore type and mining period. In addition, samples selected to represent the highest competency and major ore types (at the middle and base of the proposed pit) were assessed. The results are summarised in Table 2, Table 3 and Table 4.

The Phase 1 samples (Table 2) were heavily weighted towards primary ore and did not reflect the less competent material in the mine schedule. Table 3 summarises the comminution data from the SSKp and SWDp ore types in the primary ore and indicates significant variation in the competency of SSKp ore, presumably associated with rock fracture and/or the level of silicification.

Table 4 summarises the data selected on the basis of the interim mine schedule. This reinforced the expected lower competency of the ores to be mined early in project life and resulted in decision to design the concentrator for a maximum throughput of 14 Mtpa to allow for higher throughput and cash flow during the early payback period.

### Table 2
Summary of Phase 1 comminution results.

<table>
<thead>
<tr>
<th>Sample</th>
<th>BWI (kWh/t)</th>
<th>RWI (kWh/t)</th>
<th>Ai</th>
<th>SMC test parameters</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td>(A \times b)</td>
</tr>
<tr>
<td>SWDnu/SWDth</td>
<td>12.8</td>
<td>12.0</td>
<td>0.19</td>
<td>105</td>
</tr>
<tr>
<td>SWDpm/SWDph</td>
<td>15.5</td>
<td>14.1</td>
<td>0.44</td>
<td>70</td>
</tr>
<tr>
<td>SDM</td>
<td>10.7</td>
<td>8.8</td>
<td>0.07</td>
<td>163</td>
</tr>
<tr>
<td>SDMp</td>
<td>14.6</td>
<td>16.3</td>
<td>0.29</td>
<td>53.1</td>
</tr>
<tr>
<td>PSK1</td>
<td>11.0</td>
<td>14.4</td>
<td>0.40</td>
<td>69.0</td>
</tr>
<tr>
<td>MSK1</td>
<td>11.8</td>
<td>13.9</td>
<td>0.29</td>
<td>72.8</td>
</tr>
<tr>
<td>SSK1</td>
<td>14.4</td>
<td>17.4</td>
<td>0.40</td>
<td>44.2</td>
</tr>
<tr>
<td>Mixed skarn</td>
<td>12.4</td>
<td>14.7</td>
<td>0.14</td>
<td>57.2</td>
</tr>
</tbody>
</table>

### Table 3
Summary of Phase 2A comminution test results.

<table>
<thead>
<tr>
<th>Sample</th>
<th>BWI (106 µm screen) (kWh/t)</th>
<th>UCS (Mpa)</th>
<th>RWI (kWh/t)</th>
<th>Ai</th>
<th>JK Drop Weight test parameters</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>(A \times b)</td>
</tr>
<tr>
<td>S1-SSKp</td>
<td>12.1</td>
<td>4.2</td>
<td>10.2</td>
<td>0.12</td>
<td>108.7</td>
</tr>
<tr>
<td>S2-SWDp</td>
<td>16.3</td>
<td>7.1</td>
<td>15.4</td>
<td>0.13</td>
<td>55.9</td>
</tr>
<tr>
<td>S3-SSKp</td>
<td>14.2</td>
<td>82.7</td>
<td>17.4</td>
<td>0.42</td>
<td>34.1</td>
</tr>
<tr>
<td>S4-SWDp</td>
<td>16.2</td>
<td>11.4</td>
<td>15.4</td>
<td>0.04</td>
<td>65.5</td>
</tr>
</tbody>
</table>
Figure 3 illustrates the trend between the Bond rod mill work index and the $A \times b$ parameters derived from the JK Drop Weight tests and the SMC tests.

**Optimum grind size**

The optimum grind size was determined for selected Phase 1 samples and all Phase 3 samples. The results were consistent and indicated an optimum laboratory grind $P_{80}$ of 106 µm. Some transitional ore will benefit from a grinding circuit $P_{80}$ as fine as 75 µm. The transitional ore is typically relatively soft and the grinding circuit has some capacity to grind finer to realise some of the potential advantage of a finer grind on the softer ores.

**Flotation data**

Flotation test work was, in the main, completed on core that had been stored in the Phu Kham site core shed for periods of between three and 18 months. Core sample oxidation for the transition zone ore resulted in lower copper recoveries. A revised reagent regime based on a dithionocarbamate (DTC) collector improved copper recovery when compared with the initial high pH/xanthate reagent regime. However, difficulties were experienced in achieving the same copper grades and recoveries to concentrate with old core when compared with ‘new’ core.

Further test work was performed using further modifications to the reagent regime (higher pH in cleaners and modified reagent addition) to improve pyrite depression. The higher cleaner pH showed improved cleaner pyrite depression for oxidised core. For primary ore samples, a 1 to 2 kg/t reduction in lime consumption between old and new core under the high lime/xanthate scheme was confirmed and the DTC collector reagent regime gave better recoveries at lower operating costs due to reduced collector and lime consumptions. The modified DTC-based flotation regime with pyrite depression in the cleaner stage at moderate pH, and a weaker frother appeared optimum.

Table 5 summarises the metallurgy from some of the locked cycle tests carried out on Phase 5 composite samples. The estimated equilibrium results were calculated based on the redistribution of the middlings streams. The tests with less than 98 per cent of metal reporting to products had a middling load that was increasing throughout the test. The higher the transition ore component, the higher the middling load indicating greater deviation from the equilibrium condition.

Typically, the metallurgy of the Phu Kham ore improved with decreasing sulfur to copper ratio and increasing primary ore content. Thus, primary ore containing low pyrite content yields high copper recoveries, whilst skarn ores with high pyrite content in the transition zone yield low copper (and gold) recovery.

Figure 4 illustrates the relationship between gold and sulfur recovery in open circuit.
Other test work

Metallurgical test work was also completed on the following:

- ore samples for materials handling and mass flow design, and
- rougher concentrate to determine regrind specific energy.

No test work was conducted on concentrate dewatering or tailings thickening. The concentrate dewatering test work was not conducted due to the absence of pilot plant derived concentrates and the size distribution dependence of dewatering circuit performance. Tailings thickening test work was not done due to the absence of a tailings thickener in the flow sheet.

Tailings deposition and environmental test work (acid mine drainage) was completed by the tailings dam and environmental engineers engaged by PBM.

PROCESS DEVELOPMENT

The discussion is based on the following process plant unit operations:

- primary crushing,
- overland conveyor and stockpile,
- grinding,
- flotation and regrind,
- concentrate dewatering and handling,
- reagents,
- tailings disposal, and
- utilities and services.

The overriding principal in the design of the plant was to provide a fit-for-purpose low-cost process plant suitable for the treatment of the Phu Kham ore. The in-pit value of the reserve demanded that the design of the plant be cost effective. The designs of some facilities reflect cost optimisation and, where appropriate, the limitations of the process facilities are noted in the following discussion.

A cost minimisation approach was only adopted where the design inputs were clearly defined by test work or supporting studies.

The flow sheet is presented in summary form in Figure 5. The process plant site is illustrated in Figure 6.

Primary crushing, conveying and stockpile

A large ROM stockpile was originally included in the plant layout. However, space restrictions resulted in the ROM stockpile being relocated to inside the pit and the primary crusher ROM pad has limited storage capacity.

The primary gyratory crusher size was optimised and selected to take double dumps from Caterpillar 785 trucks, or their equivalent. This allowed the elevation and cost of the crushing station to be optimised. A 54 × 75 gyratory crusher is able to handle 1200 mm top size at approximately 2000 t/h.

The overland conveyor delivers ore from the primary crusher to the crushed ore stockpile. The conveyor is a single run approximately 1.2 km long and 1200 mm wide. A 1350 kW drive system runs the conveyor at 3.5 m/s for a maximum rate of 2500 t/h. The drive system consists of three drives and accompanying gearboxes.

The conveyor is covered to limit water runback to the crushing station during the wet season.

The coarse ore stockpile was designed as a conical structure with two reclaim apron feeders. The cheaper option of the installation of a single belt feeder and two vibrating feeders was considered and rejected due to issues associated with belt feeder design and vibrating feeder operation on potentially 'sticky' ore.

Grinding

In summary, the following process was used for each comminution test work data set.

- The total grinding circuit comminution energy was determined using the Bond equation, a circuit efficiency factor and the Bond rod and ball mill work indices for all ore types. The circuit efficiency factor was ratioed to the DWI data. The Bond ball mill work indices for a 150 µm closing screen were calculated (where not measured) using a correlation between the BWI data:

\[ BWI(150\,\mu m) = BWI(106\,\mu m) \times 0.9165 \]

- The SAG mill specific energy was determined using an Ausenco proprietary correlation between SAG mill specific energy and the JK Drop Weight and SMC parameters.

- The ball mill specific energy was determined by difference between the total circuit specific energy and the SAG mill specific energy.

The resulting SABC circuit mill specific energy values are summarised in Table 6.

A 13 MW variable speed SAG mill and a 13 MW ball mill were selected for the project. Both mills have a twin pinion drive system.

Figure 7 illustrates the mill throughput limitations based on the selected mill sizes by year.

The grinding circuit was designed to be sufficiently flexible to cope with the majority of the ore types identified in the comminution test work program.

Cyclone underflow can be split between the SAG and ball mills. This allows any deficiency in ball mill power, and the resultant coarse grind, to be corrected through the use of excess SAG mill capacity (see Century, Ridgeway and Telfer circuits for example).

Pebble crushing is not required initially and installation of the pebble crushers may be deferred until Year 3 of operation.

## Table 5

<table>
<thead>
<tr>
<th>Phase</th>
<th>Sample</th>
<th>Calculated head grade</th>
<th>Per cent metal reporting to products</th>
<th>Copper concentrate predicted metallurgy</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>Cu (%)</td>
<td>Au (g/t)</td>
<td>S (%)</td>
</tr>
<tr>
<td>5</td>
<td>Chalccite ore</td>
<td>0.64</td>
<td>0.38</td>
<td>8.48</td>
</tr>
<tr>
<td>5</td>
<td>Upper transition ore</td>
<td>0.37</td>
<td>0.26</td>
<td>2.91</td>
</tr>
<tr>
<td>5</td>
<td>Lower transition ore</td>
<td>0.73</td>
<td>0.31</td>
<td>7.46</td>
</tr>
<tr>
<td>5</td>
<td>Primary ore</td>
<td>0.92</td>
<td>0.41</td>
<td>8.03</td>
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</table>
DESIGN OF A LARGE-SCALE CONCENTRATOR FOR TREATMENT OF A COPPER SKARN OREBODY

Fig 5 - Simplified process flow sheet.
Flotation and regrind
The flotation circuit was originally conceived as a bulk sulfide flotation circuit over 20 per cent mass recovery to rougher concentrate. This was modified to moderate (12 per cent) mass recovery to rougher-scavenger concentrate with subsequent rejection of non-copper sulfides and non-sulfide gangue in the cleaning circuit. This selection was made:

- over copper selective roughing due to the potential for copper and gold associations with pyrite that require the rougher concentrate to be reground to provide sufficient liberation and optimum recovery at target copper concentrate grade, and
- over bulk sulfide flotation (>20 per cent mass recovery) due to difficulties in rejecting pyrite recovered to rougher concentrate in the cleaning circuit in laboratory test work.

The compromise approach adopted used a copper and gold mineral selective collector in roughing at moderate pH with no additional pyrite depression and the collector dosage trimmed to yield rougher pyrite recoveries that allowed acceptable final concentrate quality to be achieved in cleaning using sodium cyanide and lime as pyrite depressants.

Flash flotation was considered for Phu Kham. Flash flotation is used in many copper/gold flotation circuits to minimise the ‘overgrinding’ of free gold in the grinding/classification circuit. The occurrence of ‘free’ gold in Phu Kham ores appears to be limited as flotation gold assay balances in test work were very consistent. Flash flotation was not included in the flow sheet on that basis. A similar approach was adopted in the design of Ernest Henry and flash flotation has not been retrofitted to that operation.

The OK200TC rougher flotation cells were selected, with OK70TC cells in the first cleaner and cleaner scavenger circuit and OK20TC cells in the second and third cleaners.

A non-ferrous media regrind technology was selected for Phu Kham. A single 2600 kW M10000 ISAMill® was selected on the basis of simplicity and additional capacity, over the 1700 kW required, allowing increased regrind flexibility and potentially higher copper and gold recoveries with transition ore types. Non-ferrous regrind technology was selected due to the importance of rejecting pyrite in the cleaner circuit and the potential activation of pyrite when iron based media is used.

The three-stage cleaning circuit was adopted to maximise copper concentrate grade. The circuit is a conventional open circuit cleaner scavenger flow sheet (Figure 5).

Concentrate dewatering and handling
The concentrate thickener and filter were sized based on the 12 Mtpa feed rate using an assumed unit area thickening rate based on experience with similar concentrates and a 0.7 per cent Cu head grade. Design concentrate production rates of 39 t/h for the thickener, and 41 t/h for the concentrate filter were used. This compares with maximum annual averages of approximately 31 t/h based on 91.3 per cent utilisation for the thickener and 33 t/h based on 85 per cent utilisation for the filter.

<table>
<thead>
<tr>
<th>Year</th>
<th>Specific energy (kWh/t)</th>
<th>Ore type</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>SAG mill</td>
<td>Ball mill</td>
</tr>
<tr>
<td>1</td>
<td>4.37</td>
<td>6.22</td>
</tr>
<tr>
<td>2</td>
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<tr>
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</tr>
<tr>
<td>11</td>
<td>6.95</td>
<td>9.31</td>
</tr>
</tbody>
</table>
An IPM pressure filter was selected and discharges filter cake to a stockpile below the filter. The filter was sized by the vendor based on experience and test work on a concentrate from a comparable operation.

The covered storage facility for filter cake has been minimised to reduce the capital cost associated with the concentrate storage structure. Space has been made available for outside concentrate storage should availability of transport be limited from time to time. The outside storage facility will require stockpiles of concentrate to be sheeted temporarily to protect the stockpile from rain and reduce dust emissions.

Reagents
The reagent’s make up and storage facilities included lime slaking based on powdered quick lime supply, a sodium cyanide mixing and storage system and DTC and frother, methyl isobutyl carbinol (MIBC), storage tanks.

Tailings dewatering and disposal
Both, rougher scavenger and cleaner scavenger, tailings report to the tailings collection hopper and directly to the tailings dam without prior thickening.

An improvement in water quality is expected by the use of recovered process water from the tailings dam. This is considered important in maintaining high selectivity against pyrite in the flotation circuit.

Sampling
The sampling system is comprised of multi-fin, multi stage static samplers for the primary metal accounting streams. Secondary streams, the data from which is required to define the flotation circuit mass balance, are sampled using static dual fin launder samplers. Tertiary streams, the data from which is required for process monitoring, will be sampled using pressure pipe samplers.

All the samples are pumped to a Courier on-stream-analyser for continuous analysis and monitoring of the plant performance. The Courier return samples are gravitated back to the process point of origin.

The design and layout currently makes space allowance for the future installation of a particle size analyser on the classification cyclone overflow.

DESIGN AND LAYOUT

General
The plant site is located on a ridge (Figure 6) and slopes from the coarse ore stockpile to the tailings discharge. The sloping site presents some advantages due to the use of gravity flow and reduced pumping requirements. However, the earthworks are relatively complex to set the various relative levels and optimum slopes required by the layout.

Primary crushing
The primary crusher is located adjacent to the pit exit to minimise mine haulage costs. The crusher has two dump stations and a drive in sump to allow clean out of the dump hopper. The crusher is an open face design (eg Cadia) rather than the in-ground variety (eg Telfer, Ernest Henry).

The crusher discharges directly to a heavy duty sacrificial conveyor that delivers ore to the overland conveyor. An apron feeder was not included to reduce capital cost.

Ore conveying
The overland conveyor is designed to process a peak flow of 2750 t/h and conveys crushed ore from the primary crusher to the crushed ore stockpile. The overland conveyor is fitted with a belt condition monitoring system and weightometer for mass flow measurement of ore transferred to the stockpile.

Stockpile and reclaim
The crushed ore stockpile is located just over a ridge. This allows the stacking conveyor to project horizontally to the stacking position.

The reclaim system consists of two apron feeders each capable of delivering the full feed rate. The apron feeders are located in close proximity to maximise the live capacity of the stockpile. Whilst the reclaim chamber is constructed of concrete, ‘Armco’ style conveyor and emergency escape tunnels are provided. Dust control is via a large mobile exhaust fan adjacent the emergency exit tunnel.

Grinding
The layout of two twin pinion drive mills with outboard discharge end motors is relatively complex if footprint and layout are to be optimised. Access to the grinding floor by mobile equipment is provided by a ramp and cutting the mill feed end into the sloping terrain. This reduces the storage area required for liner/filters.

The cyclone feed pumps’ location evaluation study took some time to complete. Numerous options were considered with all options complicated by the location of the motor plinths or the motors and the need to split the cyclone underflow between the SAG and ball mills. The final solution was to put the line overhead between the adjacent SAG and ball mill drives. Access to the cyclone feed pumps, motors and lines and mill motors and gear boxes is by mobile crane.

Pebble crushing
The pebble crushing circuit is yet to be detailed. The conveyors may be installed for mill start-up to allow the trommel oversize to be recycled and the current pebble jet return system to be removed.

Flotation plant
The limited available area necessitated a very compact flotation footprint. As the concentrator is being constructed on top of a hill, the naturally steep topography was utilised to minimise the elevation of the flotation circuit thus saving capital.

The individual flotation cells are elevated on low steel structures that enable gravity flow of tailings away from the plant area without the use of tailings discharge pumps.

The compact nature of the flotation circuit resulted in restricted mobile maintenance access to the flotation circuit. As a result a tower crane has been allowed for to provide maintenance crane access. The selected tower crane has a 55 m boom length with a maximum lifting capacity of 3.6 tonne when the boom is fully extended.

Regrid
The selection of the regrid circuit was an iterative process with a number of various regrid technologies being considered. The feasibility test work indicated that a specific energy requirement of between 8 kWh/t and 10 kWh/t was sufficient to produce a product size with a P_{80} of 38 μm.
Originally it was envisaged that five Metso 350 kW SMDs would be utilised in the Phu Kham regrind circuit. The limited available footprint and the potential for better mineral liberation, grade and recovery associated with finer grinding culminated in the selection of a single 2600 kW M10000 ISAMill for the regrind circuit.

The proposed regrind circuit will operate in open circuit with rougher concentrate being submitted to the regrind stage, prior to cleaner flotation. Cleaner scavenger concentrate can also be pumped to the regrind circuit to assist with middlings liberation.

The regrind circuit is located adjacent to the flotation circuit and maintenance access is provided by the tower crane.

Concentrate handling
The concentrate handling area is located adjacent to the flotation circuit and is approximately four metres lower than the flotation circuit. This area is comprised of a high rate thickener and vertical plate pressure filter. Concentrate is stockpiled in a storage shed and loaded to concentrate trucks by front-end loader.

The concentrate thickener is fitted with a scum recovery system. Froth scum is returned to the thickener feed. Thickener overflow is collected in a separate thickener overflow tank and pumped to the SAG mill discharge for use as mill dilution water.

Maintenance access to the concentrate handling area is provided by the site tower crane.

Tailings disposal and water management
The lack of real estate, the minimum capital approach to the project, and the opportunity to improve the quality of the process water resulted in the exclusion of a tailings thickener. In addition, the positive water balance and monsoon seasons dictated that harvesting and managing process water could be conducted at the tailings storage facility.

The plant has two tailings streams, rougher tails and cleaner scavenger tails. Both the rougher tails and cleaner scavenger tails are individually sampled, prior to being combined in the final tails collection hopper.

The fall from the plant to the tailings storage facility (TSF) is approximately 150 m. The process plant tail streams gravitate to the TSF via a series of break tanks in order to control the slurry velocity and vent line pressure to atmosphere.

The tailings reclaim water is harvested using pontoon pumps, which transfer the reclaim water to a decant water transfer tank.

As with the tailings disposal system, for the decant water return system there is approximately 150 m of head from the surface of the TSF to the plant. In order to minimise the pumping head required, the decant water is returned to the process plant via a series of water transfer stations.

APPENDIX 1

Fig A1.1 - Phu Kham process plant model as at October 2006.