Kibali Gold Mine sulphide concentrate treatment – understanding the preoxidation of sulphide concentrates

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ABSTRACT
Kibali Gold Mine (KGM) processing plant treats both free milling and partially refractory ores. The flotation concentrate, is subjected to ultrafine grinding in 8 parallel FLS VXP 2500 ultrafine grinding (UFG) mills and undergoes a two stage preoxidation process at controlled pH of 10.2, prior to cyanidation. The primary objective of preoxidation is to reduce the reactivity of sulphide surfaces and thus reduce cyanide and oxygen demand downstream. Complex interactions amongst operational parameters of feed density, pH and dissolved oxygen concentration, complicates the circuit.

This work demonstrates the impact of these interactions on gold dissolution and cyanide consumption. Aggressive preoxidation conditions have shown that the sulphide oxidation reactions not only increase temperatures but also generate cyanicides consistent with observed high cyanide consumption and poor dissolved oxygen concentration.

From the extensive analysis of plant data and laboratory testwork, the plant has established and defined a niche operating domain for optimal preoxidation processes. This has been shown by the resultant drop in the sulphide concentrate residues from above 5 g/t at commissioning to below 3 g/t, currently achieved.

Introduction
Kibali Gold Mine processing plant concentrate handling circuit treats a suite of semi-refractory sulphide concentrates. The concentrate contains pyrite as the abundant carrier of the enclosed gold grains, with pyrrhotite and arsenopyrite in lesser proportion. The main problematic components of the ores are the pyrrhotite and arsenopyrite bearing ores. These ores are respectively reactive and refractory to the current circuit configuration. Processed through a two stage oxidative pre-treatment circuit prior to high cyanide concentration leaching, the main objective in this route is to reduce sulphide minerals reactivity, partially liberate gold and thus mitigate high cyanide and oxygen consumption downstream.

The concept of preoxidation and/or pre-aeration in gold processing primarily carries different objectives dependent on the nature of gold occurrence within the sulphide host minerals. For fully occluded gold, the pre-treatment process occurs in aggressive oxidising conditions aided by (i) high temperatures and oxygen pressure (Kenneth et al., 1996), (ii) strong oxidising agents (Papangelakis and Demopoulous 1991; Kadioglu et al. 1995; Li 2009) and (iii) bacteria assisted systems (Ubialdini et al., 1997; Karamanev et al. 2001; Miller and Brown 2005). Gold carrying sulphides are completely oxidised with iron precipitated as stable oxy-hydroxide compounds that do not participate in the subsequent gold cyanidation step. Sulphide sulphur on the other hand, exits the process as sulphate precipitate. While the processes generally enable effective low cyanide consumption down-stream, they carry with them a huge price of energy and lime consumption because of the high temperature and low pH conditions of operation.

As outlined in the first paragraph of this paper, Kibali ores are mainly partially refractory at normal grinds of 80% passing 75 microns. While the sulphides are fully liberated at coarse grinds of 106 microns, gold liberation is achieved through ultrafine grinding of the flotation concentrate at 80% passing 18 microns. While further grinding has a beneficial effect of liberating gold, it has a concomitant effect of increasing sulphide reactivity due to high surface area. This negative impact is carried over into the downstream processes of preoxidation and leaching where excessive consumption of reagents has been observed. Conceptually (reactions 1–4), the reaction products of the peroxidation should be non-reactive (passivate) under the cyanidation leach conditions and invariably allow for the availability of cyanide and oxygen for gold dissolution.

\[
\text{MS} + 2\text{O}_2 + 2\text{H}_2\text{O} \rightarrow \text{M(OH)}_2 + 2\text{H}^+ + \text{SO}_4^{2-} \quad (1)
\]
The oxidation of sulphide sulphur to sulphate follows the mechanism of oxidation of thiosulfate to sulfate, as shown in equations (8-10):

\[
2\text{FeS} + \text{O}_2 + 2\text{H}_2\text{O} \rightarrow \text{Fe}_2\text{O}_3^- + 2\text{OH}^- \quad (5)
\]

\[
4\text{Fe}^{2+} + \text{O}_2 + 2\text{H}_2\text{O} \rightarrow 4\text{Fe}^{3+} + 4\text{OH}^- \quad (6)
\]

\[
\text{Fe}^{3+} + 3\text{OH}^- \rightarrow \text{Fe(OH)}_3 \quad (7)
\]

The oxidation of sulphide sulphur to sulphate follows the mechanism of oxidation of thiosulfate to sulfate, as shown in equations (8-10):

\[
2\text{S}^{2-} + 2\text{O}_2 + \text{H}_2\text{O} \rightarrow \text{S}_2\text{O}_3^{2-} + 2\text{OH}^- \quad (8)
\]

\[
\text{S}_2\text{O}_3^{2-} + 2\text{OH}^- + 2\text{O}_2 \rightarrow 2\text{SO}_4^{2-} + \text{H}_2\text{O} \quad (9)
\]

\[
2\text{S}^{2-} + 4\text{O}_2 \rightarrow 2\text{SO}_4^{2-} \quad (10)
\]

Equations (11–14) highlight the oxygen and cyanide consuming reactions, characterised by pyrrhotite, the most reactive iron sulfide present in the ore. These reactions compete for cyanide and oxygen with the gold dissolution reaction (Mahlangu et al. 2007):

\[
\text{S}^{2-} + 2\text{CN}^- + \frac{1}{2}\text{O}_2 + \text{H}_2\text{O} \rightarrow \text{CNS}^- + 2\text{OH}^- \tag{11}
\]

\[
\text{Fe}_2\text{S}_8 + \text{CN}^- \rightarrow 7\text{FeS} + \text{CNS}^- \tag{12}
\]

\[
\text{FeS} + 6\text{CN}^- + 2\text{O}_2 \rightarrow \text{Fe(CN)}_6^{4-} + \text{SO}_4^{2-} \tag{13}
\]

\[
4\text{Au} + 8\text{CN}^- + \text{O}_2 + 2\text{H}_2\text{O} \rightarrow 4\text{Au(CN)}_2^- + 4\text{OH}^- \tag{14}
\]

Also the sulphide oxidation intermediate reaction products pick up significant amounts of cyanide (reactions 11–14). At the end of the day, if not curtailed by careful control of the preoxidation conditions, the pre-oxidation stage can result in a disastrous scenario as far as the economics of gold recovery is concerned in these leach systems.

This paper presents the analysis and development of the understanding of the preoxidation stage and its impact on the downstream processes at Kibali Gold Mine Processing plant. The main focus is on the key operating parameters interactions, measured against concentrate leach performance on namely (i) leach residue and (ii) reagents consumption as response parameters.

### Plant data analysis and experimental procedures

**Plant data analysis**

This work examines the daily plant data generated from the commissioning of the concentrate leach circuit to the end of 2018. During this period, the circuit has undergone various changes in configuration mainly focussing on the response of the plant residues and reagents consumption. The data analysis was carried out using:

i) Simple time based plots;

ii) Cumulative summation plots;

iii) Statistica Software to generate three parameter interdependence plots and correlation coefficients.

**Aachen assisted extended leach (AAEL) tests**

From the data analysis and general plant performance, it became apparent that the original installation and circuit configuration needed to be revisited to ensure that an increase in concentrate throughput was coupled with an efficient circuit in terms of lower solids residue values. The circuit moved from a:

i) straight two stage pH controlled preoxidation – single stage intensive cyanidation and six stage carbon in pulp adsorption stage to;

ii) two stage pH controlled preoxidation – single stage intensive cyanidation leach followed by a two stage Aachen assisted extended leach with carbon in pulp stages upgraded to an 8 stage configuration.

Aachen Assisted Leach (AAL) refers to the application of Malegwyn Mineral Services Africa (MMSA)’s high shear oxygen dispersion devices coupled with
simultaneous addition of cyanide but preceding the adsorption phase. After various permutations of testing, the leach testwork protocols settled on the one shown in Figure 1. An extended leach component was added for Kibali ores, hence the terminology Aachen Assisted Extended Leach (AAEL) used in the rest of the paper. These protocols were developed between KGM processing plant and MMSA metallurgical laboratories. The essential elements of the testwork and the subsequently implemented extended leach circuit, ensured the use of an intensive oxygenation process at high oxygen pressure and coupled with high shear on both preoxidation and cyanidation stages.

**Experimental procedure – underground stopes ore samples**

Bulk fresh ore samples were collected from both diamond drill core and underground development drive faces and muck stockpiles. Samples were then dried, crushed and milled to 80% passing 106 µm to generate flotation feed. Bulk rougher flotation tests were conducted using the current flotation reagents suite, with a target mass pull of 10% and concentrate grade of ± 25 g/t Au. The flotation concentrate was then milled in a laboratory ultrafine grinding mill to 80% passing 18 µm prior to leaching via the optimised 3 stage AAEL process.

**Ore reactivity characterisation and gold deportment**

In order to further understand the plant performance, two experimental tests were conducted, namely:

**Reactivity and/or oxygen demand characterisation**

The characterisation process included preoxidation, preleach and extended leach products. Reactivity tests, were conducted on site on samples collected directly from the plant, by measuring oxygen demand and computed to a reactivity number (RN), which is unique to each type of material and/or stage of processing. Computation of RN remains intellectual property (IP) to M Verster Consulting and is beyond the scope of this paper.

**Mineralogical examination of feed and composite plant leach residues:**

Samples were collected daily and monthly composites generated. These were submitted to AMTEL, Canada, during 2018, as part of an ongoing exercise to further consolidate the current stable performance of the circuit. Gold deportment analysis followed a standard AMTEL procedure which quantifies each form and carrier of gold from a leach perspective, using a compressive mineralogical and analytical approach:

i. Full multi-element analysis
ii. Quantitative X-Ray Diffraction (QXRD) Analysis
iii. Grain size and association characterisation of minerals by microscopy
iv. Determination of gold composites using Scanning Electron Microscope (SEM)/Energy Dispersive X – Ray (EDX) analysis
v. Quantification of sub-microscopic gold in pyrite, marcasite and iron oxides using Secondary Ion Mass Spectrometry (SIMS)
vi. Quantification of gold losses or adsorbed onto fine carbon using Time-of-Flight Laser Ionisation Spectrometry (TOF-LIMS)

Because of the associated costs, these tests are limited and get staggered over a period of once in a quarter to ensure that a full understanding of the quarterly plant performance is generated.

**Results and discussion**

**Plant data analysis**

Figure 2 shows a plot of the concentrate treatment plant data (i) pumpcell circuit feed (PCF) tonnes; (ii) pumpcell circuit feed grade and (iii) pumpcell circuit tails (PCT) solids residue. It is apparent from the data that PCT solids residues were strongly dependent on both tonnes and feed grade coming into the circuit until about end of 2016. This residue grade dependence, however, has been less sensitive to the two variables since. Pumpcell Circuit Feed tonnes have nearly doubled over the years with the transition from a pure parallel oxide – sulphide to full twin sulphide campaign circuit, where concentrate tonnes have increased from an average 1200 tpd to close to 2500 tpd.
Apart from the preceding parameters of concentrate tonnes and concentrate feed grade, the plant leach efficiency is controlled and tracked based on mainly

i. pH thus lime consumption
ii. dissolved oxygen (DO) concentration
iii. temperature
iv. cyanide consumption

In order to understand the impact of each of these parameters on the circuit efficiency, both straight graphical plots with time and cumulative summation plots have been analysed. Figure 3, shows the cumulative summation plot for the PCT residues for the period April 2014 to December 2018. This data practically defines different time periods in the operation of the plant. There are seven broad durations that define progressive changes to the circuit performance and summary of individual (i) average, (ii) standard deviation, (iii) maximum, (iv) Minimum, and (v) different levels of percentiles. These are also summarised in Table 1 and Figure 4.

It is clear from Table 1 and Figure 4, that the circuit progressive improvement was disrupted in the time period of October 2015 to July 2016 (D). Selected time periods of start-up (A); first stable period (C); disruption period (D); and the current stable period (G); are plotted in Figure 5.

**Process parameters interactions**

From the preceding discussion (Plant Data Analysis), the key parameters of pH and dissolved oxygen in the preoxidation stage were analysed to understand their impact on the concentrate leach residue grades. The focus of this work is on the second full sulphide commissioning and stable operation of the circuit, namely periods D to G.
The transition between Zones or periods C to D coincided with the second stage commissioning of the full twin sulphide. In order to understand the impact of this phase, the plant data was analysed based on the full commissioning phase of period D and the current stable operational phase of G. Key physical changes to the circuit were made, that is:

i Doubling the ultrafine grinding capacity from 4 to 8 mills;
ii Increasing the adsorption circuit from a 6 to an 8 stage circuit
iii Testing and commissioning of the Aachen Assisted Extended leach circuit

The data in Figure 6 illustrate the interaction between preoxidation parameters of dissolved oxygen (DO) and pH as well as their impact on the final concentrate leach residues. It is clear from the three dimensional (3-D) plots (Figure 6(a,b)) that the effective operating zones that gave lower solids residues correspond to running pH and DO above the average values for the given operating periods. Further analysis of the actual data for the two periods reveals that zone G operated at a higher average pH of 10.4 as compared to zone D at pH 9.93. The impact of higher and stable pH is clearly demonstrated in Figure 7(d), where pH has an inverse correlation to PCT residues.

In the related work (Figure 8) of the plant optimisation audit, it was demonstrated that higher pH values tend to drop the sulphide reactivity and as result reduce oxygen demand.

The Reactivity Number (RN) is defined as a measure of oxygen demand of feed ores and slurry flows in leach circuit (Verster, 2017). The key factors defining RN are:

(i) sulphide minerals present, i.e. pyrite vs pyrrhotite vs arsenopyrite; (ii) degree of ultrafine grinding – relating to surface area and deleterious elements liberated; and (ii) pH adjustment and cyanide addition which alters surface characteristics of the sulphides. The consequence of this observation has been to work on the addition of lime into the ultrafine grinding product prior to the high shear and intensive oxygenation pre-oxidation steps. The data in Figure 8 demonstrates that lime addition reduces the sulphide sample’s reactivity by 75%.

The main objective of the current Kibali Gold Mine Processing plant is to ensure the generation of stable passive surfaces that inhibit oxygen and cyanide consumption side reactions in the leach steps. In order to understand the real benefit of preoxidation on cyanide consumption, cyanide consumption was plotted as a function of both pH and dissolved oxygen in the

<table>
<thead>
<tr>
<th>Period</th>
<th>Start date</th>
<th>Finish date</th>
<th>Average (g/t)</th>
<th>Standard Dev (g/t)</th>
<th>Minimum (g/t)</th>
<th>Maximum (g/t)</th>
<th>25th Percentile (g/t)</th>
<th>50th Percentile (g/t)</th>
<th>75th Percentile (g/t)</th>
</tr>
</thead>
<tbody>
<tr>
<td>B</td>
<td>29-Oct-14</td>
<td>6-Jan-15</td>
<td>5.90</td>
<td>3.00</td>
<td>1.88</td>
<td>15.99</td>
<td>3.90</td>
<td>5.05</td>
<td>7.61</td>
</tr>
<tr>
<td>C</td>
<td>7-Jan-15</td>
<td>23-Oct-15</td>
<td>3.72</td>
<td>1.79</td>
<td>0.99</td>
<td>12.51</td>
<td>2.49</td>
<td>3.45</td>
<td>4.48</td>
</tr>
<tr>
<td>D</td>
<td>24-Oct-15</td>
<td>5-Jul-16</td>
<td>6.40</td>
<td>2.41</td>
<td>1.96</td>
<td>16.06</td>
<td>16.06</td>
<td>4.83</td>
<td>5.82</td>
</tr>
<tr>
<td>E</td>
<td>6-Jul-16</td>
<td>3-Oct-16</td>
<td>4.73</td>
<td>1.27</td>
<td>2.78</td>
<td>9.79</td>
<td>3.92</td>
<td>4.41</td>
<td>5.22</td>
</tr>
<tr>
<td>F</td>
<td>4-Oct-16</td>
<td>31-Dec-17</td>
<td>3.43</td>
<td>1.00</td>
<td>0.88</td>
<td>9.17</td>
<td>2.72</td>
<td>3.32</td>
<td>4.01</td>
</tr>
<tr>
<td>G</td>
<td>1-Jan-18</td>
<td>31-Dec-18</td>
<td>3.02</td>
<td>0.49</td>
<td>1.75</td>
<td>4.63</td>
<td>2.69</td>
<td>2.99</td>
<td>3.35</td>
</tr>
</tbody>
</table>

**Twin full sulphide stream (Jan 2016: zone D) through to full plant stability (Zone G: Jan 2017–2018)**

Figure 4. Progressive performance of the concentrate leach circuit based on the PCT solids residues (April 2014 to December 2018)
preoxidation stage. This is shown in Figure 9, where data was analysed for Zone G. It is clear that either at a higher DO and pH or low DO but still higher pH, the cyanide consumption was minimised.

When plotted individually (Figure 10) as functions of cyanide demand, it also becomes apparent that cyanide demand is not entirely a function of the two systems as demonstrated by the significant variation or

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**Figure 5.** Progressive improvement of plant residue from commissioning to first stability (A to C) and after second stage commissioning to current stability (D to G).

**Figure 6.** Interactions between preoxidation dissolved oxygen (a) Period D – Oct 2015 to July 2016; (b) Period G – Jan to Dec 2018; and pH (c) Period D – Oct 2015 to July 2016; and Period G – Jan to Dec 2018: on the concentrate leach solids residue.
fluctuation. This could be a function of the ore type fed in the system, primarily high pyrrhotite which is fairly common in the different lodes of the underground orebody.

Although pyrrhotite remains a minor component of the main orebody fed into the plant it has been demonstrated during the extended leach tests conducted on different geometallurgical samples that cyanide

**Figure 7.** Effect of preoxidation dissolved oxygen: (a) Period D – Oct 2015 to July 2016; (b) Period G – Jan to Dec 2018; and pH (c) Period D – Oct 2015 to July 2016 and Period G – Jan to Dec 2018; on the concentrate leach solids residue.

Kibali Sulphide feed: RN response to Lime & NaCN addition

- **y = -0.8872x + 33.954**
- **R² = 0.9673**

**y = -0.7467x**
- **R² = 0.9848**

**Figure 8.** Effect of lime and cyanide addition on the reactivity of sulphide concentrate at Kibali Gold Mine Processing Plant (Verster, 2017).
consumption can be in excess of 10 kg/t of concentrate. The following sections attempt to characterise the leach concentrate samples as well as the impact of ultrafine grinding on the ore reactivity.

Ore reactivity characterisation and mineralogical examination

Ore reactivity characterisation

Figure 11 below shows the snapshot of the reactivity number measurements conducted on the Kibali Gold Mine circuit during the months of September–October 2017. The data was generated from the two stage preoxidation followed by intensive cyanide leach part of the circuit. Oxygen demand, that is, ore reactivity was found to be high and variable. Further tests were conducted to understand the impact of solids sulphur content and grind. The results are shown in Figures 12 and 13 and it is apparent that the higher total sulphur in the solids increases the reactivity number. Achieving finer grind has a more significant impact on the reactivity number related to increased surface area. In this respect, both systems are bound to generate cyanicides based on reactions 10–13. In fact, the current plant practice is semi-optimised on ± 25 g/t Au. The later ensures that apart from managing cyanide and oxygen consumption rates, ultrafine grinding media consumption is accurately controlled based on optimal flotation mass pull of 10–12%.

Aachen assisted extended leach (AAEL) tests

Typical laboratory extended leach tests results are shown in Figure 14. The key elements that are clarified and quantified in the circuit optimisation are: (i) dissolved oxygen; (ii) pH and/or lime consumption; (iii) cyanide consumption; and (iv) stage dissolution or residue values achieved. Lime and cyanide consumption presented in Figure 14 were calculated per tonne of concentrate and translated to typical plant consumption values (calculated per tonne of ore treated) at a 10% mass pull. In the circuit optimisation and/or progression, Aachen Assisted Extended Leach (AAEL) was implemented during period F (Figures 2 and 4) and it can be seen that since then, PCT solids residue grades have been stable notwithstanding an increase in both concentrate tonnage and feed grade (Figure 1).

The results of the AAEL tests presented in Figure 14 were generated from processing samples from two underground samples generated from part of the ore body with a predominantly pyrrhotite mineralisation. Cyanide and lime consumption are reported in kg/t of concentrate with typical flotation mass pulls of between 10 and 11%. Under these conditions of lower concentrate head grade (21 g/t), concentrate leach residue grades as low as 2 g/t are feasible. The developed test protocols in Figure 1, are currently used in both AAEL and Extended Bottle Roll (BRT) leach tests of concentrate for routine work and advanced testwork in tracking various
Figure 10. Interaction between individual (a) pH and (b) DO, with cyanide demand of the circuit (Zone G: Jan to Dec 2018)

Figure 11. Reactivity number or oxygen demand survey plots for the concentrate preoxidation circuit (September – October 2017).

Figure 12. Impact of sulphur in the solids on the reactivity number (RN) – data collected on the Kibali Gold Mines Concentrate leach circuit (Verster, 2017)
ore blends and three to six months feeding rolling plans.

**Mineralogical examination of concentrate feed and leach residues**

Results of the mineralogical testwork conducted at AMTEL in 2018 are shown in Figures 15 and 16 in the form of gold deportment. In the first instance, the accurate identification and quantification of carrier minerals showed that pyrite was the major sulphide present with arsenopyrite and pyrrhotite in minor proportions. The latter are known cyanicides and oxygen consumers due to their highly reactive nature.

The assayed grade of the concentrate leach residue for October (Figure 15) and November (Figure 16) were 3.11 and 3.13 g/t, respectively. The key element of this work was to understand the efficiency and identify opportunities for circuit leach improvement based on the understanding of the full deportment of gold in the residues. The bulk of gold remaining in the tailings was in excess of 70% in the form of sub-microscopic gold, which cannot be recovered in the current Kibali Gold Mine processing plant circuit configuration. On...
the other hand, coarser gold locked in sulphide (17–19%) present an opportunity for additional 0.5 g/t gold recovery from residues. Additional or finer grind will liberate gold but a balance has to be achieved between increased surface area/reactivity of sulphides which translate to high cyanide and oxygen demand (Reactions 11–13) and additional gold recovery (Table 2).

From the preceding data it is clear that the plant needs to review the grind efficiencies, taking into account the gold loses at the current achieved grinds of 80% passing 18 µm.

**Figure 15.** Gold deportment in the Kibali Gold Mines Concentrate leach residues – Monthly composite sample October 2018.

**Figure 16.** Gold deportment in the Kibali Gold Mines Concentrate leach residues – Monthly composite sample November 2018.
**Conclusions**

1. Overall, plant data analysis based on concentrate leach residue grade as the response parameter demonstrated that:
   1. Transition from single stream sulphide to a full twin stream sulphide disrupted circuit stability;
   2. Full reconfiguration of the circuit and stabilisation of pH and dissolved oxygen control has stabilised the circuit in terms of both cyanide demand and concentrate leach residues;
2. Preoxidation pH and DO, have the strongest interaction between themselves and also to concentrate leach residue grade and cyanide consumption;
3. High pH (> 10.2) has been demonstrated to reduce reactivity of the sulphides and thus drives down cyanide consumption: possible less aggressive conditions favourable to less generation of cyanicides;
4. Circuit audit in terms of reactivity coupled with mineralogical examination of concentrate leach residues have demonstrated an opportunity to reduce residues further by ultrafine grind (> 15% of gold losses are related to inefficient grinding). The latter must be done in cognisance of the fact that reactivity has an exponential response hence a spiral effect on downstream leach processes;
5. From the extensive analysis of plant data and laboratory testwork carried out over the years, the Kibali Mine Processing plant has established and defined a niche operating domain for optimal pre-oxidation processes;
6. Niche operating domain: pH above 10.2 and DO between 6.12 and 6.9 ppm;
7. Current stable residue averaging 3.1 g/t, with an opportunity to drop to 2.5 g/t from the mineralogical data grind based estimation;
8. While this work is not exhaustive in its circuit analysis, it is considered a fair attempt to demonstrate the operational philosophy currently obtained at Kibali Gold Mine Processing plant;
9. Additional work on the feed blend vs residue as well as interactions between parameters present another opportunity in optimising the circuit. At this stage, this is not covered in this work.

**Acknowledgements**

The authors would like to express sincere gratitude to the Management of Kibali Gold Mines and the Group Technical Executive team of Barrick Gold Corporation for allowing the publication of this paper. The invaluable input of AMTEL Laboratories (Mineralogical Examination and gold Deportment), MMSA (development of the Aachen Assisted Leach Testwork) and Mr Martin Verster (Hydromet consultant) on the oxygen demand circuit characterisation and circuit audits, is greatly appreciated.

**Disclosure statement**

No potential conflict of interest was reported by the authors.

**Notes on contributors**

Thamsanqa (Tham) Mahlangu holds a Ph.D. in Hydrometallurgy, with 25 years working experience both in Research and Academia (17 years) as well as Projects and production environment (8 years). Was part of the project team that worked through feasibility, construction, commissioning and optimisation of the Kibali Gold Mine Processing Plant. Currently, providing technical support to the production team and also developing and implementing Geomet Models for the various ore bodies being processed at Kibali. Tham is a full member of the South African Institute of Mining and Metallurgy (SAIMM) and is registered as a Professional Engineer with the Engineering Council of South Africa (ECSA).

F. A. Sumaili is Metallurgist at Kibali Gold Mine with a wide variety of practical application in mineral processing and

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**Table 2.** Summarises the comparative analysis of gold losses for both months of October and November 2018, with actions that the plant has undertaken to improve the circuit performance.

<table>
<thead>
<tr>
<th>Concentrate Leach Residues (October &amp; November 2018)</th>
<th>October 2018</th>
<th>November 2018</th>
<th>Comments</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Au (g/t)</td>
<td>Dist</td>
<td>Au (g/t)</td>
</tr>
<tr>
<td>Assayed grade</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Carrier of Au</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Free gold grains</td>
<td>0.011</td>
<td>0.04%</td>
<td>0.025</td>
</tr>
<tr>
<td>&gt;7µm</td>
<td>0.047</td>
<td>1.5%</td>
<td>0.046</td>
</tr>
<tr>
<td>&lt;7µm</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Attached gold grains</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>With sulphides</td>
<td>0.029</td>
<td>0.9%</td>
<td>0.060</td>
</tr>
<tr>
<td>With rock</td>
<td>0.003</td>
<td>0.1%</td>
<td>0.009</td>
</tr>
<tr>
<td>Enclosed gold grains</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>With sulphides</td>
<td>0.528</td>
<td>16.9%</td>
<td>0.594</td>
</tr>
<tr>
<td>With rock</td>
<td>0.011</td>
<td>0.4%</td>
<td>-</td>
</tr>
<tr>
<td>Sub microscopic gold</td>
<td>2.44</td>
<td>77.9%</td>
<td>2.270</td>
</tr>
<tr>
<td>Adsorbed on Attrited</td>
<td>0.062</td>
<td>2%</td>
<td>0.167</td>
</tr>
<tr>
<td>Act. C</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Total % of Head assay</td>
<td>3.131</td>
<td>100.6%</td>
<td>3.171</td>
</tr>
</tbody>
</table>
gold hydrometallurgy. He graduated Chemical Engineer in 2015 from the University of Lubumbashi in Democratic Republic of Congo and started his career as technical sales representative at Maestro of oil and gas solutions (Congo) prior to join Kibali in 2016. He currently lives in the region of Watua/Haut Uele (DRC) and enjoys facing new challenges.

**Dieudonne Ntamb Ayiz** is Assistant Plant Manager at Kibali gold Mine with an open knowledge in mineral processing, gold and copper hydrometallurgy. He is graduated in gold Mine with an open knowledge in mineral processing, Dieudonne Ntamb Ayiz of Watsa/Haut Uele (DRC) and enjoys facing new challenges. Prior to join Kibali in 2016. He currently lives in the region of Watua/Haut Uele (DRC) and enjoys facing new challenges.

**B. M. Sindani**, an engineer with 16 years’ experience in the mining industry, starting his early career in the base metals (Copper–Cobalt), before moving to the gold industry. Beddy joined Kibali Gold Mine in 2013 as Technical superintendent, now serving as Process Manager, Beddy has a B.Sc. (Hons) in Metallurgical engineering (University of Lubumbashi).

**P. Mande**, an engineer with 14 years’ experience in the mining industry, starting his early career in the base metals (Copper–Cobalt), before moving to the gold industry. Patrick joined Kibali Gold Mine in 2017 as Metallurgical Plant Manager, now serving as Operations Manager responsible for the Mining, Processing and Supply Chain operations. Patrick has a B.Sc. (Hons) in Metallurgical engineering (University of Lubumbashi).

**Gustav Du Toit** is a General Manager with over 20 years of experience in the design, construction and operation of mineral processing plants in multiple commodities. These include gold, diamonds, chrome, manganese and iron ore. He is familiar with all aspects of process engineering, having been responsible for the lead process engineering and engineering management function on numerous projects and engineering studies. He has significant operational experience on gold and diamond processing plants within South Africa, Botswana as well as West Africa. Gustav holds a National Higher Diploma in Extractive Metallurgy, a Master’s Degree in Mineral Resource Management and is an associate member of the South African Institute of Mining and Metallurgy (SAIMM) and is registered as a Professional Technologist with the Engineering Council of South Africa (ECSA).

**Martin Verster** operates as an individual consultant in the gold hydrometallurgical field, and is based in South Africa. He holds bachelor degrees Mechanical and Metallurgical engineering from the University of Pretoria (South Africa). He has been consulting to gold mines globally since 2009, with a focus on circuit optimisation through gold leach kinetics maximisation. His methodology includes characterisation of feed ores, determination of optimal NaCN/DO leach ratios and application of best practice dosage technologies for leach kinetics reagents like cyanide, oxygen, hydrogen peroxide and lead nitrate.

**Solomon Mogashoa** is the Applications Engineer for Maegwyn Mineral Services Africa and holds a degree in Bachelor of Engineering (Metallurgical Engineering) from the University of Pretoria in South Africa. He has more than 8 years of experience in gold processing both in production and laboratory environment. He is actively engaged with cyanide destruction processes and has a great interest on environmental and sustainability-related matters.

**Mr. P. Latz**, born 1956, obtained a Swiss BTech 1975 in organic chemistry in Basel/Switzerland. His metallurgical work experience ranges from research and development conducted at MINTEK, South Africa where he ended leading the gold focused hydrometallurgy group to the application of Maelgwyn Mineral Services’ technology in the wider gold mining industry. The main focus of his activities has been gold extraction, specifically from refractory ores as well as linked sustainability issues such as cyanide and arsenic management. He has an accumulative 30 years of metallurgy- and mining-related experience.

**References**


