Precious Metals '10
Falmouth Beach Hotel, Falmouth, UK, June 15-16, 2010

Sponsored by:

NEWMONT The Gold Company   MAELGWYN MINERAL SERVICES   BARRICK   DELKOR

Tuesday 15th June

09.00 *Opening Remarks*
B.A. Wills (MEI, UK) and M.D. Adams (Mutis Liber Pty Ltd, Australia)

09.20 *Technical Session 1*
Chairmen: M. Battersby (Maelgwyn Mineral Services Ltd, UK) and C. Pickles (Queen's University, Canada)

09.20 **Modern gold deportment studies**
L.L. Coetzee, S.J. Theron and G.J. Martin (SGS South Africa, South Africa)

Modern Gold Deportment Studies include physical, chemical and mineralogical assessments which are combined to obtain a full understanding of the nature and variability of gold in the resource. The objective of a gold deportment study is to provide information which will allow a cost effective and practical processing route.

The distribution of gold, based on speciation, grain size and mode of occurrence (liberation, exposure and mineral association) is quantitatively determined by means of Scanning Electron Microscopic Techniques (QEMSCAN/MLA). Furthermore, general mineralogical characterization is undertaken in order to characterize the gangue components; with special emphasis on deleterious characteristics of the ore (e.g. cyanide consumers like secondary Cu-species, preg-robbers, passivation due to Sb-minerals or As-minerals and oxygen consumers like pyrrhotite/marcasite).

Predictions based on the mineralogical observations are confirmed by physical and chemical testwork. These include grading analyses, heavy liquid separation or gravity tests, direct cyanidation tests and diagnostic (sequential) leach tests.

09.40 **Sonochemical synthesis of gold nanoparticles using microemulsions**
Q. Losper, Z. Du Toit, Z and C. Aldrich (University of Stellenbosch, South Africa)

This paper explores the use of microemulsions in the sonochemical synthesis of colloidal gold. To theoretically characterize the sonochemical effects, a numerical simulation was done using the Rayleigh-Plesset equation. From this the average life time, bubble radius and localized temperature and pressure were calculated. The colloidal stability was also theoretically investigated by calculating the interaction energies of the particles at different sizes and particle-particle separation distances. Experimental results have showed that gold particles produced in microemulsions with a co-surfactant had a relatively small size (in the range of the reverse micelles droplet size) and exhibited an almost uniform size distribution, whereas the gold nanoparticles produced in a microemulsion without a co-surfactant also showed relatively small particles, but these particles were embedded between larger particles. It was also found that microemulsions produced much more stable colloidal systems that allow for longer irradiation times. These long irradiation times ensure that the Au (III) is completely reduced.

10.00 **Modern leach test work with simultaneous focus on process and sustainability**
S.J. van Rensburg and P.W. Lotz (Mintek, South Africa)

Traditionally, gold test work focused largely on gold recovery only. Sustainability issues were dealt with separately or not at all. Immense competition for resources such water and arable land have refocused attention on water and tailings management with respect to quantity and quality. As it makes more sense to tackle problem issues at source, there exists a strong need for additional leach process information to be generated that enable the mine metallurgical staff's proactive decision making.

Aimed at test work to be conducted on refractory sulphide ores, reality for the majority of cases these days, MINTEK developed the ALF (Advanced Leach Facility).

This facility makes it possible to modify and measure the process parameters that can be realistically changed on an operating gold plant such as: pH, Eh, oxidant addition, pulp viscosity, cyanide concentration (cyanide staging) and support reagents such as lead nitrate. During experimental execution, the impacts of such changes on gold dissolution and sustainability parameters are followed. This ensures a holistic view on the process from the start. Data highlighting the benefits of this approach is presented.

10.20 **The effect of pulp rheology on the rate of gold cyanide leaching and adsorption on activated carbon, and its impact on the design of CIL and CIP circuits**
C.A. Fleming, A. Mezei, E. Bourricaudy, M. Canizares and M. Ashbury (SGS Minerals, Canada)

The CIL and CIP processes are very robust, and high gold recoveries are achievable even after significant changes in feed composition or during times of plant upset. Consequently, little attention is paid to circuit optimization at the feasibility and plant design stages. Yet optimization of plant design and operating parameters can have a significant effect on both capital and operating costs. In particular, the impact of slurry rheology is generally overlooked, and the operating philosophy is to thicken the leach feed to the maximum practical thickener underflow density after milling, in the belief that this will minimize the size of the leach and adsorption plants and generate the lowest capital cost. But the kinetic gains that can be made by operating both the leach and adsorption sections with more fluid slurries at lower solids densities will often outweigh the losses suffered from increased volumetric flows, resulting in smaller tank sizes and improved operating performance.

The SGS Minerals approach to designing CIP and CIL circuits is to conduct rheological studies early in the life of a project, preferably prior to cyanide leaching and carbon adsorption test work. This allows the critical solids density of the leach slurry (CSD) to be determined, which is predictive of the maximum practical thickener underflow density that can be achieved. This value is then typically used to select three slurry densities (CSD, CSD – 5% and CSD – 10%, for example) for conducting batch tests to determine the rates of gold leaching and adsorption on activated carbon. The kinetic data are then fitted to rate equations, which are modeled to predict profiles of gold in solution, in the ore and on the carbon across multi-stage leach and carbon adsorption circuits. The model allows the capital and operating costs of the gold recovery circuit to be minimized and the metallurgical performance optimized with respect to parameters such as:

- pulp density of the feed,
- residence time of slurry in leaching and adsorption,
- choice of CIP or CIL for the adsorption circuit,
- number of adsorption stages,
- amount of carbon in each of the adsorption stages,
- rate of transfer of the carbon through the adsorption circuit
- size of the elution and regeneration plants.

This approach will be presented in the paper, with specific examples from projects that have been carried out in recent years.

10.40 Coffee

11.20 **Effect of background reagents on thiosulfate leaching of gold**
G. Senanayake, X.M. Zhang and M.J. Nicol (Parker Centre, Murdoch University, Australia)

Gold leaching in thiosulfate media has attracted renewed interest of many researchers due to fast kinetics in the presence of some oxidants/ligands and adaptability to gold ores which are unsuitable for cyanidation. Systematic studies based on chemical species, potential-pH diagrams and measured rate data based on flat surfaces and particles can shed light on the reaction mechanism which leads to investigate beneficial reagents for gold leaching in thiosulfate media. This paper describes a comparative study of the kinetics of dissolution of gold colloids, powders and foils based on rates per unit surface area and heterogeneous kinetic models. A comparative study is also made between the effects of $NaNO_3$, Na_2SO_4, Na_2CO_3, Na_2SO_3, NaCl and $AgNO_3$. The results are rationalised on the basis of the effect of chemical speciation on surface reaction.

11.40 **Use of AMIRA leach and adsorption modeling and simulation at Newmont Carlin Mill 5**
J. Orlich and A. Chung (Newmont, USA)

Newmont Mining Corporation's Carlin Mill 5 processes on the order of 12,000 – 15,000 tons per day through a flowsheet consists of SAG mill/ball mill grinding, froth flotation of sulfide minerals, followed by cyanidation of the flotation tailings. In 2007, the leach circuit consisted of two parallel rows of 4 CIL tanks each when an internal trade-off study was completed to evaluate options to increase gold recovery from the tailings leach circuit. Options considered included modifying the carbon screens to allow operation as 8 tanks in series, and/or conversion to CIP with the leach tank capacity. In order to properly evaluate the options, the AMIRA Leach & Adsorption Model made available through Newmont's sponsorship in AMIRA's P420C Project was selected a way of simulating the metallurgical effects of leach circuit modifications.

This presentation will describe the successful application of the AMIRA model to justify the installation of two CIP leach tanks to Mill 5 and discuss the projected versus actual results achieved.

12.00 **The difficulty associated with the estimation of gold inventory in large carbon in leach tanks**
P. Blatter, G. Blanchette (Agnico Eagle, Canada) and C. Bazin (Laval University, Canada)

The preparation of the month end gold production and financial balance is usually required by the mine owners to regularly assess the performance of the plant against budgeted values. However the preparation of a gold production report is often complicated by the large inventory of gold in the Carbon in Pulp (CIP) or Carbon in Leach (CIL) tanks used to adsorb gold from the leached solution. Indeed a 600 m^3 CIP/CIL loaded with 15 g/L of activated coal charged with 8000 g of gold/ton of coal may retain up to 72 000 g (2200 troy oz) of gold. The estimation of the inventory implies the estimation of the concentration of carbon in a tank and the estimation of the concentration of gold loaded onto the carbon. This paper assesses the errors associated with the estimation of the gold content in the tank of a CIL circuit and the impact of these errors on the gold production balance of a month.

12.20 **Morphology of gold particles generated in different grinding environments; implications on particle separation**
G. Ofori-Sarpong (Pennsylvania State University, USA) and R.K. Amankwah (University of Mines and Technology, Ghana)

The shapes of malleable metal particles, such as that of gold, change based on the forces applied during comminution. The shapes generated have implications on downstream processes such as classification and gravity concentration that are influenced by particle shape. In this study, the morphology of gold particles milled in different comminution equipment was investigated. Comminution machines utilizing impact, shear and compressive forces generated globular cigar-shaped and flaky particles respectively. Based on shape-factors, the efficiency of subsequent classification and/or gravity concentration operations increased in the order of flakes < cigars < spheres. A flowsheet that could maximize gravity gold recovery is proposed for industries, such as the small-scale mining industry, that utilize the equipment.

12.40 **Microwave heating of gold ores for enhanced grindability, liberation and cyanide amenability**
R.K. Amankwah (University of Mines and Technology, Ghana) and G. Ofori-Sarpong (Pennsylvania State University, USA)

Minerals respond differently to microwave irradiation and thus for a rock containing several mineral components, selective heating results in thermal stress cracking. In this study a free-milling gold ore containing quartz, silicates and iron oxides was subjected to microwave irradiation leading to the generation of micro and macro-cracks. Both intergranular and transgranular fracture were confirmed by Scanning Electron Microscopy (SEM) and the crushing strength of the material was decreased by 30%. In addition to the reduction in crushing strength, gold was liberated from host minerals at a coarser size, resulting in a 12% increase in gravity gold. When the microwaved material was milled to 80% passing 75 µm, leaching time was shorter by 38% compared with similar samples milled to the same size without microwaving.

13.00 Lunch

14.00 *Technical Session 2*
Chairman: C. Bazin (Laval University, Canada)

14.00 **Understanding the discrepancy between prediction and plant GRG recovery for improving the gold gravity performance**
S. Koppalkar, A. Bouajila, C. Gagnon and G. Noel (COREM, Canada)

Gravity recoverable gold (GRG) content in a sample is determined using the standard three-stage test developed at McGill University by Laplante et al. A comparison of GRG recovery in laboratory and its actual recovery in plants show that plants recover between one-third and two-thirds of the predicted GRG from laboratory test.

Research conducted at COREM seeking the identification and the understanding of potential causes of this discrepancy reveals that particle shape transformation during grinding as the major contributor among other causes like bleed percent processed, under optimized cyclone and gravity concentrator operation.

For analyzing the gold particle shapes and their deportment, samples were collected from different streams of grinding-gravity (Knelson) circuit in a gold plant. In addition, concentrate sample from a laboratory Knelson concentrator (KC MD3) was also collected and analyzed. A particle characterization based on gold morphology capable of diagnosing typical shapes generated in gold grinding-gravity circuit was developed. The study revealed that the GRG particles reported to the cyclone overflow are the most flattened particles among the samples analyzed and on the other hand gold particles recovered in the laboratory Knelson concentrator are the least flattened, highlighting the potential cause of discrepancy between the two recoveries.

In this paper, the GRG mapping in a conventional Grinding-gravity circuit and the transformation of GRG to non GRG as mainly explained by gold particle reshaping will be discussed.

14.20 **Gold recovery from magnetic concentrates at Newmont Carlin Operations**
I. Amponsah, D. Collins, J. Orlich, J. Cole and D. Brosnahan (Newmont, USA)

Studies indicate that maghemite (Fe2.6 7O4) formation results from incomplete reaction of arsenic-rich ores in the Carlin Mill 6 roaster. The maghemite formed has the ability to trap high-grade gold making its formation in the roaster undesirable. Mill 6 roaster feed is predominately arsenopyrite/arsenian pyrite; hence, the formation of maghemite in the roaster resulting in lower than budgeted Mill 6 recovery. Mineralogy analysis of magnetic material in Mill 6 tails indicated 2.5:1 maghemite:hematite ratio. Ideally, hematite to magnetite ratio of 3:1 is desirable to ensure high gold recovery and low cyanide consumption in a CIL circuit.

Achieving higher gold recovery from Mill 6 is important, hence deporting gold from the magnetic material—maghemite—recovered from the tails to reduce gold losses through the tails is pursued. This paper describes a strategy for magnetic concentrate recovery plant at Mill 6, Carlin, NV that intends to increase the overall Mill 6 recovery by 1-2%. Bench and pilot plant scale testing led to the design of the plant and subsequent gold recovery from the magnetic concentrates.

14.40 **Implementation of decanter centrifuge technology in the efficient recovery of high grade platinum group metal concentrates**
C.A. Snyders, J.J. Eksteen (Lonmin, South Africa) and N. Schwarz (Schwarz Global Consulting, South Africa)

The precious metals industry is unique in that small equipment volumes can lock up significant value if the precious metals occur in concentrated form (i.e. in the % rather than ppm concentration range). This has a significant impact on the type of technology chosen to perform reactions and solid liquid separations.

Stock takes at the Lonmin Base Metals Refinery (BMR) plant in South Africa has shown significant lock-up of Precious Group Metals (PGM) at times in various sections of the plant, especially in the thickeners. Release of PGM's gives immediate revenue for which the cost of mining and smelting has long been accounted for.

The solid-liquid separation of PGM concentrates posed interesting challenges, as the media is highly corrosive and is at elevated temperature, whilst the particles are extremely small (all the particles below 30 micron and 18 % of the particles smaller than 5 micron). Moreover, the solid-liquid separation device had to be integrated into an existing multi-compartment-autoclave circuit that provided an erratic feed and where the overflow and thickened paste had to be processed continuously in based on a just-in-time processing philosophy. In addition, ramp-up and turn-down occur frequently depending on the short-term mining production profile, which is also impacted upon by upstream plants such as concentrators and the smelter, each of which introduces their own perturbations to the PGM ounces produced on a daily basis.

Scroll type centrifuges or decanters have been identified to replace one of the thickeners which have been shown to contain a significant amount of PGM's. Decanters are small and operate continuously on the basis of a difference in density between the solids and liquids with the material subjected to centrifugal forces which will move the denser material radially outward.

Although well established in many other industries (mining, food, sewage), centrifuge technology is not proven in the PGM industry and laboratory and pilot scale test work was therefore conducted first before installation on the plant. Particle size distribution, feed rate, viscosity, particle densities and solids content are factors that were taken into consideration for the design. The centrifuges have been commissioned and it has been shown that the centrifuges achieved its design separation efficiency at much reduced MIP.

15.00 **The pros and cons of reductive matte smelting of PGMs**
R.T. Jones and I.J. Geldenhuys (Mintek, South Africa)

Process changes are required in the smelting of PGM (platinum group metals) ore concentrates in order to accommodate the increasing amounts of UG2 concentrates that are being produced in the South African Bushveld Complex. These concentrates contain much higher levels of chromium oxides and much lower levels of base metal sulphides than the previously typical concentrates from the Merensky Reef.

Addition of carbon to PGM-containing concentrate in a DC arc furnace results in some of the iron (and other metallic) oxides being reduced to the metallic state. To some degree, the chromium oxides from UG2 concentrates will be reduced to CrO, thereby becoming more readily soluble in the slag. These advantages are similar to those enjoyed by the ConRoast process (where sulphur is removed in a fluidized-bed roaster prior to smelting, and the PGMs are collected in an iron-rich alloy), yet there are significant differences between 'reductive matte smelting' and 'reductive alloy smelting'.

Reductive matte smelting is constrained to produce a sulphur-rich matte product, either for reasons of crushability or for compatibility with an existing downstream process. This negates the advantage of the alloy having a high liquidus (melting) temperature that is compatible with that of the slag; and there is considerably higher risk of the matte breaking out of the furnace.

Because of the ever-present risk of furnace failure caused by super-heated matte, it is prudent to operate the furnace at a lower power intensity than is the case with the alloy-based ConRoast process. Based on work carried out in Mintek's 3 MW DC arc furnace at throughputs of over 1000 tons per month, it has been found that the PGM 'lockup' (inventory inside the furnace) in a process that used reductive matte smelting of UG2 concentrates was also much greater than that of reductive alloy smelting. Further differences exist in the better working environment and lower emissions of SO_2 in the case of the ConRoast process.

The proposed solution to the problem of smelting UG2 concentrates involves roasting of the concentrates to remove most of the sulphur prior to smelting, followed by reductive alloy smelting in a DC arc furnace. This process is unconstrained by the amount of chromium present in the feed, is much environmentally cleaner in terms of sulphur emissions, is much less prone to failure of furnace containment, and achieves very high recoveries of PGMs. The resulting alloy can be treated further either hydrometallurgically, or pyrometallurgically by converting to remove the iron to produce a product very similar to a conventional converter matte.

15.20 **Recovery by hydrometallurgical extraction of the platinum group metals from car catalytic converters**

D.J. de Aberasturi, R. Pinedo, I.R. de Larramendi, J.I.R. de Larramendi and T. Rojo (Universidad del País Vasco, Spain)

The car industry is one of the technological applications which more platinum group metals (PGM) employs. Therefore, the recovery of the PGMs from the car catalytic converters could be an important source to obtain these precious metals, with economic and enviromental consecuences.

In this work the car catalytic converters have been characterized through different techniques as X-ray diffraction, scanning electron microscopy and induced coupled plasma, and a new alternative to recover at least the 95% of the PGMs from the car catalytic converters is proposed.

In order to suggest an enviromentally friendly method for the PGMs recovery some conditions as the reagents concentration and composition, energy optimization and pre-treatment in hydrogen atmosphere are studied. In addition, the potential recovery or inertization of the rest of materials will be evaluated.

15.40 **Gold recovery from the Skaergaard deposit in Greenland**

M. Phillips, N. Ricketts (AMEC Minproc, Australia), T. Abraham-James and R. Mosig (Platina Resources, Australia)

The Skaergaard volcanic intrusion on the east coast of Greenland has been widely studied from a geological perspective for decades. The layered intrusion contains a number of layers with significant enrichment in gold and/or platinum group elements, largely dominated by palladium. Recent investigations into recovery of gold from one of the layers at Skaergaard have shown that the gold and PGM recoveries from flotation and gravity processing were very high. Rougher flotation gold recoveries of 88% were achieved into a mass of 3-6%. Gravity test work achieved 78% gold recovery at 75 µm into a mass yield of less than 20%. Subsequent cyanide leaching of the rougher flotation concentrates achieved greater than 99% gold recovery to solution in less than 6 hours. The better than expected recoveries have been attributed to the unique mineralisation at Skaergaard. The deposit was very low sulphur content and most of the gold is present as a gold cupride.

16.00 Coffee

Wednesday 16th June

09.00 *Technical Session 3*
Chairmen: R.T. Jones (Mintek, South Africa) and I. Amponsah (Newmont, USA)

09.00 **Hydrometallurgical processing of Platreef flotation concentrate**
M. Adams (Mutis Liber Pty Ltd, Australia), K. Liddell (Lifezone Ltd, Switzerland) and T. Holohan (Platmin Ltd, South Africa)

Ivanhoe Nickel and Platinum Ltd has been investigating potential alternatives to conventional pyrometallurgical matte smelting treatment of flotation concentrate arising from their Platreef project in the northern limb of the Bushveld Complex in South Africa. These potential alternatives are hydrometallurgical process routes to extract base metals and platinum group metals (PGMs) from flotation concentrate from ore containing ~0.36% Pt, 0.54% Pd and 0.09% Au, as well as 0.21% Ni and 0.13% Cu.

From a total of eleven hydrometallurgical process types that were considered potentially suitable for application to Platreef concentrates, four were selected for evaluation in a testwork programme comprising the assessment of over twenty process flowsheet variants. Consideration was made of value metal recovery outcomes, along with reagent consumptions and considerations such as commercial status, level of development robustness, scale-up potential and technical risks.

The hydrometallurgical flowsheet that could provide a viable process for treating Platreef concentrate is the Kell process, which broadly comprises the selective leaching of base metals by pressure oxidation, followed by the leaching of PGMs after a roasting pretreatment step. The process embodies a clear separation of the value metals (PGM and base metals into separate streams with minimal crossover) and impurity elements (Fe, Al and gangue tailings). Key elemental deportments from the unoptimized process were Ni 98%, Co 96%, Cu 96%, Pt 96%, Pd 94% and Au 87%.

Preliminary work indicates that further improvements in value metal recoveries, impurity separations and reagent consumptions will be gained from further optimization of the individual unit processes. Moreover, the recycle of key reagents may be feasible, rendering the process more economically and environmentally favourable. Initial assessment suggests that the Kell hydrometallurgical process route should offer both economic and environmental benefits over conventional pyrometallurgical matte smelting treatment of Platreef flotation concentrate.

09.20 **Experimental determination of the kinetics of pressure leaching of first stage leach residue and the solubilisation of Rh, Ru, and Ir**
C. Dorfling, G. Akdogan, S.M. Bradshaw (University of Stellenbosch, South Africa) and J.J. Eksteen (Lonmin, South Africa)

Hydrometallurgical process routes are typically used for separation of platinum group metals (PGMs) from base metals in Ni-Cu converter matte. Nickel dissolution is primarily achieved in the first stage leach (high pressure or atmospheric leaching, or a combination of the two), which is followed by second stage high pressure sulphuric acid/oxygen leaching to dissolve copper and the remaining nickel. PGMs are recovered from the leaching residue, and their dissolution must hence be limited. The leaching of base metals in the first stage has been characterised, but there is a limited understanding of the behaviour of metals, and more specifically PGMs, in the second stage pressure leach. This research presents the results of laboratory work performed to investigate the kinetics of leaching in the second stage pressure leach. The influence of key operating parameters such as the temperature, pressure, initial acid concentration, and slurry density on PGM dissolution was also investigated.

09.40 **The effect of the Fe-endpoint during Peirce-Smith converting on the atmospheric leaching characteristics of converter matte**
F. van Schalkwyk, G. Akdogan, E.L. Thyse (University of Stellenbosch, South Africa) and J.J. Eksteen (Lonmin, South Africa)

In the Lonmin process, Ni-Cu matte is converted to a specific iron end point, the matte is granulated. The granulated matte is subjected to leaching in the first stage of hydrometallurgical treatment. This research focuses on the atmospheric leaching of matte in the first stage, using an aqueous solution of sulphuric acid and copper sulphate. The function of the atmospheric leach is the dissolution of nickel, with the simultaneous precipitation of copper, rhodium, iridium and ruthenium from the copper spent electrolyte, via cementation reactions. The study investigates the effect that the residual iron content and related mineralogy of the converter matte on the acid leaching characteristics on which there is very little in depth study in the public domain. The influence of various other factors including copper and acid concentration on leach kinetics is also being investigated. By investigating the effects of abovementioned factors on the leaching characteristics of the matte, it is envisaged that a basis can be formed for a feed-forward control strategy, where the operating conditions in the leaching circuit can be adapted to compensate for variations in the iron-endpoint of the matte. Preliminary tests were performed in order to simulate the plant conditions in the laboratory scale reactor. The results from solution ICP and XRD of solids indicated fast kinetics in dissolution of Ni sulphides as well as metallic converter products. Also the kinetics of cementation of Cu appeared very fast. There is ongoing work to establish the effect of mineralogy in the leaching-cementation kinetics.

10.00 **The Leachox™ refractory gold process - the testing, design installation and commissioning of a large scale plant at the VASGOLD Gold Mine, Kazakhstan**

S. Flatman, R.M. Battersby, R. Imhof, M. Battersby (Maelgwyn Mineral Services Ltd, UK) and S. Ibrayev (JSC Vasilkovsky GOK, Kazakhstan)

The Leachox™ refractory gold process has to date been applied to the selective treatment of several refractory gold ores but on a relatively small scale. In this paper the successful implementation of the process, from initial testing through to commissioning, at one of the worlds largest refractory gold mine is described. The Leachox plant has been designed to treat 50 tons per hour of a sulphide concentrate. This is over 50% higher than the largest bio-oxidation plant currently operating worldwide and represents a major breakthrough in the treatment of traditionally difficult to process refractory gold ores. The process offers a low capital and operating cost route, but with similar high gold recoveries to bio-oxidation.

10.20 **Fungal-biotransformation of sulfides: a potential pretreatment process for refractory sulfidic gold ores**

G. Ofori-Sarpong, K. Osseo-Asare and M. Tien (Pennsylvania State University, USA)

Decomposition of sulfides in refractory gold ores is required to liberate submicroscopic gold particles that are usually locked up in the sulfides. Decomposition may be achieved by several techniques including microbial oxidation. This study assessed the capability of the fungus *Phanerochaete chrysosporium* to solubilize sulfides in order to develop a single-stage microbial process for pretreating double-refractory gold ores. The fungus has already proven its ability to effectively biotransform carbonaceous matter (CM) and reduce its preg-robbing ability. The extent of biotransformation was tested by analyzing for iron, sulfur and arsenic in incubation solutions and for the forms of sulfur in the treated materials. About 15% oxidation in sulfide-sulfur was realized for pure pyrite (with initial sulfur content of about 54%) and 57% oxidation for flotation concentrate (with initial sulfur content of about 14.7%). These initial results are promising and provide a good platform for further research.

10.40 Coffee

11.20 **Bio-reduction of sulfides investigated as pre-treatment for refractory ores**

A. Hol, R.D. van der Weijden, G. Van Weert, P. Kondos and C.J.N. Buisman (Wageningen University, The Netherlands)

To liberate gold, refractory sulfides are commonly oxidized via methods that require a lot of energy and generate acids that need to be neutralized and disposed of in an environmentally acceptable way. An alternative would be to liberate gold via bio-reduction of sulfide minerals. Chemical reduction of pyrite is an already described reaction, proceeding only at elevated temperatures: $FeS_2 + (1\text{-}x)H_2 \rightarrow FeS_{(1+x)} + (1\text{-}x)H_2S$. If it is possible to let this reaction proceed with microorganisms at ambient temperatures, bio-reduction would be a method with a lower energy demand and environmental impact. Furthermore, generated H_2S can be re-used to produce elemental sulfur or gold lixiviants.

The authors will present results obtained for bio-reduction of pyrite and arsenopyrite in gas lift loop reactors under a variety of conditions.

11.40 **Comparison of kinetic models for gold leaching and cyanide consumption in gold cyanidation of refractory gold concentrate**

W. Srithammavut, I. Turunen, A. Laari (Lappeenranta University of Technology, Finland), S. Luukkanen (Geological Survey of Finland, Finland) and T. Kankaanpää (Outotec Research Oy, Finland)

Gold cyanidation of refractory gold concentrate was studied experimentally and by mathematical modelling. Experiments were carried out in stirred reactors at different conditions varying oxygen and cyanide concentrations. Samples taken from the reactor were analyzed for gold and other minerals using fire assay AAS and ICP methods. Based on the experimental results, kinetic models were developed for gold leaching and cyanide consumption. The effects of dissolved oxygen and free cyanide concentrations, amount of leachable gold in the concentrate and solid-liquid mass transfer were taken into account. Kinetic parameters were estimated by fitting the model parameters to the experimental data. The rate controlling step was determined by comparing the surface reaction and mass transfer limited models. The obtained results show good agreement between the measured and predicted concentrations. The reliability of the estimated parameters was studied using Markov chain Monte Carlo methods. All the experimental parameters were found to be well-identified.

12.00 **Process plant design considerations for high silver gold ores**

D. Yan and D. Connelly (Mineral Engineering Technical Services Pty Ltd, Australia)

Merrill Crowe preceded Carbon In Pulp technology and is now only used for very high grade gold projects or where the silver to gold ratio exceeds ten to one. CIP technology is generally superior with respect to operating cost and efficiency for lower grade ores whereas Merrill Crowe is superior for high grade ores.

Gravity processes are not as efficient for high silver ores as for gold. Metallic silver or electrum can be recovered at modest recoveries whereas acanthite (Ag_2S) exhibits poor gravity gold recovery even with centrifugal concentrators.
Leaching of high silver ores requires significantly higher cyanide (>1500 ppm) and oxygen levels and extended leach times (>72 hrs) and usually the recovery is lower than gold.

The adsorption capacity needs to be higher with a larger inventory of carbon and more carbon movement. Carbon activity and solution loss become more critical and a tendency for more stages to be required in order to achieve equilibrium.

Silver elutes first off carbon and at a lower temperature. These conditions are at odds with the gold elution and what generally happens is that operations optimise to maximise gold recovery from the carbon. The silver cyanide complex ($AgCN_2$) is less stable than gold cyanide ($AuCN_2$) and breaks down at the lower temperature and forms metallic silver remaining on the carbon. This has a negative impact on carbon activity and hence solution loss.
Where high silver values are present during electrowinning the cathodes can be washed with high pressure water and the electro won gold and silver comes off very easily. In addition high sludging of the cell is not uncommon as the dendrites are loosely held. Some designs exploit this feature-self sludging cells.

The gold elect potential is -0.63 volts and for silver it is -0.45 volts. This means the silver will preferentially plate out of solution before the gold slowing the electrowinning process. In addition there is more metal to electro win and more cell capacity is required.

Conventional processing of silver-bearing materials containing gold group metals involves smelting the material to bullion and purification of the silver by electro refining, along with recovery of gold and platinum group metals from the slimes generated by the electro refining process. An alternative process has been developed that allows extraction of the silver from such residues using a combination of chloride and nitrate chemistry. An advantages of the process is the production of very high-purity silver. A number of examples are cited and the good design aspects highlighted.

12.20 **Gold and silver separation from thiosulfate leaching solutions in a rectangular open vessel**
A.R. Alonso **(**Universidad Nacional Autónoma de México, Mexico) and G.T. Lapidus (Universidad Autónoma Metropolitana Iztapalapa, Mexico)

The use of thiosulfate solutions for silver and gold leaching has been widely studied, mainly because of its low toxicity and the recycling feasibility. In order to obtain the gold and silver in its metallic state from these solutions, some processes use solvent extraction or cementation on zinc and copper. However, both of these methods introduce contaminants into the thiosulfate solutions, diminishing its capacity to be recycled. On the other hand, direct electrolysis adds no new contaminants to solution; additionally, it is possible to obtain the precious metals in different forms, such as nano-sized particles. In the present work, a single rectangular open vessel was used as an electrochemical reactor and the deposition behavior was studied, with and without solution stirring, using a titanium plate as cathode and copper, titanium or boron doped diamond as anode, to minimize the thiosulfate oxidation.

12.40 Lunch

14.00 *Technical Session 4*
Chairmen: N.O. Lotter (Xstrata Process Support, Canada) and M.D. Adams (Mutis Liber, Australia)

14.00 **Oxidation-reduction process for refining of silver-copper alloys**
C. Harris, C. Pickles and J. Peacey (Queen's University, Canada)

The high temperature oxygen refining of molten silver-copper alloys is typically associated with relatively high silver losses to the slag. In order to better understand this process a thermodynamic model has been developed to elucidate the silver loss mechanism. Silver oxide has a strong affinity for copper oxide and this accounts for the increase in silver loss with refining time. A series of equilibrium experiments were performed to investigate the silver loss mechanism. As predicted by the model, the equilibrium results demonstrate that by maintaining a low copper oxide content in the slag, the silver losses can be minimised. Then oxygen refining tests were performed under a borosilicate slag in order to further investigate the silver losses to the slag. A proposed process for the oxygen refining of silver copper alloys using both oxidation and reduction stages is discussed.

14.20 **A review of the selective leaching of gold from oxidised copper-gold ores with ammonia-cyanide and new insights for plant control and operation**
D.M. Muir (CSIRO Process Science and Engineering (Minerals)/Parker Centre for Integrated Hydrometallurgical Solutions, Australia)

The ammonia-cyanide leach system was first patented over 100 years ago and stands out as a unique method of selectively leaching up to 90% Au and <1% Cu from oxidised copper-gold ores using <10% NaCN used in conventional cyanidation processes. However, the system has proved to be difficult to control and predict performance with different ores due to a lack of understanding of the chemistry and mechanism and proper process control. Several laboratory studies have been carried out using various empirical concentrations of ammonia and cyanide with mixed success and only a few commercial operations have been successful.

This paper reviews some of the recent applied and fundamental studies on the leaching of copper-gold ores with the ammonia-cyanide system and provides insights into the mechanism to give a better appreciation of the key parameters required for the optimum leaching of gold with minimum copper dissolution. Recommended leach compositions, Eh and pH are provided to enable process control measures to be adopted for a variety of ores. The selective recovery of gold from the leach solutions by cementation or adsorption onto activated carbon or ion-exchange resins is also discussed.

14.40 **Industrial plant for gold extraction by ammonia-cyanide leaching process from sulphide ore**
M.K. Razi, A.S. Mohammadi (Islamic Azad University, Iran) and M.Mohammadi (Mouthe Gold Complex, Iran)

The operation of gold extraction from sulphide ore of Senjedeh mine which includes high grade of copper in ammonia- cyanide system was studied in industrial plant. Entrance feed involved pyrite and chalcopyrite behaves as the host ore of gold.

The average of copper grade is 1700 ppm and 2 ppm for gold, the average of mesh size is 63µ for<64%. The residence time for leaching and aeration is 48 hrs. Based on experimental findings, the consumption of cyanide, ammonia (18%) and lime on the average is achieved 1.2kg/t, 8 L/t and 4 kg/t respectively.

The efficiency is obtained 85%. This plant has 30 t/hr tonnages as nominal.

15.00 **Phytoextraction of copper and gold from mine tailings**
Victor Wilson-Corral et al (Centro de Estudios Justo Sierra. Mexico), M. Arenas-Vargas (Universidad Autónoma Metropolitana-Unidad Xochimilco, Mexico) and C. Anderson (Massey University, New Zealand)

To examine the feasibility of gold and cooper phytoextration technology laboratory and field scale experiments were carried out using mine tailings in Sinaloa, Mexico. Local available plant species *Helianthus annus* (sunflower) and *Kalanchoe serrata* (magic tower) were used. The results shows that copper concentrations was above 4 mg/kg (dry mater) in *K. serrata* tissues and in *H. annus* specie, copper average concentrations were: 118 mg/kg in roots, 141 mg/kg in stem and, 119 mg/kg of dry mater of leaves. In addition, *K. serrata* specie plant species showed gold concentrations above 9 mg/kg of dry mater where as in *H.annus* specie gold average concentrations were 15 mg/kg in leaves, 16 mg/kg in roots and, 21 mg/kg in stem. This data confirm that phytoextraction technologies can be used to recover precious metals from mine tailings other wise a waste product.

15.20 Coffee

16.00 **SART / AVR circuit design and operation at Yanacocha Gold Mill**
G. Guzman, V. Mamani, H. Arevalo, S. Vicuña, L. Vargas and Ben Burger (Minera Yanacocha, Newmont Mining, Peru)

Commercial application of the SART process on cyanide leach solutions from the treatment of Au-Ag-Cu whole ore has been successfully demonstrated at Minera Yanacocha's Gold Mill facility in northern Peru. The SART circuit allows commercial exploitation of transitional ores containing higher copper and silver values than carbon-in-leach circuits can normally tolerate and negates cyanide detoxification requirements for cyanide tailings solutions. This paper discusses the process development to suit the variable Yanacocha ore types, solution chemistry of cyanide leaching, SART and AVR (cyanide recovery), as well as operational experiences and lessons learned.

16.20 **The ICMI Cyanide Management Code- a model for other compliance issues?**
P. Lotz (Mintek, South Africa)

The last two decades have seen the gold industry, amongst others, subscribe to a plethora of codes and practices over and above the applicable local government regulations. Global and generic codes such as ISO 14001 can sometimes lead to selective aspect focusing and hence by-passing important objectives. A system of tightly specified target objectives, whilst leaving room for practical implementation measures was chosen for the ICMI (International Cyanide Management Institute) cyanide management code. Since its roll out, experience indicated that the linkage of industry specific engineering requirements to vigorous auditing and verification are better suited to risk limitation than the often hazy stipulations contained in generic standards.

Furthermore, the fact that ICMI was based on peer industry consensus ensures a higher degree of "buy-in" into what is regarded as gold industry best practice.

During the past years it could be demonstrated, that the technical requirements not only served to limit cyanide incidents, but often led to substantial savings as it forced a measure of plant stability and continuity benefiting the entire process.

Experiences supporting this claim resulting from ICMI code implementation and auditing will be presented.

16.40 **The Lantern Retort – a new retort for small-scale gold extraction**

R.K. Amankwah (University of Mines and Technology, Ghana) and G. Ofori-Sarpong (Pennsylvania State University, USA)

The Thermex glass retort was introduced by UNIDO to help reduce mercury emissions in small-scale gold mining operations. Unfortunately it is not patronized by small-scale miners due to the extended heating periods resulting from the low thermal conductivity of glass. Steel retorts are also not popular because it is reported that there is darkening of gold after retorting and its opaque nature prevents miners from observing the process. This paper proposes a new retort, the Lantern retort, which combines the high conductivity of steel with the transparency of glass to improve upon retorting efficiency and visual examination of the process. The Lantern retort has a shorter warm-up and amalgam distillation time. The rate of heating in the Lantern retort is 1.81^{o}C/sec while that of the Thermex is 1.49^{o}C/sec. Other advantages of the lantern retort are that it is cheaper and can withstand the harsh terrain of small-scale gold mining.

17.00 Closing remarks

M.D. Adams (Mutis Liber, Australia) and A.J. Wills (MEI, UK)