MONDAY, AUGUST 22, 2005, P.M.

SESSION 4: INTERNATIONAL SYMPOSIUM ON THE TREATMENT OF GOLD ORES

ORE CHARACTERIZATION AND RECOVERY OF GOLD BY GRAVITY
Sponsor(s): Hydrometallurgy Section, The Metallurgical Society of CIM, Canadian Mineral Processors Division of CIM, CANMET-MMSL
Room: Imperial Ballroom 3
Chair(s): J. ABOLS, Canadian Operations, Gekko Systems, Canada and A.R. LAPLANTE, McGill University, Canada

14:00 — OPENING REMARKS
G. DESCHÈNES, CANMET, Canada

PAPER 4.1—14:05 (KEY NOTE)

Recovery Creep in Mine Development Programmes and Barrick’s Response with Associated Mitigating Measures
P. KINVER, Barrick Gold Corporation, Canada
Abstract not available.

PAPER 4.2—14:45
RECOVERY OF FINE GOLD PARTICLES USING A FALCON ‘B’ SEPARATOR.
B. GEE, P. HOLTHAM, Julius Kruttschnitt Mineral Research Centre, Australia
R. DUNNE and S. GREGORY Newmont Australia Ltd., Australia

Many gold treatment plants have sulphides and coarse particle gangue components that contain gold in their tailings (cyanide leach residues and flotation tailings). Coarse and very fine liberated gold may also be present in some tailings streams. Two issues usually govern the economics of recovering additional gold from gold occluded in sulphide and gangue particles. The liberation characteristics of the gold in the sulphide and coarse gangue (gold liberation versus grind size) is the first and most important of the two. If fine grinding can liberate a reasonable amount of the gold, then the next important issue is how effectively these two constituents can be concentrated. Flotation will achieve high sulphide recovery but is usually costly. Selectivity is also an issue because of entrainment. Continuous centrifugal gravity concentration offers a viable solution to the recovery of both sulphides and coarse gangue however selectivity is also likely to be a problem. The possibility of improving concentrate grade by using a Falcon B separator was investigated. This paper describes a program of test work carried out on a gold leach tailings using a laboratory Falcon ‘B’ separator. The program involved three different bowl angles: 10, 12 and 14 degrees and two bowl speeds (giving 250 and 350G accelerations). A number of tests were also performed on samples that had free gold present to ascertain the size-by-size recovery of gold in these samples.

COFFEE BREAK—15:10-15:25

PAPER 4.3—15:25
GOLD CHARACTERIZATION OF A SAMPLE FROM MALARTIC EAST (QUÉBEC) USING CONCENTRATION BY HYDROSEPARATOR.
R. LASTRA, J. PRICE, Natural Resources Canada, Canada
L.J. CABRI, Cabri Consulting Inc., Canada
N.S. RUDASHEVSKY, V.N. RUDASHEVSKY, Center for New Technologies, and G. MCMAHON, Harvard Medical School & Brigham and Women’s Hospital, U.S.A.

A gold sample assaying 11.4 g Au/t was selected for this study. Approximately 99% of the gold in this sample occurs as discrete gold minerals; the balance occurring as invisible gold in pyrite (determined by SIMS analysis). The sample was coarsely ground and sieved, yielding ~38, 17, 22 and 22 wt.% in the -400+160 µm, -160+80 µm, -80+40 µm and -40 µm size fractions. Assays of the size fractions gave a gold distribution of ~32, 20, 21, and 27% respectively. Processing with the hydroseparator (model HS-02) gave two tailings products and one concentrate from each size fraction. Assays of the concentrate from each size fraction gave gold recoveries of ~7, 7, 24, 19%, respectively. Polished sections were prepared from each hydroseparation product and studied by image analysis to automatically search for gold minerals. Calculations were done using the surface area of the grains of gold minerals.
found on the polished sections. Using only information from the concentrates, it was determined that ~78% of the gold is free, compared to ~66% free gold in the combined tailings plus concentrate. The calculated distribution of the gold minerals was ~79% electrum, 11% gold tellurides, and ~10% native gold in the concentrates comparing closely with ~79% electrum, ~13% gold tellurides, and ~8% native gold in the combined tailings plus concentrate. The calculated grain size distribution of the gold minerals showed that in the concentrates ~78% of the gold is between 26 and 74 µm compared to ~61% of the gold between 26 and 74 µm for the combined tailings plus concentrates. Thus, the characterization of the gold minerals derived from the concentrates is similar to the one derived for the reconstructed sample. The total area of gold grains found in the four polished sections of the concentrates is ~22,219 µm², which is 10 times that found in 10 polished sections of un-ground and unprocessed sample. These results clearly indicate the benefits of pre-concentration for gold characterization. Though the tests showed that hydroseparation did not concentrate all the gold, it yielded rich concentrates representative of the ore, which does simplify the mineralogical gold characterization.

PAPER 4.4—15:50
MAXIMIZING GRAVITY RECOVERY THROUGH THE APPLICATION OF MULTIPLE GRAVITY DEVICES.
J. ABOLS and P.M. GRADY, Canadian Operations, Gekko Systems, Canada
Recovery via gravity is one of the oldest mineral processing methods available. Unfortunately, the use of gravity techniques for gold recovery has been in decline for the past century as more effective chemical processes such as flotation and leach/CIP have been developed. Recently, with the push towards more sustainable environmental outcomes, the benefits of gravity separation have become more apparent. While gravity as a unit process is not usually capable of achieving as high a recovery as flotation or whole ore cyanidation, a combination of gravity devices alone or in conjunction with these processes can offer significant advantages to the operator.

This paper reviews the range of gravity devices available, their application and the results that can be achieved by maximizing gravity through the use of a combination of gravity recovery devices. Case studies are provided.

PAPER 4.5—16:15
GRAVITY RECOVERY OF GOLD – AN OVERVIEW OF RECENT DEVELOPMENTS
A.R. LAPLANTE, McGill University, Canada and
W. STAUNTON, Murdoch University, Australia
Gold recovery early in the flowsheet, typically from the primary grinding circuit, has seen significant advances in the past three years. Some of the advances achieved under the auspices of the AMIRA P420B Gold Processing Technology are reviewed. The areas covered are gravity-recoverable gold (GRG) characterization, GRG behaviour in classification, gravity and flash flotation simulation and circuit optimization. Examples of improvements and their impact on overall recovery are presented. Also discussed is how circuit design can benefit from these advances and where future research avenues lie.

PAPER 4.6—16:40
CONTINUOUS GRAVITY IMPROVEMENTS AT THE PORCUPINE JOINT VENTURE DOME MINE’S GRAVITY CIRCUIT.
J.A. FOLINSBEE, T.Y CHONG, PJV, Canada and
M. FULLAM. The Knelson Group, Canada
PJV Dome Mine has used some form of gravity concentration since start of milling operation in 1909 to recover free milling gold. Amalgamation and conventional jigs were first used and were replaced with the Knelson Concentrators in 1993. The Knelson concentrates produced were upgraded on a conventional shaking table prior to smelting and refining into gold bullion. In May 2002, a comprehensive review of the latest gravity recovery technology led to a piloting campaign aimed at further improving gravity recovery. The Knelson concentrates, subjected to intensive cyanide leaching in an Acacia pilot plant, were found to be amenable to intensive cyanide leaching with over 98% of the gold being dissolved within 24 hours. An Acacia Intensive Leach Reactor was successfully installed and commissioned in December 2002, to replace the shaking table. his paper describes pilot scale testing, installation, commissioning and optimization of the Acacia Intensive Leach Reactor. Also, the economic impacts on down stream processes are highlighted.

TUESDAY, AUGUST 23, 2005, A.M.

SESSION 13: INTERNATIONAL SYMPOSIUM ON THE TREATMENT OF GOLD ORES
ORE CHARACTERIZATION AND RECOVERY OF GOLD BY GRAVITY II
Sponsor(s): Hydrometallurgy Section, The Metallurgical Society of CIM, Canadian Mineral Processors Division of CIM, CANMET-MMSL
The effect of carbonaceous matter on gold extraction was investigated in cyanide systems with pure gold as well as pre-robbing sulfide gold ores. Auger studies demonstrated that the carbonaceous matter preferentially smeared on iron sulfide rather than aluminosilicate surfaces during wet or dry grinding. During mechanically mixed leaching, coating and stripping of carbonaceous matter occurred simultaneously. TEM and XPS studies indicated preferential coating in the form of elemental (graphitic) carbon at the edges and the defect sites of particles. Interestingly, carbonaceous matter extracted from the gold ore was amorphous carbon and not preg-robbing itself. However, the carbonaceous coating had a significantly detrimental effect on gold dissolution when gold was pre-ground with the gold ore. Artificial coating on gold surfaces with the natural carbonaceous matter from the ore was also found to substantially retard gold dissolution. With the addition of the natural carbonaceous matter from the gold ore, gold extractions from two non preg-robbing sulfide gold ores were inhibited owing to carbonaceous coating. The presence of carbonaceous matter significantly reduced the current density for gold oxidation and increased the current density for pyrite oxidation. This could explain the fact that carbonaceous coating hindered gold dissolution while enhanced sulfide preg-robbing.

NEW LEACHING TECHNOLOGIES
PAPER 13.4—9:45
THIOSULPHATE DECOMPOSITION IN THE PRESENCE OF SULPHIDES.
D. FENG and S.J. VAN DEVENTER, University of Melbourne, Australia

The mechanism of thiosulfate decomposition and the products formed were investigated in the presence of pyrite or pyrrhotite in copper-ammonia solutions used for leaching gold. The presence of the sulfides significantly increased the decomposition of thiosulfate. The sulfide-surface-catalyzed oxidation of thiosulfate to tetrathionate by dissolved oxygen or the cupric tetra-ammine complex was proposed to be the dominant thiosulfate decomposition mechanism. The catalysis of the sulfides in this reaction could originate from their strong affinity for aqueous sulfur species and their semiconducting properties. Oxygen was found to be the initial driving force for thiosulfate decomposition in the absence of cupric ions and the driving force for the continuous decomposition of thiosulfate in the presence of cupric ions. Under an oxygen free atmosphere, the decomposition of thiosulfate was limited to reduce the cupric ions and the catalytic effect of the sulfides was also marginal. The dominant oxidative product of thiosulfate was trithionate in alkaline solutions especially in an oxygen rich or cupric ion rich environment. Tetrathionate was found in relatively smaller amounts in an oxygen and cupric ion deficient environment. Tetrathionate was not stable and would further decompose after extended periods. Trithionate concentrations increased almost linearly with time.

COFFEE BREAK—10:10-10:40

PAPER 13.5—10:40
FUNDAMENTALS AND APPLICATIONS OF ALKALINE SULFIDE LEACHING AND RECOVERY OF GOLD.

The latter part of the 20th century saw great advances in the treatment of refractory gold ores coupled with increased reliance on the use of cyanide for gold processing. Now, in many parts of the world, there is social pressure to limit or eliminate the use of cyanide. As well, treatment of some refractory ores or concentrates which have excessive cyanide consumption, gold cyanide pregrubbing or significant sulfide content remain difficult. This paper will outline the history of the development of alkaline sulfide leaching as an ancillary process to nitrogen species catalyzed (NSC) pressure leaching. Electrochemical fundamentals and the applicable thermodynamics of the alkaline sulfide hydrometallurgical system will be outlined. As well, examples of refractory gold recovery with alkaline sulfide hydrometallurgy such as an arsenopyrite gold concentrate application, a chalcopyrite gold concentrate application, a pregrubbing gold ore application, and a cyanide consuming gold ore application will be delineated in this paper.

PAPER 13.6—11:05
CHLORIDE LEACHING OF GOLD FROM SULFIDE CONCENTRATES.
J. LEPPINEN, M. HÄMÄLÄINEN and O. HYVÄRINEN, Outokumpu Research, Finland

Outokumpu has developed a new chloride leaching process called HydroCopper®. In this process copper sulfide concentrates are leached at atmospheric pressure in concentrated sodium chloride solution using cupric ions as the oxidant. Copper is recovered from the purified leach solution by precipitating cuprous oxide which is then reduced to pure copper and further melted and cast directly to copper products. The key reactants sodium hydroxide, hydrogen and chlorine are generated from the spent sodium chloride using chlor-alkali technology. An important feature in the HydroCopper® process is that the gold contained by copper concentrates is leached as chloride complexes and recovered from solution by activated carbon. Conditions for the leaching of gold-bearing pyrite can also be achieved in the chloride system by optimizing the redox conditions. This paper deals with basic factors affecting gold leaching from copper sulfide concentrates. Opportunities for the recovery of refractory gold associated with arsenopyrite and pyrite will also be discussed.

PAPER 13.7—11:30
THE INTEC GOLD PROCESS - A HALIDE-BASED ALTERNATIVE FOR THE RECOVERY OF GOLD FROM REFRATORY SULFIDE DEPOSITS.
J. MOYES, F. HOULLIS, J-L. HUENS, University of Sydney, Australia and D. SAMMUT, Consultant, Canada

The Intec Gold Process has been developed as a halide-based alternative for the recovery of gold from refractory sulfide deposits. The halide medium allows sulfide oxidation and gold extraction to be performed concurrently at moderate temperature and atmospheric pressure sourcing oxygen from direct air injection. The dissolved gold is loaded onto activated carbon and stripped in a conventional Zadra circuit. High gold extractions have been achieved from a range of refractory gold concentrates at laboratory scale, which has led to a continuous locked-cycle pilot plant program. The pilot plant operated at >99% availability, with a maximum of 96.5% gold extraction from a concentrate.
containing 58.6 g/t gold. >99% of the dissolved gold was loaded onto carbon at up to 1% w/w, with no loss of carbon activity detected over five loading/washing/elution cycles. This paper describes the process and presents both laboratory and pilot plant results as well as economic data from two costing studies.

TUESDAY, AUGUST 23, 2005, P.M.

SESSION 24: INTERNATIONAL SYMPOSIUM ON THE TREATMENT OF GOLD ORES

NEW LEACHING TECHNOLOGIES II
Sponsor(s): Hydrometallurgy Section, The Metallurgical Society of CIM, Canadian Mineral Processors Division of CIM, CANMET-MMSL
Room: Imperial Ballroom 3
Chair(s): J. MCMULLEN, Barrick Gold Corporation, and W. STAUNTON, Murdoch University

PAPER 24.1—14:00
THE KINETICS OF THE COPPER-CATALYZED OXIDATION OF THIOSULFATE IN AMMONIACAL SOLUTIONS.
C. VAN WENSVEEN and M. J NICOL, Murdoch University, Australia

The kinetics of the oxidation of thiosulfate by copper(II) in ammoniacal solutions have been studied in the presence and absence of dissolved oxygen. The formation of an intermediate mixed ammine/thiosulfate copper(II) complex ion has been confirmed and the rate of the reaction monitored by simultaneous on-line spectrophotometric, dissolved oxygen and potentiometric measurements. The rate of the anaerobic oxidation of thiosulfate by copper(II) was found to be orders of magnitude slower than that in the presence of dissolved oxygen and is inhibited in the presence of ammonia and anions such as sulfate and chloride as a result of competition with thiosulfate for the axial coordination sites on the copper(II) tetrammine complex. In the presence of dissolved oxygen, it was found that the copper(I) thiosulfate complexes are only very slowly oxidized while the copper(I) diammine complex is rapidly reoxidized to the copper(II) state. Thus, the initial rapid reaction with copper(II) leads to a pseudo steady-state condition in which the concentration of copper(I) is relatively constant and the rate of oxidation of thiosulfate can be predicted from the measured rate of oxidation of copper(I) by oxygen in ammoniacal solutions. The effects of the concentrations of ammonia and thiosulfate on the oxidation rate can be rationalized on the basis of the above coupled to the species distribution of copper(I) in the ammonia/thiosulfate system. It is shown that, with relatively simple measurements of dissolved oxygen and copper concentrations, the rate of oxidation of thiosulfate can be predicted under typical leaching conditions.

PAPER 24.2—14:25
A NOVEL THIOSULFATE LEACH PROCESS FOR THE TREATMENT OF CARBONACEOUS GOLD ORES.
P.G. WEST-SELLS, Placer Dome Research Centre, Canada and R.P. HACKL, Placer Dome Technical Services Limited, Canada

Carbonaceous gold ores are particularly attractive for leaching by ammonium thiosulfate as the gold thiosulfate complex is not significantly adsorbed by the organic carbon present in the ore. A technically and economically feasible process for treating these ores with ammonium thiosulfate has been developed. In this process, gold is extracted by leaching the ore with ammonium thiosulfate, the solution is separated from the ore by counter – current decantation, and the gold is recovered by precipitation with ammonium sulfide. The precipitation filtrate is recycled back to the grinding step to close the solution balance. Reagent consumption is not excessive. Besides precipitating gold, the precipitation step removes most deleterious solution impurities and converts a portion of the polythionates back to thiosulfate. This paper discusses the optimization of the leaching and precipitation steps, and the influence of grinding and solution recycle on leaching. The results of a 10 kg/h pilot plant campaign are also discussed.

PAPER 24.3—14:50
LEACHING OF A GOLD ORE USING THE HYDROGEN SULFIDE-BISULFIDE-SULFUR SYSTEM.
B. WASSINK, D. DREISINGER, University of British Columbia, Canada, P. WEST-SELLS, Placer Dome Research Centre, Canada and N. FISHER, Independent Metallurgical Laboratories Pty Ltd., Australia

Gold leaching from a commercial cyanide-amenable gold ore was studied at 25°C using sodium bisulfide-hydrogen sulfide solutions containing elemental sulfur. Gold extraction from the ore using 0.55 M NaSH, 4.1 atm (60 PSI) H2S, 1 g/L S, and 26% solids by weight was about as good as the best extraction obtained by cyanidation. Maximum extraction was achieved in roughly 48 hours. Sulfur dissolves in the bisulfide solutions to form a mixture of polysulfides. Simple speciation calculations indicate that gold solubility as bisulfide complexes should be roughly ten
times higher than in the absence of added zerovalent sulfur. The principal bisulfide complex is believed to be [Au(HS)$_2$]$. Gold dissolution is favoured by increasing HS$^-$ concentration and by increasing H$_2$S pressure. Polysulfide complexes of gold are also likely to be formed, and may contribute substantially to gold leaching. Two experiments using pure gold were also performed. Up to 100 mg/L gold was dissolved in 140 hours using 0.55 M NaSH, 1 g/L added elemental sulfur and 6.2 atm H$_2$S.

COFFEE BREAK—15:20-15:30

PAPER 24.4—15:30
AN ELECTROCHEMICAL STUDY OF AN ALTERNATIVE PROCESS FOR THE LEACHING OF GOLD IN THIOSULFATE SOLUTIONS
H. ZHANG, M.J. NICOL and W.P. STAUNTON, Murdoch University, Australia
The leaching of gold with thiosulfate solutions in the absence of ammonia and copper ions was investigated using electrochemical and kinetic methods. In sodium thiosulfate solutions, the electrochemical oxidation of gold was hindered and the leaching rate using pure oxygen as the oxidant was very slow. It was found that the addition of a small amount of thiourea or formamidine disulfide to the thiosulfate solution greatly enhanced the oxidation of gold. However, the enhancement to the overall dissolution of gold using dissolved oxygen as the oxidant was limited. This was attributed to the fact that the oxygen reduction process was unaffected by the additives and remained slow, which limited further improvement to the overall kinetics. An alternative oxidant, the ferric EDTA complex having a reduction potential of approximately 0.12 V, was found to be promising. The cathodic reduction of ferric EDTA was fast and reversible at neutral pH. While the ferric EDTA complex was effective in oxidizing gold, it did not appear to significantly oxidize thiosulfate. A thiosulfate solution containing thiourea and ferric EDTA has been shown to quickly dissolve gold in the form of a powder as well as from gold ores. In the leaching with the ores high gold recoveries of above 90% was achieved with the best performance obtained in neutral solutions (pH 6-7).

PAPER 24.5—16:55
IRON SULFIDE MINERALS IN THIOSULFATE-GOLD LEACHING PROBLEMS AND SOLUTIONS.
C. XIA and W.T. YEN, Queen’s University, Canada
The negative effects of pyrite and pyrrhotite on thiosulfate gold leaching have been investigated. Possible remedies to reduce some problems are discussed. In a gold bearing pure silicate ore, 92% of gold could be extracted within 3 hours using a solution containing 0.2 mol/L ammonium thiosulfate, 0.9 mol/L ammonia, 30 ~ 300 ppm copper sulfate and 8.6 ppm dissolved oxygen at pH 10.26. The thiosulfate consumption was 7.1 kg/t under these conditions and could be further reduced to 3.5 kg/t by decreasing the dissolved oxygen to 0.3 ppm. Addition of 16% pyrrhotite or pyrite into the silicate slurry led to a decrease in gold extraction by 2% or 9% respectively and an increase of thiosulfate consumption by 11 kg/t or12.8 kg/t respectively in three hours. Several adjustments, such as increasing copper ion and ammonia concentration and reducing the dissolved oxygen level to 0.3 ppm resulted in increased gold extraction and lower thiosulfate consumption despite the presence of these sulfides.

PAPER 24.6—16:20
LEACHING AND RECOVERY OF COPPER DURING THE CYANIDATION OF COPPER CONTAINING GOLD ORES
P.L. BREUER, CSIRO Minerals, Australia
M.I. JEFFREY and X. DAI, Monash University, Australia
In the leaching and recovery of gold from copper containing gold ores using cyanidation, some copper minerals also dissolve which can have a detrimental effect on the process efficiency and economics. There are also environmental concerns with current operations discharging cyanide and copper cyanide species into tailings dams, and strict regulations prohibit this for greenfield plants. Cyanide destruction however is not an economic option for new plants treating copper-gold ores. This paper evaluates a potential recovery process to overcome the copper cyanide problem. Despite gold dissolution occurring in copper cyanide solutions having no free cyanide, gold recovery from ores is lower compared to free cyanide. Thus, cyanide addition has to be optimized to maximise the gold recovery whilst endeavouring to eliminate free cyanide at the completion of leaching (CN$^-$/Cu less than 3:1). Subsequently, copper cyanide can be recovered using activated carbon. The loaded copper cyanide is easily recovered from the carbon with a cyanide soak and distilled water wash such that the eluted CN$^-$/Cu ratio is less than 4:1. By decreasing the pH of the eluant to 5 the copper could be recovered efficiently by electrowinning. The remaining cyanide solution, after electrowinning, is then neutralisation to form NaCN and recycled.

PAPER 24.7—16:45
SOLUTION CHEMISTRY OF TRITHIONATE WITH RELEVANCE TO GOLD LEACHING BY THIOSULFATE: A REVIEW.
N. AHERN, D. DREISINGER, University of British Columbia, Canada and
In the thiosulfate leaching of gold, degradation of the thiosulfate reagent is a major cost for the process. One of the major intermediary degradation products is trithionate, which also inhibits gold recovery from solution by ion exchange. Little attention has been given to this species in the context of gold leaching - the limited information on trithionate and its solution chemistry available in the literature is seldom directly related to gold leaching conditions. This review is a compilation of the published knowledge on trithionate, with a specific focus on identifying the relevance of this information to gold leaching and the shortcomings in this knowledge base.

WEDNESDAY, AUGUST 24, 2005, A.M.

SESSION 35: INTERNATIONAL SYMPOSIUM ON THE TREATMENT OF GOLD ORES

PLANT PRACTICE AND PROJECT DEVELOPMENT

Sponsor(s): Hydrometallurgy Section, The Metallurgical Society of CIM, Canadian Mineral Processors Division of CIM, CANMET-MMSL
Room: Imperial Ballroom 3
Chair(s): J.S.J. VAN DEVENTER, The University of Melbourne, and D. HODOUIN, Laval University, Quebec

PAPER 35.1 — 8:30
INSTITUTIONALIZING ADVANCED CONTROL ON NEWMONT’S CARLIN TREND.
A. COLLINS and D. DANNINGER, Newmont Mining Corporation, U.S.A.

Newmont has installed advanced control and online optimization on the dry grinding circuit and both roasters on its Carlin Trend refractory ore treatment plant. Subsequently, Newmont has developed a master plan for institutionalizing advanced control and optimization on all its processing units. Newmont is committed to a program of applying advanced control and optimization from crushing of ore to pouring of gold. This paper describes Newmont’s experience with implementing advanced control and optimization and the benefits received. In addition, identification of advanced control and optimization opportunities and their economic justification is discussed.

PAPER 35.2 — 8:55
BARRICK GOLD CORPORATION — LAGUNAS NORTE & VELADERO PROJECTS.
S. HAGGARTY, J. MCMULLEN, Barrick Gold Corporation, Canada and R. WALTON, SNC-Lavalin, Canada

Barrick’s growth and development plan will add three major new mines during 2005 including the Lagunas Norte Project in the La Libertad Province of Peru, the Veladero Project in the San Juan province of Argentina, and the Tulawaka Mine in north-western Tanzania. Lagunas Norte and Veladero are similar in that they are both green-field open pit deposits at elevations over 4,000 meters above sea level, with two stage crushing and heap leaching followed by Merrill Crowe zinc precipitation for precious metal recovery. Operating sites will each generate in excess of 700,000 ozs Au per annum with commissioning and full scale operation planned during the third and fourth quarter of 2005. Details with regards to regional geography, development schedule, production and process specifics for Lagunas Norte and Veladero will be discussed in this presentation providing an overview for the respective properties.

PAPER 35.3 — 9:20
STUDY OF THE KINETICS OF LEACHING GOLD FROM A SOFT SULPHIDE ORE AT THE SADIOLA MINE.
G. DESCHÊNES, Natural Resources Canada, Canada and T. MULPETER, SEMOS, Mali

Sadiola Gold Mine is an open pit operation located in Mali and owned by AngloGold Ashanti Ltd, Iamgold Corporation and the State of Mali. A conventional cyanidation process (cyanidation/CIP) treats 15,000 tpd (of a mixture of oxide and sulphide ores) at an average grade of 3 g/t Au. In 2002, the increasing tonnage of sulphides caused an increase in production costs and a decrease in gold extraction (from 95% to 70-76%). Because of the short retention time of the leach circuit (18 hours) and the refractory nature of the sulphides, gold leaching kinetics are critical. A laboratory investigation was initiated to evaluate actual plant practice and identify a means of improving the gold leaching rate. The study was conducted on a “soft” sulphide gold sample containing 4.96 g/t Au, 0.4 g/t Ag, 2.5% pyrite, 1.2% arsenopyrite, and 0.2% pyrrhotite with a P80 of 74microns. Pre-leaching significantly influenced the gold leaching rate. A short pre-leach (2 hours) performed at pH 8, a DO of 8 ppm, and with 300 g/t lead nitrate added at the start or the end, produced the preferred conditions. Pre-leaching for more than 2 hours was detrimental to gold extraction. The cyanide concentration had to be maintained at 450 ppm NaCN to reach a maximum gold extraction.
within the retention time of the circuit. A maximum extraction of 81.3% Au was obtained in 20 hours (leach residue at 0.97 g/t Au). A high level of dissolved oxygen in the pre-leach was not of any benefit. Extension of gold leaching beyond 20 hours indicated that all leachable gold had been extracted by that time. Plant modifications were made to allow leaching of oxides and sulphides in separate circuits. Treatment of oxide ores is now performed at a higher rate. Sulphides are processed at a lower throughput, with 300 g/t lead nitrate, to allow a retention time of 22 hours. Combined throughput remains the same, and total gold recovery has increased from 76% to 80% for an additional gold production of 16,000 ounces per year.

PAPER 35.4 — 9:45
THE RECOVERY OF GOLD FROM LOW GRADE AND REFRACOTORY ORES.
J. MCMULLEN, BARRICK Gold Corporation, Canada and
J.R. GOODE, J.R. Goode and Associates, Canada

Recognizing the global inventory of low grade and refractory gold deposits, low cost process alternatives that could turn these challenging deposits into mines are of interest. Such gold deposits have low value and potentially high operating costs and so pose special challenges for the mining engineer and metallurgist. Conventionally, refractory gold ores have been processed by conventional whole ore pressure oxidation and roasting, and flotation followed by roasting, pressure oxidation or bio-oxidation. Halide-based processes have recently been developed that might be applied to concentrate processing. Lower grade ores have been processed using heap bio-oxidation followed by heap leaching or milling. Other processes are emerging. Gold ores that contain cyanide-soluble copper can be difficult to economically process but can be amenable to other processes such as the Mt. Leyshon process for heap pre-leach removal of copper, the Hannah process for the IX recovery of gold, copper and cyanide or the Hunt process using the ammonia-cyanide leaching regime. Barrick has performed an inventory of the current and emerging technologies. A summary of the current “state of the art” will be presented with the view that with additional development some of these technologies or a combination thereof, may unlock the value of these deposits.

COFFEE BREAK — 10:10 – 10:30

PAPER 35.5 — 10:30
WHEN HEALTH AND SECURITY MEANS SAVINGS AT THE LARONDE REFINERY.
F. ROBICHAUD, J. FOURNIER and D. FORTIN Agnico-Eagle, Canada

Since milling began in 1988, the Laronde mill has been expanded several times, the most recent, increasing the milling rate from 5 000 to 7 000 mtpd. Laronde uses SAG/ball mill grinding followed by copper and zinc flotation circuits. A cyanidation/Merrill-Crowe process recovers the remaining gold and silver in the flotation tails in order to cope with much higher ratio of silver/gold associated with the discovery of the Penna shaft. Over the last four years, several modification and expansion projects were performed in order to reduce health and safety hazards since the commissioning of the Merrill-Crowe refinery in 2000. Remarkably, several of these improvements have contributed at increasing productivity and Net Smelter Return of the refinery while reducing worker’s injuries. In 2004, the refinery was audited to evaluate and quantify the circuit performance. Metallurgical and mechanical operating conditions of the refinery are now understood, and areas for further improvements are well defined. This paper describes some of the factors that have contributed to this improvement and describes how ergonomics and recovery can further be improved.

PAPER 35.6 — 10:55
FROG’S LEG GOLD PROJECT: A NON TRADITIONAL APPROACH TO DEVELOPMENT.
J. P. NICLOUD, Mines & Resources Australia, Australia and
D. CONNELLY, Mineral Engineering Technical Services Pty Ltd., Australia

The Frog’s Leg project is strategically located in the Kundana mining centre 22 kms west of Kalgoorlie, Australia. The Mines and Resources (MRA) Dioro Exploration joint venturer has had considerable exploration success centred on the high grade Frog’s Leg discovery. Gold resources of 749,500 ounces have been defined. At Frog’s Leg, there are three discrete resource domains recognised – Supergene, Contact, and Quartz Lode. This paper describes the metallurgical test work undertaken. The selection of the drill core was a joint effort by the MRA Senior Geologist and Mineral Engineering Technical Services (METS) to ensure the samples were representative of the ore body. Optimisation of gold recovery was essential since both contact and quartz have exceptionally high percentages of free gold and significantly higher grade than the typical open pit ores in Western Australia. The Metallurgical scope of work included documenting all of the metallurgical test results, establishing the comminution characteristics, determining the percentage of gravity recoverable gold, the cyanide leach characteristics and the tailings characteristics. This included describing the basis of design for a flow sheet to treat both ores based on test results and the advantages and disadvantages of the alternatives. Investigations also included toll milling options at various plants. Second hand and transportable plants were also considered. A considerable technical effort has been put into ore characterisation by way of test work aimed at managing the process risk at each stage of project development including mitigation. MRA also carried out detailed mineralogical analysis of the ores. The Frog’s Leg project development result has been characterised by innovation and unique solutions to maximise the returns for a small gold project. This included using
toll milling in order to generate funds to sustain ongoing exploration and hopefully project life. The alternative was to build a stand alone mill. It is a classical example of project development and delivery of small high grade gold projects including managing the project schedule and costs. The relationship with the toll milling operator has also been one of cooperative technical continuous improvement including shared risk and rewards.

PAPER 35.7 — 11:20
CONTINUOUS EVOLUTION OF PORCUPINE JOINT VENTURE (PJV) DOME MINE’S EFFLUENT TREATMENT PLANT.
J.A. FOLINSBEE and T.Y CHONG, PJV, Canada

Since milling began in 1909, the PJV Dome mill has been expanded several times, to the current throughput of 11,500 tpd. Dome recovers free gold using crushing and grinding followed by gravity concentration. The remaining gold in the gravity tails is recovered by a cyanide leaching followed by a Carbon-In – Pulp circuit. Prior to 1993, cyanide tailing were impounded in the tailing pond where cyanide was allowed to degrade naturally. Effluent was discharged seasonally during the summer and early autumn to maintain the water balance. In 1992, an Effluent Treatment Plant (ETP) was designed and constructed for the purpose of removing dissolved metals and suspended solids. In 1997, an INCO SO2/Air cyanide destruction process was added to the ETP. Even though the ETP was successful in meeting the cyanide, metals and TSS discharge limits, often the treated effluent failed to pass toxicity tests. Following successful laboratory and plant scale testing, the addition of the chelant EDTA (ethylene diamine-tetra-acetic-acid) was commenced in the 1999 operating season. To date discharged effluent has passed all toxicity tests. This paper describes the evolution of the ETP from its early conception to the various phases of upgrade needed to meet the MISA discharge limits.

WEDNESDAY, AUGUST 24, 2005, P.M.

SESSION 47: INTERNATIONAL SYMPOSIUM ON THE TREATMENT OF GOLD ORES

MODELLING AND ENVIRONMENTAL
Sponsor(s): Hydrometallurgy Section, The Metallurgical Society of CIM, Canadian Mineral Processors Division of CIM, CANMET-MMSL
Room: Imperial Ballroom 3
Chair(s): L. LORENZEN, University of Stellenbosch, South Africa and J.A. FOLINSBEE, PJV, Canada

PAPER 47.1 — 14:00
GOLD LIBERATION MODELLING OF DIAGNOSTIC LEACHING DATA USING NEURAL NETWORK ANALYSIS.
L. LORENZEN, N. MUSEE and N. KORNELIUS, University of Stellenbosch, South Africa

This study presents a neural network approach to modelling the liberation of gold bearing ores. A complete mineralogical analysis of unmilled and milled ores, including gold deportment and gangue content are used as inputs to a self-organising neural net which generates order preserving topological maps. The arrangement and shapes of these clusters are coupled to unmilled free gold data to predict gold liberation in milled ores (absolute error: 4.2%). Moreover, the self-organising maps were diagnostic of the quality of data used, indicating that the relationship between particle size and gangue material content requires further investigation.

PAPER 47.2—14:25
SIMULATORS FOR THE DESIGN, OPTIMIZATION AND CONTROL OF GOLD ORE PROCESSING PLANTS.
L.R.P. DE ANDRADE LIMA, McGill University, Canada, and D. HODOUIN, Université Laval, Canada

The paper presents a survey of the possibilities of the simulation techniques for improving the design and operation of gold processing plants. The procedure to construct kinetic models that are used in simulators is first reviewed, and examples are given for cyanide gold dissolution and gold adsorption on carbon. Then, an overview of the structures of the plant simulators is proposed, and the simulation equations formulated for a carbon-in-leach process. Finally, the potential of simulation methods is illustrated by four examples. The first one shows how it is possible to use a simulator to design a cascade of leaching tanks while optimizing an economic performance criterion. The second one illustrates the role of simulators for optimal tuning of the cyanide addition strategy in a leaching circuit. The third example focuses on the power of simulation to design carbon transfer strategies in a carbon-in-pulp process. Finally, the last example shows the role of on-line simulators for optimally supervising the automatic control of cyanide addition into leaching tanks.
While effective in treating refractory sulfide ores, pressure oxidation leaves carbonaceous matter unscathed and able to adsorb gold from cyanide solutions. Barrick Goldstrike’s strategy to mitigate this problem is to minimize contact between dissolved gold and native carbon in advance of the carbon-in-leach (CIL) process. Measures currently in use include limiting carbonaceous matter in the plant feed by meticulous blending of ore, minimizing chloride in the process streams, controlling the degree of oxidation in the autoclaves and blinding native carbon surfaces with lauryl sulfate. Residual cyanide in the process water remains to be a cause of premature gold dissolution and loss of gold by adsorption on carbonaceous matter. This cyanide is introduced by water that is recycled from the tailing pond. Several treatment options were considered and oxidation by peroxymonosulfuric acid, also known as Caro’s acid, was deemed to be the most effective. Laboratory and pilot plant tests show that Caro’s acid can oxidize free cyanide, weak-acid-dissociable (WAD) cyanide and thiocyanate, resulting in significant improvements in gold recovery downstream. The process requires Caro’s acid at 125 % of stoichiometric and pH in the range of 9.5 to 10 to accomplish the desired destruction in about twenty minutes. A cyanide destruction plant is currently being designed and slated for completion early in 2005.

PAPER 47.4—15:35
CYANIDE RECOVERY BY SOLVENT EXTRACTION IN A CONTINUOUS PILOT PLANT.
K. LARMOUR-SHIP, E. BUCHALTER Bateman Engineering, Canada, 
D. DREISINGER, B. WASSINK, University of British Columbia, Canada, 
R. HACKIL, M. HAMES, Placer DomeTechnical Services Limited, Canada, 
B. GRINBAUM, IMI TAMI Institute for Research & Development, Israel

Sodium cyanide continues to be the lixiviant of choice for gold dissolution world-wide. Many gold ore-bodies contain a significant amount of cyanide-soluble copper which increases the cost of processing due to high cyanide consumption and high cyanide destruction and disposal costs. Several cyanide destruction processes are in use today, however, there are few cyanide recovery circuits operating. A solvent extraction process utilizing the organophosphoruous extractant Cyanex 923 in Bateman Pulsed Columns. Copper cyanide species are first converted to dissolved HCN by acidification, with or without sulphide addition. HCN is then extracted and the organic stripped using a base (eg. NaOH) resulting in a high strength cyanide solution (eg. NaCN) for recycle to the milling circuit. Continuous mini-pilot testwork on HCN extraction using a 40 mm Bateman Pulsed Column demonstrated a cyanide recovery of greater than 99% with final levels of cyanide in solution of less than 10 ppm. These results illustrate the potential of this technology to recycle cyanide and detoxify final effluent streams.

PAPER 47.5—16:00
USE OF A ROTATING BIOLOGICAL CONTACTOR FOR THE REMOVAL OF NITROGENOUS COMPOUNDS FROM GOLD MILL EFFLUENT.
D. GOULD, S. MORTAZAVI, A. KAPOOR, P. BEDARD and L. MORIN, Natural Resources Canada, Canada

The degradation of thiocyanate in a simulated gold mill effluent was studied using a laboratory scale rotating biological contactor (RBC). Complete SCN⁻ removal was obtained at temperatures of 8 and 12°C, and a hydraulic retention time of approximately 3.3 h. The NH₄⁺ produced by thiocyanate hydrolysis was oxidized by nitrifying bacteria to NO₂⁻ and NO₃⁻. The HS⁻ produced by thiocyanate hydrolysis was chemically and biologically oxidized to elemental S, which was subsequently biologically oxidized to sulfate. Incomplete sulfur oxidation occurred in the RBC, particularly at 8°C and elemental sulfur deposition was observed on the RBC disks. The RBC feed exhibited toxicity to rainbow trout but the RBC was effective in removing the acute toxicity of thiocyanate.

PAPER 47.6—16:25
CYANIDE REMOVAL FROM GOLD PROCESSING FACTORY’S WASTE WATER USING CALCIUM AND SODIUM HYPOCHLORITE.
A. KHODADADI, P. TAIMORY and M. ABDOLADI, T. Modares University, Iran

Cyanide is used by the mining and chemical industries in tremendous quantities without attention to its hazardous problems. The use of cyanide compounds in mining is one of the most important environmental issues due to the acutely toxic properties of many cyanide compounds to human. Cyanide tends to react readily with most other chemical elements, producing a wide variety of toxic, cyanide-related compounds. Because cyanide is carbon based, an organic compound, it reacts readily with other carbon-based matter, including living organisms. This research is aimed at
investigating a feasible and economical technique for removal of cyanide from tailing wastewater of Muteh Gold Mine. In this research, removal from Muteh factory’s waste water was achieved at Tarbiat Modares University’s Mineral Processing Laboratory with the oxidation of cyanide by calcium hypochlorite. Cyanide oxidizes to cyanate (CNO-), which is environmentally 1000 times less toxic than cyanide. Cyanide titration was performed with and silver nitrate and rhodanine as indicator. Cyanide concentration in wastewater was 270 mg/L. Cyanide removal was achieved at an optimum pH of 12.3. At higher temperatures, cyanide was completely removed at this pH level. Optimum dosage for complete removal of cyanide using calcium hypochlorite was 1.43g/L.