Highlights of the Past Five Decades of Gold Ore Processing in Canada

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Introduction

Canada was the second largest producer of gold in the world in 1961, with 4.44 million ounces. South Africa was the largest producer with 22.94 million ounces in the same year (World Mineral Statistics Archive). The situation has changed significantly since then. In 2007, China became the leading gold-producing nation for the first time. They were followed by Australia, the United States and South Africa; with Canada in seventh place. Gold production in South Africa has halved since the 1960s owing to declining ore grades and rapidly increasing costs (Goldsheet Mining Directory). Cyanidation is the process that accounts for more than 80% of worldwide gold production, with the balance (< 20%) recovered either as a by-product of processing copper/gold flotation concentrates or by direct smelting of high-grade gravity concentrates.

The past five decades have seen dramatic economic and technological changes in the mining of gold deposits in Canada. The US Government set an official gold selling price of US$35/oz in 1934, which remained in effect for almost 40 years until several devaluations of the US dollar in the early 1970s effectively ended price fixing of gold. There was very little new mine development or technological innovation during this period of austerity. A significant proportion of gold was produced at an economic loss, with mining companies being subsidized by governments to sustain employment in the industry. The Canadian industry was sustained from 1948 until the 1970s by a subsidy, the Emergency Gold Mining Assistance (EGMA), with the aim of keeping remote mining industries alive. By 1971, 80% of Canada's gold was produced under this cost-aid system. Only the Campbell mine, with an ore grade of 20 g/t (0.6 oz), was not subsidized (Canada-Mining Introduction).

The rapid rise in gold prices in the mid to late 1970s triggered an increase in new mine development and several technological innovations were introduced. These included the developments of heap leaching, the carbon-in-pulp (CIP) process and new treatment methods for refractory gold deposits.

Much to the consternation of investors and mining companies, the price of gold declined rapidly from an extraordinary $815/oz in 1980 to about $400 two years later. It then found itself moribund in a price range of $300/oz to $450/oz for the next 25 years. Gold mining was profitable and sustainable in many parts of the world at these prices, in part because of the innovations that had been introduced in the 1980s. Gold production worldwide increased steadily in this period, doubling from ~45 million ounces in 1985 to ~85 million ounces by 2000. Production has stabilized at ~ 85 million ounces a year since then. The major mining areas of South Africa, Canada, Australia and the US have declined steadily in their output, while emerging countries such as China, Peru and Russia have increased. There is unlikely to be any significant increase in annual output from this level in the future without a significant, sustained increase in the gold price.

Since the early 1990s, most new gold projects have had to face challenges such as declining deposit ore grades, the transition from surface to deep underground deposits, growing complexity of mining methods and mineralogical associations of gold and stricter environmental conditions. Optimization of strategies for using reagents such as cyanide, oxygen and lead nitrate, as well as assaying and automated control methods have been the main areas in which these challenges have been addressed. Further innovations and improvements in processing methods and equipment design will be needed if these technical challenges are going to be met and gold production of around 85 million ounces per annum is to be maintained. The increase in price to > $1,000/oz in 2010 could see a new wave of technical innovations if it proves to be sustainable.

Mineralogy

Gold mineralogy plays an important role in gold ore processing and Canadian mineralogists and metallurgists made great contributions to it in the past 50 years. The developments in gold mineralogical study in Canada can be divided into three main eras: 1960s - 1970s, 1980s - 1990s and 2000s.

In the 1960s and 1970s, gold mineralogy was mainly focused on microscopic (visible) gold characterization and mainly involved the use of optical microscopes and the electron microprobe. During the late 1970s and early 1980s, an instrument for determining the quantities and liberation of minerals including gold was developed at CANMET (MS-SEM-IPS) by integrating a microprobe, energy dispersive X-ray analysers (EDS) and an image analyser (Petruk, 1989; Lastra and Petruk, 1994). In the late 1980s and early 1990s, as more and more refractory gold ores were discovered and processed, Secondary Ion

Mass Spectrometry (D-SIMS) was introduced into gold mineralogy for detecting and quantifying the submicroscopic gold contained in sulphide minerals in these ores (Chryssoulis et al., 1987; Cabri et al., 1989; Chryssoulis and Cabri, 1990; Cook and Chryssoulis, 1990; Fleet et al., 1993; Cabri and McMahon, 1996; Chryssoulis, 1997; McMahon and Cabri, 1998; Petruk, 2000). The application of MS-SEM-IPS and D-SIMS, along with some other techniques, brought gold mineralogy to a new level with regard to the characterization of microscopic and sub-microscopic gold.

Into the 21st century, the major efforts in gold mineralogy were directed at determining the deportment of gold in a sample, to predict the likely response of a gold ore to various processing methods, and to determine the cause of problems encountered in processing. AMTEL, CANMET and Lakefield Research (now SGS Canada) were the leaders during this period. In the late 1990s and through the 2000s, AMTEL and CANMET conducted a number of fundamental and applied studies using microbeam and surface analysis techniques (Chryssoulis et al., 2003; Chryssoulis and Dimov, 2004; Chryssoulis et al., 2004; Chryssoulis and McMullen, 2005; Deschênes et al., 2002a; Guo et al., 2005; Paktunc et al., 2006; Pratt et al., 2007). In the early 2000s, a comprehensive approach including a number of conventional and advanced mineralogy techniques (such as pre-concentration, microscope, electron probe and SIMS) was developed and implemented at SGS Canada to quantitatively investigate the deportment of gold in various ores and products. This approach has become a standard procedure in numerous gold projects at several laboratories since then. By using this approach, the mineralogist is able to quantify and balance the various types of gold occurrence in an ore and to predict the gold recovery of an ore or to determine the cause for gold loss in a tail (Zhou and Wang, 2003; Zhou, et al., 2004; Zhou and Cabri, 2004; Zhou et al., 2005; Zhou and Fleming, 2007; Zhou et al., 2009).

Here are two examples of this approach: Figure 1 shows native gold (Au) associated with calaverite (Cal, $AuTe_2$) and frohbergite (Fr, $FeTe_2$) in a Te-bearing gold ore. Calaverite is known to be slow dissolving during cyanide leaching and can cause gold loss due to its incomplete dissolution. Finer grinding of the ore or concentrate and a higher cyanide dosage will help improve the dissolution of calaverite. The recovery of the native gold locked in frohbergite and calaverite by direct cyanide leaching will be limited, and finer grinding is indicated to liberate or expose the locked gold grains. Figure 2 shows a liberated aurostibite (Ar, $AuSb_2$) grain with arsenopyrite (Apy) inclusions and native gold (Au) locked in arsenopyrite (Apy) in a Sb-bearing gold ore. Like calaverite, aurostibite is also slow dissolving during cyanide leaching and can cause gold loss due to its incomplete dissolution, while arsenopyrite is insoluble in cyanide solution. The extraction of gold locked in these two mineral phases will be very limited under normal operating conditions. Finer grinding would be required to improve the gold recovery from aurostibite, while chemical oxidation of the sulphide is generally required to liberate gold locked in arsenopyrite.

Other organizations, such as AMTEL and the University of Western Ontario, have also conducted gold-related studies using surface analysis techniques in recent years (Chryssoulis et al., 2003; Chryssoulis and Dimov, 2004; Dimov et al., 2009). Compared with those in other countries, Canadian researchers have done a lot of work and are in the leading position in this area.

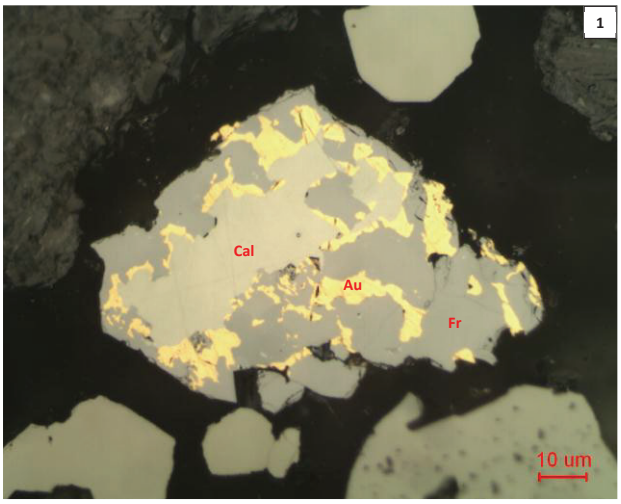

Figure 1. Photomicrograph showing a composite particle of calaverite (Cal), native gold (Au) and frohbergite (Fr) in a Te-bearing gold ore

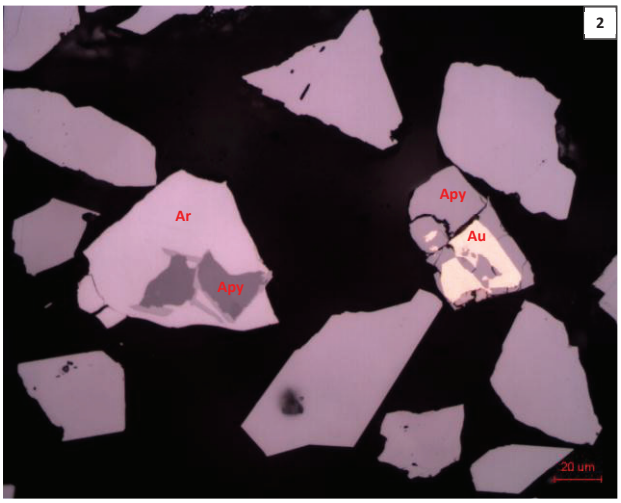

Figure 2. Photomicrograph showing a liberated aurostibite (Ar) and native gold (Au) locked in arsenopyrite (Apy) in a Sb-bearing gold ore

New Technologies

Milling

Autogenous grinding (AG, Figure 3) was used in iron ore

processing in 1959 (Jones, 1996). In the 1970's, the addition of small balls at up to 12% of the mill load was shown to improve mill throughput significantly, and semi-autogenous grinding (SAG) was born and became a more popular option when the hardness of the ore suited the process. Canadian engineer A. R. (Art) MacPherson was at the forefront of these developments and developed methods for testing ores and designing full scale industrial SAG mills (MacPherson, 1977). It was only in 1988, however, that SAG mills, Figure 3, began finding their way into new gold mines (Wells and Patel, 1991).

Figure 3. Ball mill and SAG mill in a cyanidation plant

Activated Carbon

Before the advent of carbon-in-pulp (CIP) technology, the Merrill-Crowe process (cementation with zinc powder) was the only process used for recovering gold from cyanide solution. It was the U.S. Bureau of Mines that developed the CIP process in the early 1970s (Hill, 1986), and it spread rapidly to the rest of the gold producing world in the 1980s and 1990s as a result of process innovations (pulp screening and pumping, fundamentals of loading, elution and reactivation) and large-scale CIP plant implementations in South Africa (Davidson, 1977; Fleming, 1982; Fleming and McDougall, 1987; Laxen et al., 1979). The robust price of gold ($400 to $800/oz) sustained much research and development effort through the 1980s, which led to further innovations, such as the carbon-in-leach (CIL) process. Canadian companies benefited from this research, which was conducted primarily in South Africa and Australia.

The main advantage of CIP and CIL over the Merrill Crowe process lies in the fact that they do not involve the use of solid liquid separation (filtration or countercurrent decantation). As such, CIP and CIL have reduced capital and operating costs, while also improving overall gold recovery. The CIP and CIL processes have proven to be reliable and robust, able to treat many different types of feed material and able to accommodate fluctuations in the process without a significant decrease in yield. Most new gold plants built in the world since 1980 have adopted activated carbon based technologies, and many old Merrill Crowe plants (particularly in South Africa) have converted from Merrill Crowe to CIP or CIL.

The mill at the Kiena Mine was one of the first in Canada using a CIP circuit (Jackson et al., 1986). Thereafter, all new Canadian cyanidation mills were designed with CIP circuits, and some existing plants were modified to use the technology.

Agitation

Mechanical agitation of cyanidation tanks gradually replaced pachucas. This new concept led engineers to modify air injection systems. Injection of air into the central axis of the unit was replaced by injection into pipes that exited perpendicular to the agitator blades (Richards and Wells, 1987).

Unless an ore contains minerals that consume oxygen, the diffusion of oxygen from the air above an agitated leach tank is sufficient to maintain a saturated oxygen concentration in solution (typically 5 to 8 mg/L) and sustain the kinetics of oxidation of gold. Consequently, injection of air into the slurry is no longer practised in gold plants treating "clean" non-refractory ores.

Refractory Gold Ores

Certain deposits that are "refractory" in nature (i.e., produce low gold yields in the cyanidation process) require pre-treatment in order to liberate gold. Prior to 1980, all gold plants treating refractory ores first concentrated the gold-bearing portion of the ore by flotation, and then roasted the float concentrate to convert the sulphides to SO_2 gas and produce a hematite calcine. Gold recovery from the calcine by conventional cyanidation and Merrill Crowe was typically in the 80 to 90% range. New technologies of pressure oxidation and bioleaching were developed and implemented after 1980. These processes offered a clean alternative to the toxic gases (sulphur dioxide and arsenous oxide) that were produced during roasting, and also generally resulted in improved gold recovery (typically 90 to 98%).

Many of the technical developments that led to the implementation of large scale refractory gold pressure oxidation plants came out of Sherritt Gordon (Weir, 1984, Weir and Berezowsky, 1984, 1986, 1987), who was already a world leader in the field of base metal pressure oxidation. For engineering and climate-related reasons, bioleaching has not been used commercially in Canada, but pressure oxidation has been implemented at several sites in Canada and is now widely used in other countries by Canadian mining companies.

Gravity Recovery

Recovery of gold by gravity was the main method of gold recovery prior to the implementation of the cyanidation

process around 1900. The cyanidation process was so efficient that gold recovery by gravity fell from favour for most of the 20th century, and there was only very limited application in Canadian gold mines and in mines outside of Canada. By 1988 only two of the ten largest gold mines in Canada were using this technology (Wells and Patel, 1991).

The recent development and industrial commercialization of highly efficient centrifugal gravity concentrators by two Canadian suppliers (Falcon and Knelson) has led to greatly improved gold recoveries by gravity separation, particularly finer gold particles, and this has revolutionized the industry over the last 30 years. The persistent efforts of the late professor André Laplante of McGill University, who worked in close collaboration with the supplier of one of the centrifugal concentrators over a period of two decades, convinced the gold mining industry of the value of incorporating gravity gold recovery in most milling circuits. Professor Laplante established a solid foundation for understanding gravity gold recovery in a Knelson concentrator (Figure 4) and for optimizing industrial gravity circuits.

Figure 4. View of a Knelson concentrator

The recovery of coarse gold by gravity separation enhances gold recovery by removing slow leaching gold grains prior to leaching, and it can also reduce capital costs by allowing for shorter leach residence time. In addition, gold that is recovered by gravity can be processed inexpensively to final bullion (often by direct smelting), so the diversion of gravity gold away from the CIP or CIL processes, where operating costs are higher, leads to an overall reduction in operating costs.

On-line Analysis of Cyanide

At the end of the 1980s, the Golden Giant Mine was the first Canadian mine to use an on-line free cyanide titrator called CYANOSTAT. In the same period, the Zelweiger analyzer was in common use in Australia in locations such as Mt Morgan and KCGM (Ellis, 2004). Use of the CYANOSTAT was halted after a few years because it was deemed unreliable. An Australian version based on the same principle, (i.e., spectrophotometry) and called the Alkay Analyser (Kaye and Jackson, 1991) was no more successful. The CYANCOR and the Cyanochem on-line analyzers were the next generation of automatic titrators introduced and used on a continuous basis in Canada (Dufresne et al., 1994; McMullen et al., 1999). Cyanide titration is performed with silver nitrate and is based on a potentiometric end point. A similar prototype, called Chemtronics, was commercialized in Australia (Hyde, 1995).

Numerous problems had to be solved to keep these instruments operational. While most of the difficulties associated with the sampling system were solved during the CYANOSTAT implementation phase, other problems had to be addressed, including robustness, reliability, corrosion resistance, maintenance and automation of sampling. The variation of the solution composition made the interpretation of results complex. It was also necessary to develop an automated cyanide addition strategy that would include the integration of the cyanide sensor. Many factors have delayed the adoption of this new technology, including the ease of manual titration of free cyanide, the capital requirements associated with the installation of an on-stream cyanide analyzer in the plant, labour requirements and lack of reliability, but results have indicated that the economic payback of on-line cyanide analyzers can be very rapid. Because the cost of cyanide is a major expense for many gold operations, a reduction as small as 10% in consumption represents significant savings. The use of the CYANCOR analyzer at the Yvan Vézina plant (Dufresne et al., 1994) and the Cyanochem analyzer at East-Malartic Division (McMullen et al., 1999) resulted in reductions in cyanide consumption of up to 40%.

Reports of successful implementations along with the pressure to reduce operating costs in all plants have contributed to the wider use of on-line cyanide analysis. It is estimated that about 40% of Canadian gold plants use on-line cyanide measurement today. Analyzers that have been installed over the past five years at Dome Mine, New Britannia Mine and Musselwhite Mine have contributed to a reduction in cyanide consumption. Agnico-Eagle, Laronde Division, uses two on-line analyzers supplied by Cyplus and called CyPlus® CCS 1100 potentiometric titration (Morin, 2006). Down times related to line clean-up and calibration represent only 5% of operating time. Monthly maintenance is performed for quality control.

Kalgoorlie Consolidated Gold Mines (KCGM), owned 50% by Barrick Gold, is using two types of analyzers: Process Analytical TAC 1000 and Cyantific Cyantist, both of which use potentiometric titration with silver nitrate. Figure 5 shows a photograph of the TAC. The on-line analyzers provide the information to a distributed control system (DCS) which adjusts cyanide addition to maintain a desired set point. The maximum addition rate is capped in case the analyzer gives a false reading at zero. For quality control, operators do manual titrations every 3 hours and compare with the analyzer, and a technician does daily manual titrations.

Figure 5. TAC on-line cyanide titrator

Use of Oxygen

Pure oxygen was first used to improve the rate of gold leaching in the cyanidation process by Air Products in South Africa in the 1980s (Arnold and Stephens, 1988). The practice was introduced in Canadian plants at about the same time (McMullen and Thompson, 1989). The Lac Minerals plants were the first to demonstrate the faster leaching kinetics associated with dissolved oxygen and lead nitrate. Use of oxygen-assisted leaching caught on quickly in the industry. The Canadian scientific contributions related to the use of oxygen in cyanidation were from Queen's University (Liu and Yen, 1995) and CANMET (Kondos et al., 1995). Air Liquide (Jara and Harris, 1994) and Inco (McLaughlin et al., 1999) proposed new devices for enhancing oxygen dispersion.

The practice of continuously monitoring and controlling dissolved oxygen in solution led to stabilization of the cyanidation performance and compensated for disturbances related to variability on the oxygen demand of an ore or concentrate. Improvements to the design of oxygen probes have added further robustness to the leaching operation and control strategy. The probe used at the East-Malartic/Bousquet operation has proven to be reliable with a high operating availability and minimum maintenance, owing to the ease of its calibration routine (McMullen and Thompson, 1989).

Cyanidation of Silver Ores

Most Canadian gold deposits are relatively low in silver (3 to 10 g/t). Canadian mining companies exploit mines with high silver grades in countries such as Mexico, Peru, Bolivia, Chile and Russia. Silver generally occurs in nature as the sulphide mineral acanthite, which is soluble in cyanide solution, but only leaches slowly. Industrial plants recovering silver from high-grade ores or concentrates have traditionally used high concentrations of cyanide (1,000-2,000 ppm) to overcome the slow leaching kinetics, but most plants still produced leach residues containing a lot of silver (up to150 g/t), despite long leach residence times of 72 hours or more (Martinez, 2006; Clingan, 2009; Osario, 2009). A technology has been developed in Canada (CANMET Enhanced Leaching Process, or CELP) to enhance the kinetics of silver (and gold) leaching at much lower concentrations of cyanide (500 ppm). This process has the potential to reduce operating costs, lower heavy metal content of liquid tailings, increase silver recovery and eliminate the need for cyanide recycling (Rajala et al., 2010). The first commercial application of CELP was at Kinross Gold's Kupol operation in Russia.

Cyanidation of Ores Containing Sulphide Minerals

There are potential problems in the cyanidation of free-milling sulphide-bearing gold ores, such as slower leaching kinetics, lower extraction of gold and higher consumption of cyanide and oxygen. The common practice for overcoming slow mass transfer in cyanidation is to increase the concentration of cyanide. The injection of oxygen (rather than air) into the slurry also greatly improves gold leaching kinetics in the presence of sulphide minerals.

The first review of cyanidation plants indicated that there was very little information available on the use of lead salts in commercial Canadian cyanidation plants. In 1994, a consortium project was launched to improve the understanding and role of lead nitrate in cyanidation. Seven cyanidation plants and Laval University's Department of Mining and Metallurgy participated in this project. The performance of the participating plants was improved and the fundamental mechanisms were identified. However, the project demonstrated that no reliable method of on-line control of lead nitrate addition was available. A second project, sponsored by a consortium of Australian, South African and Canadian gold producers and CANMET/MMSL led to the development of an on-line method that was demonstrated commercially at the New Britannia Mine (Duquet-Harvey and Deschênes, 2003).

CANNMET/MMSL conducted three surveys of cyanidation plants in collaboration with the Centre de Recherches Minérales du Québec, in 1993, 1995 and 1999. The objectives of these reviews were to facilitate the transfer of new technology to plant practice, to ensure that gold producers obtained supplementary knowledge to increase their productivity, to lower operating costs and to improve process efficiency. The last survey involved exchanging data from a survey conducted by the A.J. Parker Cooperative Research Centre for Hydrometallurgy in Australia in 1999 (Deschênes et al., 1999a).

Specific leaching strategies were developed to improve cyanidation of gold ores containing sulphide minerals such as pyrite, pyrrhotite, chalcopyrite, stibnite, realgar and chalcopyrite (Deschênes et al., 2000, 2002a, 2009a, 2009b). The addition of a lead(II) compound with or without oxygen significantly alleviated the negative effect of these minerals. Characterisation of the surface chemistry of leached gold and sulphide minerals improved the understanding of the leaching mechanisms. Application of findings to plants practice improved productivity and

reduced the toxicity of effluents (Deschênes et al., 1999b, 2002b; Deschênes and Fulton, 2002d; Deschênes et al., 2003, 2005, 2009a). Chebi (1992) contributed in an assessment of the influence of gold ore pre-oxidation on cyanide consumption.

The mechanisms involved in the role of lead nitrate in enhancing gold dissolution and decreasing cyanide consumption have been studied by electrochemical methods (May et al., 2005). A combination of sulphides and lead nitrate resulted in a potential fall in the active direction, which led to a much higher corrosion rate of a gold electrode in cyanide solutions. The mechanism of acceleration is believed to be due to the reduction of lead ions on gold in cyanide solution, forming lead-gold alloys ($AuPb_2$ and $AuPb_3$), which can be more readily oxidized than gold by dissolved oxygen, and transformed into gold-cyanide (Deschênes et al. 2000; Jin et al., 2002). Studies by Petre et al. (2008) and Azizi et al. (2010), identified secondary leaching mechanisms involving gold grain surface passivation or galvanic interaction with other minerals.

By studying the galvanic corrosion behaviour between several sulphide minerals coupled with gold, Yen and Aghamirian (2002) and Aghamirian and Yen (2005a) deduced that chalcocite and chalcopyrite exhibit a negative effect on gold dissolution, whereas galena, pyrite and pyrrhotite have a positive effect on gold dissolution. In addition to these sulfides, ferrocyanide ions at high concentration have a positive effect on gold dissolution, but copper cyanide at moderate concentrations has a slightly negative effect on gold dissolution (Aghamirian and Yen, 2005a, 2005b).

Heap Leaching

Although heap leaching was first practiced over five hundred years ago, it is only in the last fifty years that it became accepted as an effective method for treating low-grade gold and silver ores (Kappes, 2005). This practice expanded the amount of reserves and significantly increased the world gold production in the 1980s.

Heap leaching in Canada was put into practice in the early 1980s, with the first operation at the Caribou property of Anaconda Canada Explorations Limited near Bathurst, N.B. (Richards and Wells, 1987). This operation lasted just one year before the economically treatable ore supply was exhausted. The second operation began in 1984 at Gordex Minerals Limited's property, also in New Brunswick (Keating, 1999). The first large-scale operation was commissioned in 1987 at the Hope Brook mine in Newfoundland (Richards and Wells, 1987).

There are two main reasons for the limited application of this process in Canada, the first and foremost being the climate (Richards and Wells, 1987). The ideal operating environment for heap leaching is a temperate semi-arid desert location, and these are not found in Canada. The relatively high precipitation found at most mining districts in Canada and low ambient temperature for much of the year present a number of problems and can result in significantly lower precious metal recoveries. The rate of reaction is slowed as the solution temperature approaches its freezing point and as its viscosity increases. This increase in viscosity associated with colder temperatures affects the thickness of the meniscus on the surface of particles over which the solution flows, thereby tying up more process solution than heaps at warmer temperatures. For these reasons, and due to the fact that cold temperatures can cause freezing or slumping of the heap, the heap leaching season in Canada can be short, usually only around 6 to 8 months. Attempts to prolong this season through the addition of anti-freeze agents or the practice of distributing leach solution onto the heap under a thermal blanket of snow or waste rock are possible. Advances in these areas are well demonstrated at the large-scale low-grade Fort Knox heap leach operation in Fairbanks, Alaska, owned by Kinross Gold, which operates all year round (Ford et al., 2011) in an Arctic climate.

The second reason for the slow adoption of heap leaching practices in Canada is the generally unfavourable characteristics of Canadian ores (Richards and Wells, 1987). Refractory ores, as well as ores that require fine grinding for gold liberation and ores that contain species that consume excess cyanide (such as oxidized sulphides of As, Sb, Zn, Fe and Cu) or oxygen (such as pyrrhotite), are not well-suited to this process. This is due to the fact that it is difficult to control the leaching chemistry within the heap.

Effluent Treatment

Cyanide is used in the mineral processing industry as a flotation reagent and in the precious metals industry to recover gold and silver. Residual cyanide in leach solutions or tailings slurries must be destroyed or recovered prior to disposal due to its toxicity and potential for negative environmental impact. There have been two significant developments in Canada in this area.

The Inco SO_2/Air Cyanide Oxidation Process

Canadian nickel miner Inco developed a process that uses a combination of SO_2 and air, along with soluble copper added as a catalyst, to oxidize and destroy cyanide by the following reaction (Botz et al., 2005):

$$CN^- + O_2 + SO_2 + H_2O \rightarrow CNO^- + H_2SO_4$$

The process has several advantages over other cyanide destruction processes, such as alkaline chlorination, and gained widespread acceptance in the global gold mining industry after 1980. The process selectively attacks cyanide and cyanide complexes in both slurries and solutions, and readily reduces cyanide to low levels (e.g. less than 0.1 mg/L). Chlorination, by contrast, produces a far more strongly oxidizing environment, and other species in the

ore, such as sulphides, are attacked and oxidized, leading to high reagent consumption and high metal loadings in solution. The SO_2/air process also removes iron and iron-cyanide complexes from solution as insoluble ferrocyanide salts, whereas chlorination can stabilize iron-cyanide complexes in solution. If not removed from solution, these iron-cyanide complexes decompose during exposure to sunlight and slowly release free cyanide. The SO_2/air process also uses simple operating equipment and smaller amounts of less costly reagents (i.e. lime, SO_2, air, and copper sulphate), resulting in lower capital and operating costs compared to other cyanide destruction processes. Metal ions are precipitated from solution as hydroxides in the process.

The process was first piloted at the mill at Campbell's Red Lake Mine (now Goldcorp, Inc.) in 1982, with a Canadian patent issued for the process in 1984. This technology has been commercialized at over 75 sites worldwide (Devuyst et al., 1982).

The kinetics of ozone gas-cyanide reactions have been studied as an alternative approach for the treatment of cyanidation effluents, which claims several advantages for the oxidation of cyanide compounds.

The Acidification Volatilization Reneutralization (AVR) Process

The first commercial application of the AVR technology for recycling cyanide was in Flin Flon, Manitoba (Davis et al., 1946). The process involves acidification of the gold plant tailings to a pH of < 7, which converts the free cyanide and weak acid dissociable cyanide in the tailings to HCN gas. This gas is then stripped from the tailings in a stream of air, and the air/HCN mixture is scrubbed in a packed tower containing sodium hydroxide. This converts the HCN gas back to NaCN in solution, which can then be recycled to the leaching step. CANMET invested several years of research in perfecting this process.

A more recent Canadian development is the Sulfidation Acidification Recycling and Thickening (SART) Process, which is particularly well suited to gold leach solutions that contain a significant amount of copper cyanide (Fleming et al., 1998, MacPhail et al., 1998). The tailings are acidified in the SART Process, as in AVR, but in this case sulphide ions are also added, and react with copper to form chalcocite (Cu_2S). This precipitate is recovered by thickening and filtration, and the cyanide that was associated with the copper complex is recycled to the leach process as free cyanide. There are six commercial plants using this technology and several feasibility studies are underway that involve flowsheets incorporating SART.

Cyanide recovery involves a significantly higher capital investment than cyanide destruction, but a strong case for cyanide recovery can be made in many cases, particularly when excessive cyanide is consumed by reaction with copper minerals in the leaching process (Fleming et al., 2001).

Research and Development

Cyanide Substitutes

The gold industry has managed to work for more than 100 years, striving to operate safely while using a highly toxic cyanide reagent. Even though very few accidental human deaths have been recorded, the industry reputation has been damaged in recent years by several highly publicised cyanide spills in rivers. Consequently, increased environmental and regulatory pressures (along with the outright banning of cyanide in some countries) have stimulated a great deal of research to find alternatives to the cyanide leach process. This has proven to be an elusive pursuit up to now, mainly because cyanide is so extremely efficient that all other processes pale in comparison, both in terms of gold recovery and also reagent consumption and cost. Only a few processes have been developed at laboratory and pilot plant scale, using either thiosulphate or halide/halogen leaching chemistries, but none have advanced to commercial scale.

The greatest potential for the commercialization of an alternative leaching process is a thiosulphate leaching process for so-called preg-robbing ores. In this case it is possible to demonstrate significantly better gold recovery with thiosulphate than with cyanide, and this process is attracting a lot of interest from the major gold mining companies. SGS Lakefield and Barrick Gold are among the organizations that made a substantial contribution to research and development in this field (Yen and Wyszlouzil, 1985, Thomas et al., 1998, Fleming et al., 2001, 2003).

Thiosulfate leaching remains an area of active research at a number of institutions around the country, including the University of British Columbia, Queen's University and CANMET Mining and Mineral Sciences Laboratories, independent laboratories like SGS Lakefield and mining companies like Barrick Gold Corporation and (previously) Placer Dome Inc. Joint efforts involving Placer Dome and Lakefield were published by Ji et al. (2003). West-Sells and Hackl (2005) investigated the performance of thiosulphate leaching in presence of carbonaceous matter. The thiosulfate process is not robust and not suitable for all ores. The mineralogy of the material to be treated is known to impact the process. Current work using alternative catalysts has reduced thiosulfate consumption by 90%, increased the leach kinetics and minimized or eliminated the need for ammonia.

Barrick is actively working toward commercialization of the thiosulfate process for preg-robbing gold ores. SGS Lakefield and Barrick developed an effective thiosulfate resin-in-pulp system to treat slurry from the Goldstrike Mine in Nevada, using commercial strong-base anion exchange resins. For the double refractory ore types tested, the thiosulfate system achieved better gold recoveries than the cyanidation/CIL flowsheet. Some efforts were invested in situ leaching with thiosulphate (Choi et al., 2006).

Research at Queen's University investigated the effect of iron sulphide minerals during thiosulphate leaching (Xia and Yen, 2005) and lead nitrate (Xia and Yen, 2008). Queen's has introduced novel concepts and, by the use of chelating agents, thiosulfate consumption was reduced from 17 kg/t to 10 kg/t (Yen et al., 1996, 1999, 2002; Xia et al., 2002, 2003). Gold recovery from thiosulfate solutions was investigated using resins and by cementation with various metal powders (Arima et al., 2002; 2003; 2004). Recovery was above 95% using zinc and aluminum powders at metal:gold mass ratios of 30:1 and 50:1 in the case of copper powder.

Substantial efforts by Laval University and CANMET led to the development of a flowsheet for the extraction of gold from a copper concentrate using thiourea. Deschênes (1989) assessed the influence of the various process parameters: granulometry, leaching solution concentration, nature and concentration of oxidizing agent and reducing agent, acidity, agitation, pulp percentage, temperature and redox potential. The kinetics of the reactions were studied and an optimized system was suggested. Links were established between E-pH diagrams of Au-H_2O and Au-$SC(NH_2)_2$ (thiourea) H_2O, and the results of the leaching study. This effort paved the way for the treatment of gold-bearing chalcopyrite deposits and of its sulphide tailings. Dupuis and Ghali (1988) studied the direct electrolysis of the leach solution and the influence of ferrous and ferric ions was thoroughly investigated. Using a cationic membrane, gold recovery was above 90%. In addition to electrolysis, a study was carried out of the influence of key parameters on the reduction of the gold-thiourea complex by hydrogen (Deschênes, 1989; St-Amant, 1990).

Lacoste-Bouchet et al. (1998) and Deschênes et al. (1994) work on the development of thiourea leaching for gold-copper ores. Application of thiourea to underground gold leaching was investigated by Tremblay (1996). Percolation leaching of gold by thiourea was carried out in the presence of sulphuric acid and hydrogen peroxide for two low content gold minerals in the province of Québec. Acid consumption was the limiting factor, although gold leaching was in the range of 76% to 83%. Other investigations by Deschênes et al. (2002c) were on developing gold underground leaching by thiourea.

Bioleaching

For the past ten years, research efforts at Queen's University have been geared towards finding novel microbes that have the potential to degrade carbonaceous matter and improve gold recovery. The coal solubilizing actinomycete, *Streptomyces setonii*, and the fungus *Trametes versicolor* have been tested, and in all cases increases in gold recovery were obtained. Two stage pre-treatments were conducted with several combinations including chemolithotrophic bacteria leaching followed by either *Streptomyces setonii* or *Trametes versicolor.* For the ore types tested, cyanidation after treatment with the chemolithotrophs yielded between 70% and 82% recovery. After treatment with *Streptomyces setonii* or *Trametes versicolor*, gold extraction increased to between 87% and 95% (Choi et al., 2009; Yen et al., 2008; Afidenyo et al., 2008; Amankwah et al., 2005a).

Resins

Significant effort has been invested in the development of resin in pulp (polymers with a strong or weak base functional group) as an alternative to CIP or CIL. The process accounts for about 10% of world gold production and is used quite widely in Russia and the Commonwealth of Independent States (CIS) countries. It has found very limited appeal in the west, apart from a few niche applications (and none in Canada). Each process (carbon or resin based) has its advantages and disadvantages and both are very efficient extractants for gold cyanide from solution or pulp in most applications.

Resins are favoured in Russia and the CIS today because they were locally manufactured and readily available in the USSR, when gold processing plants were being established and when resins were already widely used in the region' uranium industry (Laskorin et al., 1974). Carbon, on the other hand, had to be imported, which was difficult to do under the Soviet regime.

Once the iron curtain was lifted and imports such as activated carbon became available, there was little incentive to replace an already efficient resin-based process with activated carbon (Dementyev, 2005). The reverse happened in the west. Most of the resin-in-pulp research and development work was done in the 1980s and 1990s at the same time, or soon after the period when carbon-based plants were being rapidly built in South Africa, Australia, Canada and the USA. By the time resin plants could be rolled out as an alternative process, CIP and CIL were already well established and there was little incentive to replace these very efficient carbon based processes with resins, even through incremental advantage could be demonstrated in some cases.

The greatest potential use of anion exchange resins in the gold mining industry in the west is to recover and recycle free cyanide and metal cyanide complexes in the tailings. Resins are able to adsorb cyanide complexes very efficiently, and present the opportunity to recover cyanide and metals from tailings and, by doing so, significantly lower tailings detoxification costs. (Fleming, 1998; Fleming and Trang, 1998).

Grinding

Microwave pretreatment has been tested to augment the grinding of a free-milling gold ore containing quartz, silicates and iron oxides. Under microwave irradiation, selective heating of the different mineral components

resulted in thermal stress cracking. After microwave processing, the grindability of the ore was improved and the crushing strength and the Bond Work Index were reduced by 31.2% and 18.5%, respectively. In addition to the enhanced grindability, gold was released from the matrix of the host minerals at a coarser size, resulting in a significant increase in free gold recovery by gravity concentration (Amankwah et al., 2005c).

Measurement, Modelling, Simulation, Control and Optimisation

Gold Grain Surface Liberation

Gold ore leaching kinetics are primarily controlled by the probability of contact occurring between gold grain surfaces and the leaching phase. This has been quantified by the convolution of the two random distributions of ore particle sizes and gold grain sizes. The concept of gold surface liberation has been studied by two different stochastic geometry approaches at Université Laval. In the 2D approach, matrix fragmentation is represented by a distribution of Voronoï cells and the gold phase by a random distribution of rectangular grains of different sizes in the gangue matrix (Khalesi et al., 2009a, 2009b; Khalesi, 2010). In the 3D approach, both the ore matrix fragments and the unbreakable gold grains are assumed to be cubic (Bellec, 2011; Bellec et al., 2009). The two approaches have been indirectly validated in laboratory cyanidation tests through coupling with a leaching diffusion model.

Activated Carbon

The CIP and CIL processes have proven to be metallurgically and mechanically robust, and high gold recoveries are generally achieved with minimal optimization or appreciation of the factors that may cause one plant to perform differently from another. The stellar attributes of the carbon-based gold plants has led to complacency in the industry and many CIP and CIL plants are over-designed at the construction stage, using basic "rules of thumb" that were developed as design criteria for earlier CIP plants. This leads to higher gold losses and/or higher capital and operating costs than necessary, but there is little incentive to improve performance when most plants operate quite well without optimization.

The rate of adsorption of gold cyanide onto activated carbon is very slow by normal industrial standards, and it can take months to attain true equilibrium loading. Because of this, all CIP and CIL plants are operated under conditions in which the amount of gold loaded on the carbon in each stage is far from equilibrium loading. Thus, gold extraction efficiency is always based on the kinetics of adsorption and is never limited by equilibrium constraints. With CIP and CIL plants operating far from equilibrium, there is always plenty of capacity on the carbon to absorb more gold. Understanding and modeling the rate at which gold loads can translate to lower capital and operating costs, either by decreasing the size or number of adsorption stages, or by increasing the loading of gold on the carbon. Also, CIL should only be selected (over CIP) when the rate of leaching is very fast. Knowledge of both leaching and adsorption kinetics must be understood and modeled to make the correct process selection.

The CIP and CIL model that is probably used most often in Canada is based on the early models of Nicol et al. (1984a and 1984b), which have been adopted and adapted by SGS Minerals and used both for new plant design purposes in a number of feasibility studies (Fleming et al. 2010), and also for retro-optimization of the performance of existing plants (Fleming et al., 2011).

Two kinetic models for describing the reversible gold cyanide adsorption reaction on carbon have been proposed. The first one (Carrier, 1989) is based on an empirical equation involving three first order terms with respect to gold contents in the aqueous and carbon phase and free adsorption sites in the carbon phase. The second one (Chefaï, 1990) makes use of three diffusion and adsorption mechanisms: diffusion to the external surface, diffusion through macropores, and finally diffusion through micropores. Both models are sufficiently flexible for correctly representing adsorption test results.

Microwave energy has been used to combust waste activated carbon and the resulting ash was treated by conventional cyanide leaching to recover the gold. Over 95% gold extraction could be achieved within eight hours. The results of this research demonstrate that microwave combustion of gold-containing waste activated carbon is technically viable for gold recovery (Amankwah et al., 2005b).

Cyanidation Kinetics

By conducting leaching tests in an instrumented leaching vessel (Bissonnette, 1989; Kane, 1991; Hodouin et al., 1990; de Andrade, 2001), it has been shown that gold dissolution kinetics can be represented by a second order empirical rate function for some Abitibi area ores. The associated rate constant depends upon the oxygen and cyanide contents of the leaching phase and of the ore particle size (de Andrade and Hodouin, 2005b; Bazin et al., 2009; Egan, 2007). This phenomenological approach has been compared to multivariate statistical approaches (de Andrade and Hodouin, 2003). It was also established in laboratory scale leaching tests (Bellec, 2011) that chemical diffusion of reactants to the free gold surface (Habashi, 1967) is most probably the limiting kinetic phenomena for agitation levels corresponding to those of industrial equipment. Coupling surface liberation and diffusion models, it was possible to mathematically represent leaching test results performed on the same ore comminuted under various conditions.

Data Reconciliation and Model Calibration

Measured values of gold contents in liquid and solid phases (ore and possibly carbon) and in the size intervals of the solid phases, are always inconsistent in the sense that they do not verify the law of gold mass conservation. This is usually quite critical when collecting data from an industrial plant because of sampling errors, measurement errors and unavoidable dynamic variations in the operating conditions (Bellec, 2011). The problem is also observed in controlled laboratory tests, although less critical (Bissonnette, 1989; Egan, 2007). Methods have been developed to reconcile data, i.e. for measured variables adjustment as well as unmeasured variables estimation, and thus to produce data that obey mass conservation laws (Cimon et al., 1987; Bellec et al., 2007). Data pre-processing by reconciliation techniques increases the reliability of gold extraction performance indicators and of the model parameters calibrated from experimental data (breakage, leaching, adsorption reactions rates, and classifier separation coefficients as functions of particle size and gold content).

Plant Simulators

Various plant simulators have been developed, both for dynamic and steady-state operating regimes (Hodouin and de Andrade 2005; Bazin et al., 2007; Bellec et al., 2009). They include grinding-classification circuits, leaching circuits, adsorption circuits, and integrated leaching-adsorption circuits. In addition to the kinetic models for grinding, leaching and adsorption, these simulators contain mixing (de Andrade and Hodouin, 2005a), material transfer, and hydrocyclone models, as well as computational convergence control, optimal tuning, real-time optimisation, and automatic control modules. Some examples are described below.

Optimal Plant Design and Tuning

The adsorption plant simulator (Hodouin et al., 1987; Carrier et al., 1987) was used to study the effects of the tank number and sizes, and for the determination of the optimal quantity of carbon to be maintained in the tanks and to be transferred from tank to tank. It was also shown how to transfer carbon optimally to minimize gold losses.

Using the grinding circuit simulator, it has been shown that the preferential recycling of gold bearing particles by hydrocyclones is a major factor explaining why gold recovery is high when performing cyanidation directly into the grinding circuit (de Andrade and Hodouin, 2006b).

The cyanidation circuit simulator was used to select optimal tank sizes and cyanide distribution into the tanks (de Andrade and Hodouin, 2005c and 2006a), as well as to explore the option of separate cyanidation of fine and coarse particles, for minimizing equipment size and cyanide consumption, at a targeted gold recovery (Bellec, 2011).

Automatic Control and Real Time Optimization

Raw ore property variations such as gold content, grain size distribution, mineral composition and grindability changes, as well as modifications of external factors such as gold market price and energy or reagent costs are disturbances that modify the gold extraction process behaviour. In order to compensate for their effects, automatic control strategies that are able to track cyanide contents in the various leaching tanks have been evaluated by simulation (Hallab, 2010). A strategy for optimal supervision of the cyanide content set-points has also been proposed for maximizing the net revenue or gold recovery.

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