ALTA 2023

27 YEARS

30 April - 6 May | Perth, Australia

27TH Annual Conference Proceedings

Gold-PM
Conference

Sponsored by

FLSMIDTH

ALTA Metallurgical Services, Melbourne, Australia

www.altamet.com.au

PROCEEDINGS OF
ALTA 2023 Gold-PM SESSIONS

4 May 2023

Perth, Australia

978-0-6458390-2-9

ALTA Metallurgical Services Publications

Gold-PM Contents

Sponsored by

	Page
Keynote Address: Chloride – A Precious Metals Leaching Medium Yet To Reach Its Potential **Mike Adams,** *Lifezone Limited, Isle of Man*	1
The Amira P420 Gold Processing Benchmark Surveys **Erica Avelar,** *Curtin University Western Australian School of Mines, Australia*	15
Stripping and Electrowinning Optimization in Gold and Silver Process **Göksel Sendur**, *Zenit Madencilik, Turkey*	28
Metso Outotec Flowbottom, First Reference Experiences And Look Back To Development **Tuosmas Hirsi,** *Metso Outotec, Finland*	36
Solving Scale Problem In Process Tanks With Swirl Flow Agitation **Bon Nguyen,** *CSIRO Mineral Resources, Australia*	44
Driving Stagewise Development Of Gold Projects With The Jameson Concentrator And Albion Process™ **Laurie McDonnell,** *Glencore Technology, Australia*	56
Novel Corrosion-Resistant and Erosion-Resistant Coating Qualified for Use In Pressure Oxidation Severe Service **Evelyn Ng,** *Callidus Process Solutions, Australia*	64
Process Development for Hydrometallurgical Recovery of Base and Precious Metals from Waste Printed Circuit Boards **Huan Li,** *Curtin University Western Australian School of Mines, Australia*	77
Mintek's iGoli Process. History, Piloting and Techno Economics **Elmar Muller,** *Mintek, South Africa*	86
The Diversity of Recyn Technology **Malcolm Paterson,** *Greengold Advisory, Indonesia*	107
Alternative Gold Refining Procedure Using 'Organic Aqua Regia' **Akihiro Yoshimura,** *Chiba University, Japan*	114
Introducing the Third Force In Metallurgy **Melinda Moore,** *Viinca Technologies, Canada*	122
Update on Cyanide Free Leaching of Gold Ores Using Glycine Leaching Technology **Glen O'Malley & Elsayed Oraby,** *Mining and Process Solutions*	127
Update on Glycat Leaching of Gold Ores and Concentrates **Glen O'Malley,** *Mining and Process Solutions*	139

GOLD-PM KEYNOTE

CHLORIDE – A PRECIOUS METALS LEACHING MEDIUM YET TO REACH ITS POTENTIAL

By

Mike Adams, Keith Liddell

Lifezone Limited, Isle of Man

Presenter and Corresponding Author

Mike Adams
mike@kellprocess.com

ABSTRACT

For centuries, oxidising chloride medium has been well known for its ability to dissolve precious metals including gold, silver, and platinum group metals (PGM) – platinum, palladium, rhodium, ruthenium, iridium. In modern times, gold and silver are mainly recovered by alkaline cyanidation and PGM by smelting – both of which process routes suffer environmental disadvantages.

Cyanidation has been the most widely used process for the extraction of gold from its ores for over 120 years. More recently, commercial application of cyanide in gold mining has been under increasing pressure around environmental concerns, particularly after high-profile cyanide spills at Baia Mare, Romania, and elsewhere.

Processing of PGM concentrates is typically by smelting at ~1550^{o}C to a green matte, converting at ~1350^{o}C to a white matte, removal of base metals by medium-temperature and pressure sulphuric acid leaching. The resultant precious metals refinery (PMR) feed contains 30-70% PGM and is suitable for chlorination leaching, typically using small-scale equipment operating on a batch basis. This process route is energy-intensive with a resulting high carbon footprint and PGM smelters are mostly located in regions with unstable electricity supply. Moreover, many smelters continue to allow sulphur dioxide to be emitted to the atmosphere with no sulphur abatement or off-gas scrubbing measures in place.

Chloride as a low-emissions and low-toxicity leaching system is receiving increasing interest but has yet to realise its full potential. Chlorination chemistry is well understood and the leaching rates are extremely fast – on the order of minutes as opposed to hours or days for cyanidation. This results in relatively small equipment and makes for low capital intensity and low precious metals inventory lockup in-process.

Current chloride-based leaching processes are reviewed, considering their development status and pending or current applications. Of particular note is the application of Lifezone's hydrometallurgy technology to several applications treating PGM concentrates in South Africa to produce refined metal products (Pt, Pd, Rh, Au, Ru, Ir, Ni, Co, Co) at the minesite in a footprint area considered to be about 10-15% of that for the equivalent pyrometallurgical plant. Lifezone hydromet flowsheet development and implementation status shall be presented. These initiatives are significant milestones, representing a potential game changer in the broader application of chloride-based hydrometallurgy to precious metals separation and recovery.

Keywords:

Chloride, precious metals, hydrometallurgy, gold, platinum, palladium, rhodium, PGM, Kell Plant

Background

Chloride leaching of precious metals

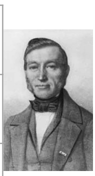
Discovery of Aqua Regia and Chlorine

Aqua Regia was discovered by Persian alchemist Jabir Ibn Hayyan (720-813) by the mixture of hydrochloric and nitric acids

Chlorine was discovered by the Swedish chemist Carl Wilhelm Scheele (1742-1786) by the action of hydrochloric acid on manganese dioxide in 1774

Early Chlorination Processes for Gold Extraction

Process	Date	Pretreatment	Equipment	Treatment	Leach	Recovery
Plattner	1848	Dead roast	Damp fixed bed vat	Cl_2 permeation	Water percolation	$FeSO_4$ reduction
Duflos	1848	-	Damp fixed bed vat	In leach	Supersaturated Cl_2 percolation	Reduction
Mears	1877	-	Rotating barrel	Cl_2 300 kPa	Water leach	Reduction
Thies	1881	-	Rotating barrel	In leach	$CaOCl_2/H_2SO_4$ leach 30 kPa	Reduction
Lange	1885	-	Earthenware pots	Cl_2	Water leach	H_2S reduction
Munktell	1888	Dead roast	Damp fixed bed vat	Hot water then HCl or H_2SO_4 for Cu/Fe removal	$CaOCl_2/H_2SO_4$ leach	Lead acetate



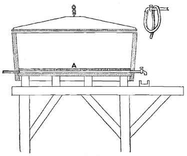
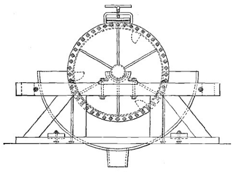


Fig. 62.



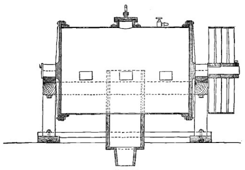


Fig. 64.

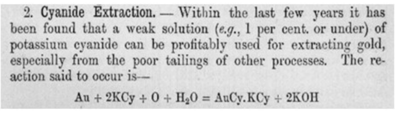
Early Processes for Generation of Chlorine:

$2NaCl + MnO_2 + 2H_2SO_4 = Na_2SO_4 + MnSO_4 + 2H_2O + Cl_2$

$CaCl_2 + Ca(OCl)_2 + 2H_2SO_4 = 2CaSO_4 + 2H_2O + 2Cl_2$

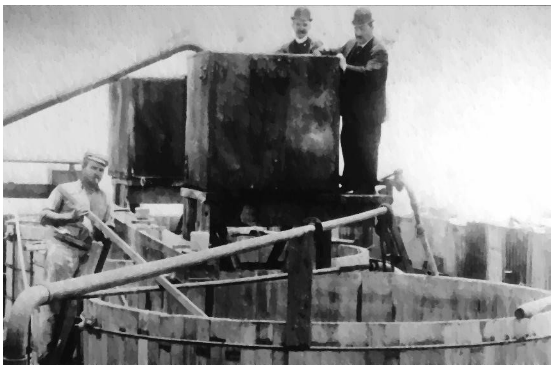
Factors in the Replacement of Gold Chlorination by Cyanidation

- Excess chlorine losses due to volatilisation from leaky vessels
- Excess chlorine consumption due to reaction with sulphides
- Excess chlorine consumption due to reaction with base metals

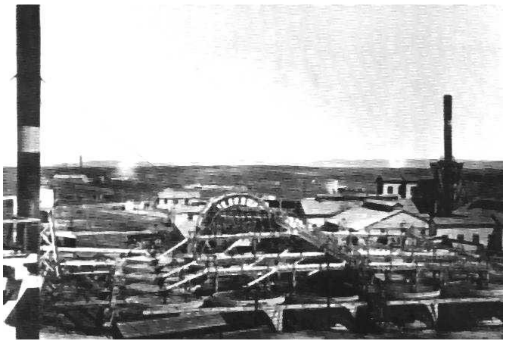


2. **Cyanide Extraction.** — Within the last few years it has been found that a weak solution (*e.g.*, 1 per cent. or under) of potassium cyanide can be profitably used for extracting gold, especially from the poor tailings of other processes. The reaction said to occur is—

$$Au + 2KCy + O + H_2O = AuCy.KCy + 2KOH$$



Robinson GM, 1890



Simmer & Jack, 1893

Chemistry of Chlorination

Precious metals leaching reactions

Chloride Leaching Chemistry

Gold chlorination chemistry

$2Au(s) + 3\,Cl_2(g) + 2\,HCl(aq) = 2\,HAuCl_4(aq)$

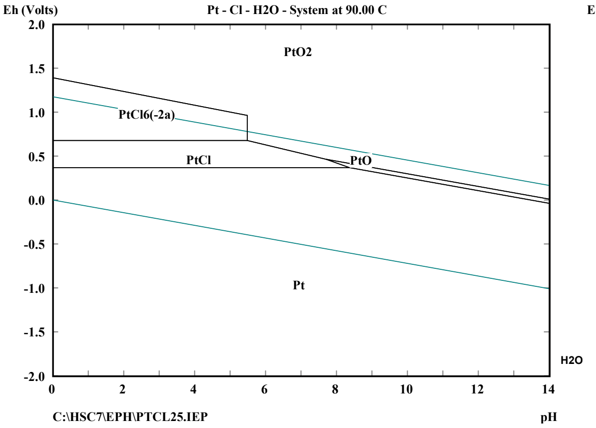
PGM chlorination chemistry

$Pt(s) + Cl_2(g) + 2\,HCl(aq) = H_2PtCl_4(aq)$

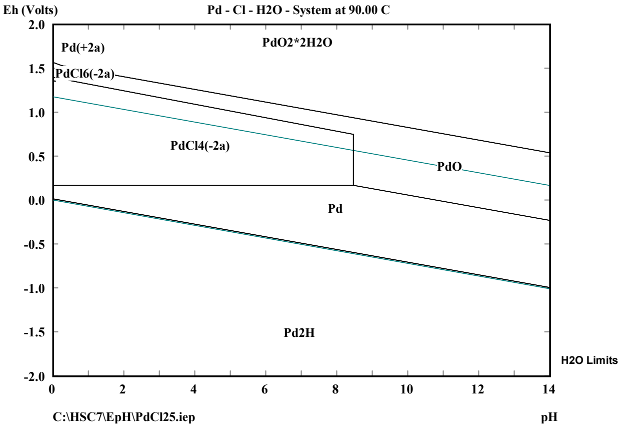
$Pt(s) + 2\,Cl_2(g) + 2\,HCl(aq) = H_2PtCl_6(aq)$

$Rh(s) + Cl_2(g) = RhCl_2(aq)$

Potential-pH Diagram for Pt-Cl-H_2O and Pd-Cl-H_2O Systems

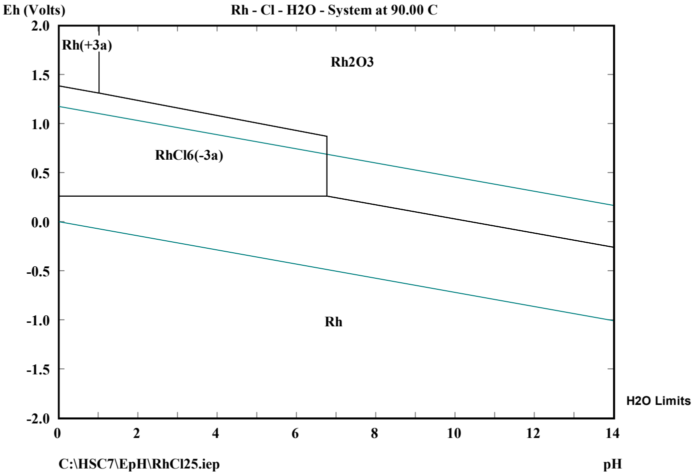


ELEMENTS	Molality	Pressure
Pt	4.100E-05	1.000E+00
Cl	6.200E+00	1.000E+00



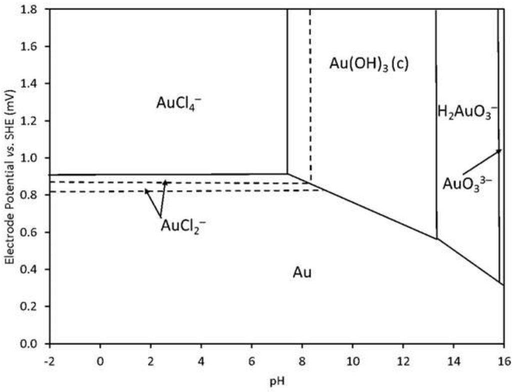
ELEMENTS	Molality	Pressure
Pd	4.100E-05	1.000E+00
Cl	6.200E+00	1.000E+00

Potential-pH Diagram for Rh-Cl-H_2O System

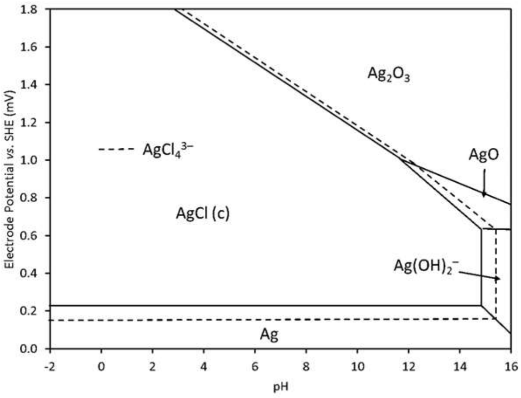


ELEMENTS	Molality	Pressure
Rh	4.100E-05	1.000E+00
Cl	6.200E+00	1.000E+00

Potential-pH Diagram for Au-Cl-H_2O and Ag-Cl-H_2O Systems



$a_{Au} = 10^{-3}$ and a_{Cl} = 1 (solid) and = 5 (dashed); 25°C
Kelsall et al., 1993



$a_{Au} = 10^{-4}$ and a_{Cl} = 1 (solid) and = 2 (dashed); 25°C
Welham et al., 1993

Chloride-Based Leaching Processes

Gold and PGM recovery

Chloride-Based Leaching Processes for Gold

Process	Ligand	Oxidant	Tested	Application Level	References
Hydrocopper	Cl^-	Cu^{2+}, O_2	D		Hyvärinen and Hämäläinen (2005), Canadian Mining Journal (2006), and Infomine (2008); Lundström et al (2009, 2011, 2012)
Intec	Cl^-/Br^-	Cu^{2+}, Fe^{3+}	P		Moyles (1999), Severs (1999), and Monument Mining Ltd (2016)
Lifezone Metals	**Cl^-**	**Cl_2**	**P**	**FS (2x sites, South Africa)**	**Liddell (2009), Liddell et al. (2011), Liddell and Adams (2012a,b), Adams et al. (2015, 2019, 2020)**
N-Chlo	Cl^-/Br^-	Cu^{2+}, Fe^{3+}	D		Nippon Mining and Metals (2008)
Neomet	Cl^-	Cu^{2+}, Fe^{3+} HNO_3 O_2 H2O2	P		Harris and White (2011a,b, 2013)
Nichromet	Cl^-/Br^-	Cl_2/Br_2	P		Lalancette (2009), and Lemieux et al. (2014)
Outotec	Cl^-/Br^-	Cu^{2+}, Fe^{3+}	P		Miettinen et al. (2013)
Platsol	Cl^-	O_2	P	FS (Northmet, USA)	Ferron et al. (2000, 2003), Wardell-Johnson et al. (2009), Dreisinger et al (2016)

Chlorination in Platinum Refineries

A heated agitated 2.2 m^3 titanium vessel for batch dissolution of a PGM concentrate via intermittent chlorine sparging

Photograph courtesy of Impala Platinum
(after Crundwell et al 2012)

Platinum-Group Metals Refineries

Parameter	Unit	Lonmin	Kratsvetmet	Vale Acton	JM/Anglo	Impala
Main feed material		Leached matte	Anode slime	Concentrate/ Catalyst/ E-Waste	Leached mag concentrate	Leached matte
HCl	g/L	219	175			109
Temperature	C	65	70	90-98	120	65-70
Pressure	bar	1	3	1	4	0.7
Residence time	h	6	2			8
Material		Glass-lined	Ti			Ti Gr2; jacketed
Volume	m3	2.2	1			2.5
Impeller		Pitched-blade turbine	Hollow shaft inductor			Pitched-blade turbine
Free Cl_2 removal		Air sparging				"Dechlorinated"
Gold recovery method		N2H4 reduction/HCl	Precipitate with concentrate	SX	SX	IX
Gold product		Crude gold	99.99% Au	99.99	99.95	99.95
Typical gold recovery	%	99	98	99	99.5	94
First-pass yield	%	92		98	84	98

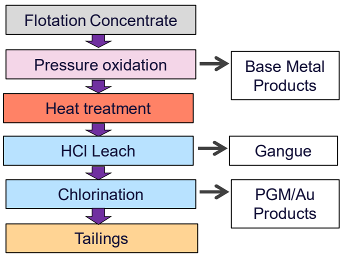
Lifezone Metals

Gold and PGM Hydromet Processing

Lifezone Metals Hydromet – Overcoming the Chlorination Limitations

The globally patented hydrometallurgical Lifezone Metals Process:

- ***Chlorine losses*** due to volatilisation
  - minimised via properly engineered vessels
- ***Chlorine consumption*** due to reaction with sulphides and base metals
  - eliminated via prior removal of these reagent consumers by pressure oxidation
- ***Gas-liquid-solid contact*** constraints in static and rolling reactors
  - eliminated by use of modern CSTR reactors
- ***Refractory, complex*** and deleterious PGM-Au-gangue mineral phases
  - eliminated by heat treatment conditioning and preleach prior to chlorination



Lifezone Metals Hydromet Process Background

The globally patented hydrometallurgical Lifezone Metals Process:

- Developed for the extraction of platinum group metals (PGM), gold, silver and base metals from PGM sulphide flotation concentrates without smelting or cyanidation
- Successfully demonstrated on various PGM and polymetallic concentrates, including UG2, Merensky, Platreef and Great Dyke from southern Africa and polymetallic concentrates from North America, Australia and Africa.
- High recoveries (94 % - 99 %) for selective extraction of a range of value metals, including precious metals (Pt, Pd, Rh, Au, Ag) and base metals (Ni, Co, Cu, Zn, Sn, Pb, Mo) with stabilization of impurities such as As and S without gaseous emissions.
- KellGold process variation for refractory gold, Cu/Au and preg-robbing gold concentrates, enabling production of refined metals at the mine site without the use of cyanide
- Licensed as Kell Process in SADC region via KellTech Ltd, along with shareholders Sedibelo Platinum and IDC (Industrial Development Corporation of South Africa)

Lifezone Metals Hydromet – Technical Features

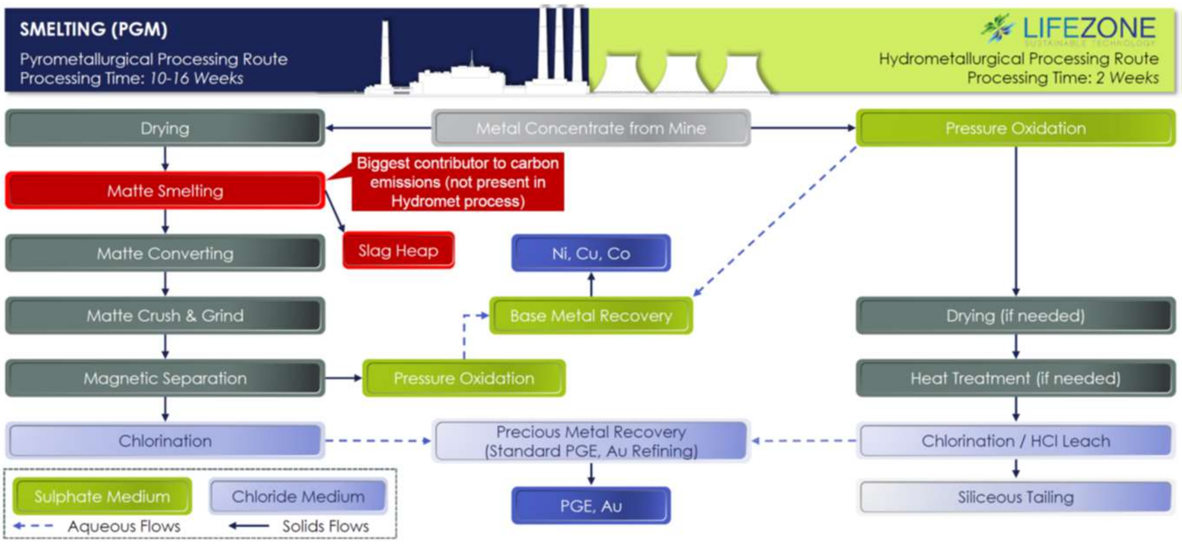
Patented process comprising four basic sequential steps, all of which are well proven and commonly used in the metallurgical industry and provide high recoveries of base and precious metals:

1. **Aqueous pressure oxidation** in an acidic sulphate medium to dissolve the sulphides and remove the base metals while minimising dissolution of the precious metals, producing copper, nickel and cobalt products;
2. **Heat treatment** of the pressure oxidation residue as required, to condition the mineral phases, rendering the material amenable for subsequent leaching;
3. **Atmospheric leaching** of the iron and other gangue components in chloride media;
4. **Atmospheric leaching of the PGM and gold** in chloride medium in a similar manner as typically used in PGM refineries, with PGM and gold recovery to high-purity products

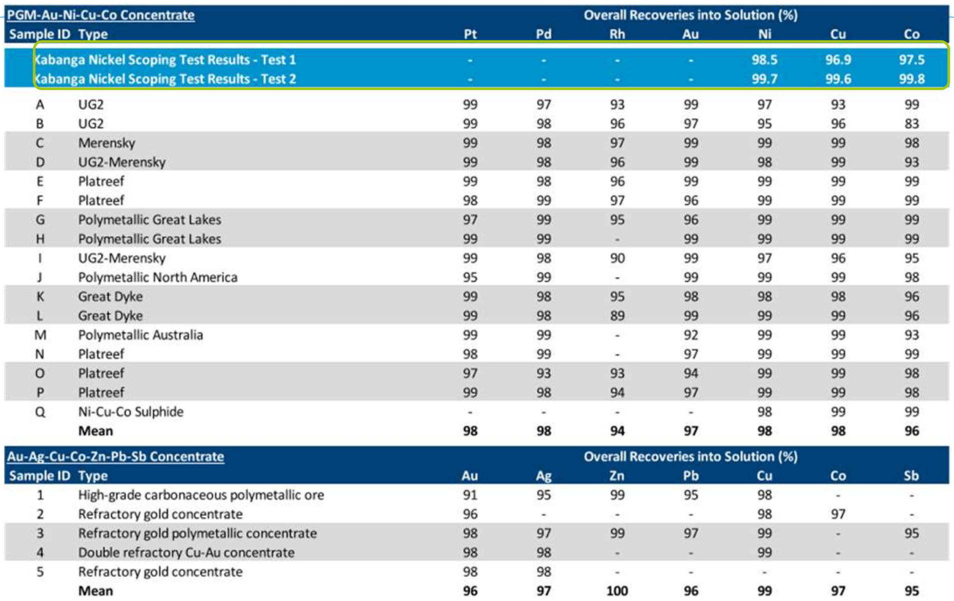
Separate leaching stages for the precious metals and base metals:

- keeps sulphate and chloride chemistries separate
- small autoclave and tanks – fast leach kinetics
- optimizes selection of materials of construction
- fast production of metals and low lock-up of value

Lifezone Metals Hydromet – Comparison with Smelting



Lifezone Hydromet Testwork – Metals Extraction Results

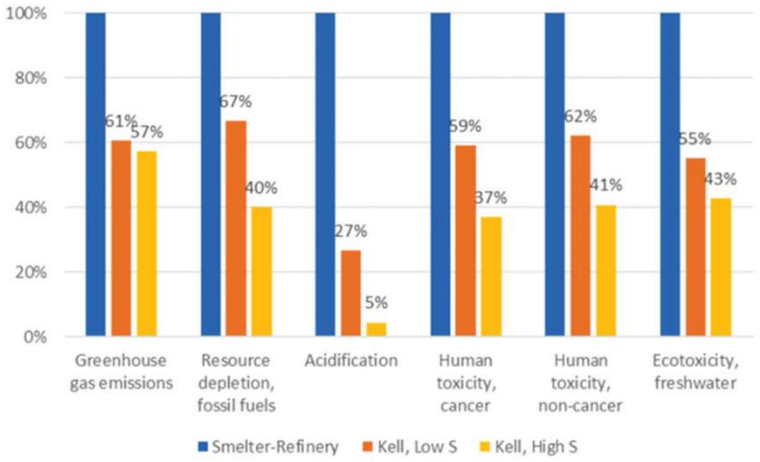


PGM-Au-Ni-Cu-Co Concentrate		Overall Recoveries into Solution (%)						
Sample ID	Type	Pt	Pd	Rh	Au	Ni	Cu	Co
	Kabanga Nickel Scoping Test Results - Test 1	-	-	-	-	98.5	96.9	97.5
	Kabanga Nickel Scoping Test Results - Test 2	-	-	-	-	99.7	99.6	99.8
A	UG2	99	97	93	99	97	93	99
B	UG2	99	98	96	97	95	96	83
C	Merensky	99	98	97	99	99	99	98
D	UG2-Merensky	99	98	96	99	98	99	93
E	Platreef	99	98	96	99	99	99	99
F	Platreef	98	99	97	96	99	99	99
G	Polymetallic Great Lakes	97	99	95	96	99	99	99
H	Polymetallic Great Lakes	99	99	-	99	99	99	99
I	UG2-Merensky	99	98	90	99	97	96	95
J	Polymetallic North America	95	99	-	99	99	99	98
K	Great Dyke	99	98	95	98	98	98	96
L	Great Dyke	99	98	89	99	99	99	96
M	Polymetallic Australia	99	99	-	92	99	99	93
N	Platreef	98	99	-	97	99	99	99
O	Platreef	97	93	93	94	99	99	98
P	Platreef	99	98	94	97	99	99	98
Q	Ni-Cu-Co Sulphide	-	-	-	-	98	99	99
	Mean	**98**	**98**	**94**	**97**	**98**	**98**	**96**

Au-Ag-Cu-Co-Zn-Pb-Sb Concentrate		Overall Recoveries into Solution (%)						
Sample ID	Type	Au	Ag	Zn	Pb	Cu	Co	Sb
1	High-grade carbonaceous polymetallic ore	91	95	99	95	98	-	-
2	Refractory gold concentrate	96	-	-	-	98	97	-
3	Refractory gold polymetallic concentrate	98	97	99	97	99	-	95
4	Double refractory Cu-Au concentrate	98	98	-	-	99	-	-
5	Refractory gold concentrate	98	98	-	-	-	-	-
	Mean	**96**	**97**	**100**	**96**	**99**	**97**	**95**

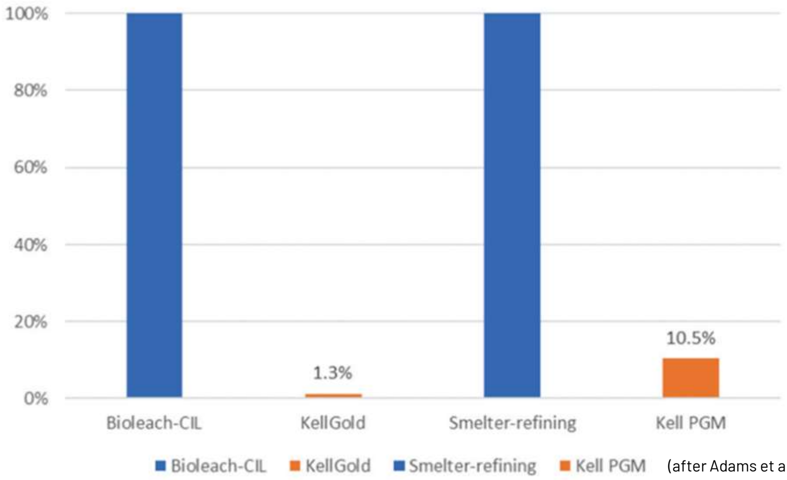
Liddell, K.S., Adams, M.D., Smith, L.A., and Muller, B 2019. Kell hydrometallurgical extraction of precious and base metals from flotation concentrates – Piloting, engineering, and implementation advances. Journal of the Southern African Institute of Mining and Metallurgy.

Comparison of Environmental Profiles



(after Smith, Adams, and Liddell, 2019)

Comparison of Working Capital Value Lockup (In-process Metal Inventory)



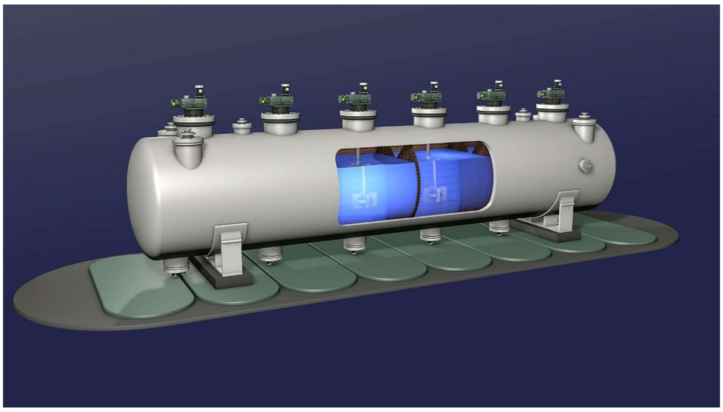
(after Adams et al 2019)

Outlook

Chloride-Based Leaching Systems

Outlook for Chloride-Based Leaching Systems

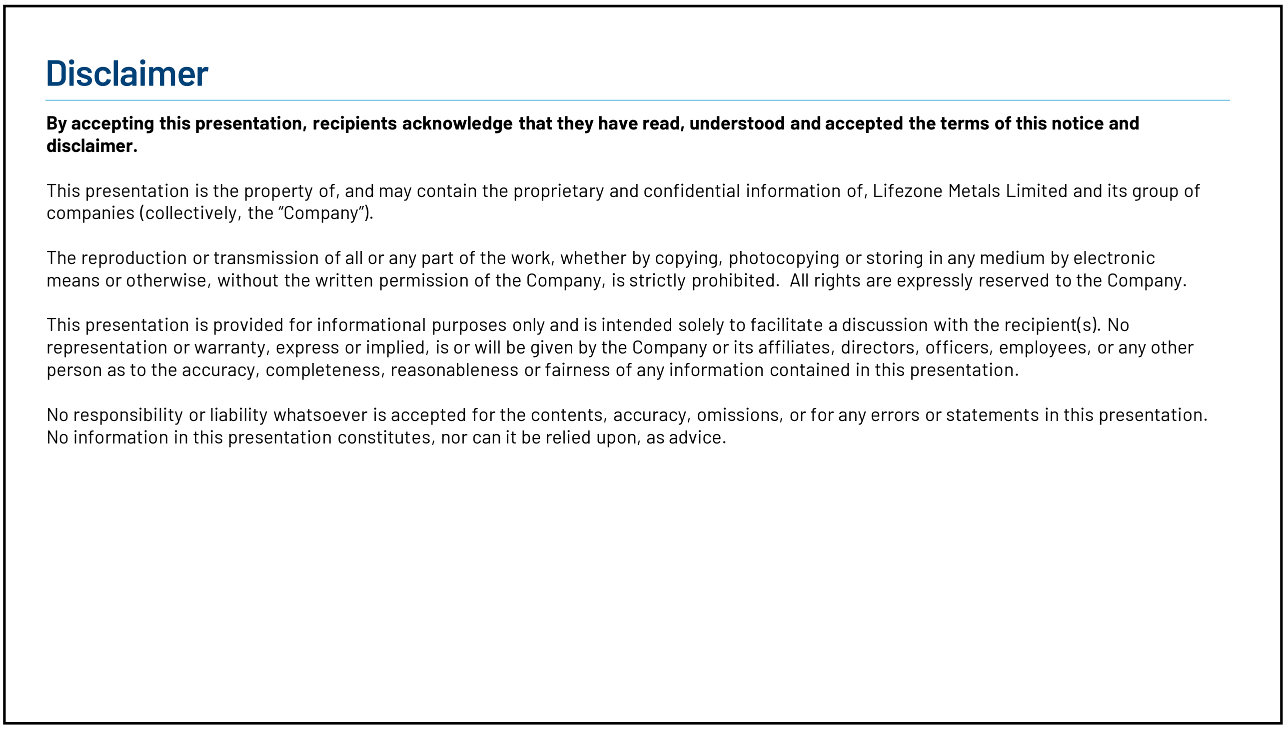
- Several chloride-based leaching systems have been tested in recent years at batch or pilot scale
- Lifezone Metals technology is at DFS or Detail Design stage for several commercial-scale applications
- Implementation of bulk chloride-based leaching systems at commercial scale is likely to take place in the next few years



Acknowledgements

Lifezone Metals

Simulus Engineers & Laboratories



Disclaimer

By accepting this presentation, recipients acknowledge that they have read, understood and accepted the terms of this notice and disclaimer.

This presentation is the property of, and may contain the proprietary and confidential information of, Lifezone Metals Limited and its group of companies (collectively, the "Company").

The reproduction or transmission of all or any part of the work, whether by copying, photocopying or storing in any medium by electronic means or otherwise, without the written permission of the Company, is strictly prohibited. All rights are expressly reserved to the Company.

This presentation is provided for informational purposes only and is intended solely to facilitate a discussion with the recipient(s). No representation or warranty, express or implied, is or will be given by the Company or its affiliates, directors, officers, employees, or any other person as to the accuracy, completeness, reasonableness or fairness of any information contained in this presentation.

No responsibility or liability whatsoever is accepted for the contents, accuracy, omissions, or for any errors or statements in this presentation. No information in this presentation constitutes, nor can it be relied upon, as advice.

THE AMIRA P420 GOLD PROCESSING BENCHMARK SURVEYS

By

Erica Avelar[1,2], [1]Alan Bax, [1]Bill Staunton, [3]Teresa McGrath

[1]Curtin University, Western Australian School of Mines: Minerals, Energy and Chemical Engineering, Perth, Australia
[2]ARC Centre of Excellence for Enabling Eco-Efficient Beneficiation of Minerals, Curtin University Node, Perth, Australia
[3]Anglogold Ashanti, Perth, Australia

Presenter and Corresponding Author

Erica Avelar
erica.avelar@curtin.edu.au

ABSTRACT

Benchmarking through industry practice surveys has been a key component of the Amira P420 project throughout its history. These activities can be referenced as four main benchmark survey events: the 1994 Australasian Industry Survey, the 1999 Gold Processing Survey, the 2012 Worldwide Industry Survey and the 2021 Amira P420G Benchmarking Surveys. Those surveys were designed as questionaries to identify gold processing practices amongst the participating sites, and the wider industry. The survey questionnaires were completed by site personnel, with the information received being compiled and anonymised by the Amira P420 Gold Technology Group. The questionaries covered unit processes from milling through to the goldroom.

This paper will present the general trends observed in gravity recovery, leaching and adsorption, and carbon management from the latest 2021 survey, and compares it to data from the previous surveys. The benchmark exercise aimed to identify changes in operational philosophy and improvements.

Keywords: gold, benchmark, processing, gravity, classification, leaching.

INTRODUCTION

The Amira 420 benchmarking has been a tool to identify major changes in the gold industry, as well as to identify best practices. Industry practice surveys have been a key component of the Amira P420 project throughout its history. To date four main benchmark surveys were conducted: the 1994 Australasian Industry Survey, the 1999 Gold Processing Survey, the 2012 Worldwide Industry Survey and the 2021 Amira P420G Benchmarking Surveys.

1994 Industry Survey

As a part of the P420 project gold plant operating practices were surveyed by way of a general questionnaire. The questionnaire contained 452 questions relating to operating conditions, the level of process control used in the circuit, monitoring and sampling practice and a variety of other items covering aspects of the operation from grinding to tailings disposal and storage. The questionnaire was sent to 76 operating gold plants in Australia, New Zealand, Papua New Guinea and France. A total of 66 responses were returned, 30 of which were from companies sponsoring the project. The survey results were manually collated from the questionnaires and summarised in a number of different reports. The survey results were not added to any spreadsheets or databases and the 1994 survey isn’t included in this review.

1999 Australasian Industry Survey

The 1999 Gold Processing Survey set, comprised four surveys: an Australasian Industry Survey, an Elution & Electrowinning Survey, a Carbon Management Survey and a Canadian survey.

- **1999 Australasian Industry Survey** – A total of 41 Australian sites completed a detailed plant survey questionnaire. The questionnaire was divided into ten main sections covering various aspects of the plant including types of ore treated, grinding circuit flowsheet, gravity concentration circuits, leach and adsorption circuit configuration and operating conditions, tailings disposal, sampling, carbon management, elution and electrowinning, gold room practices and metallurgical accounting methods.

- **1999 Elution & Electrowinning Survey** – this was a detailed survey questionnaire concerning elution and electrowinning circuits carried out at seventeen sponsors’ operations. The questionnaire covered various aspects of elution and electrowinning including typical levels of gold and contaminants found on the carbon, acid washing, elution conditions, details of electrowinning operation, sampling strategies and management issues and cost structure.

- **1999 Carbon Management Survey** - A detailed survey questionnaire of carbon management practices was carried out at fifteen sponsor sites. The questionnaire was divided into five main sections covering various aspects of carbon management including carbon residence time distribution studies, carbon regeneration conditions, carbon concentration determination, carbon movement philosophy and carbon management guidelines.

- **1999 Canadian Survey** - This survey presents general information and operating data collected from eleven Canadian gold plants and assembled by the Mining and Mineral Science Laboratories of CANMET in collaboration with the Centre de Recherches Minérales de Quebec. It is the third such review of operations and it was reciprocally shared with the Amira P420A project in return for the 1999 Australasian Survey data.

2012 Worldwide Industry Survey

Completed in 2014, this Gold Industry Practices Survey had twenty-nine companies covering 77 mining operations for the online survey, with locations in Australia, Asia-Pacific, Africa, Latin America and North America. The survey included 568 questions divided into 15 sections. Twenty-eight of the 77 registered sites completed the survey fully and 21 sites completed less than 10% of the questions with the remaining 28 sites completing between 10 % and 100 % of the questions.

2021 Amira P420G Benchmarking Surveys

The 2021 Amira P420G Benchmark Survey had the participation of a total of 8 gold companies and was completed by 39 sites. It consisted of 12 questionaries Dissolved Oxygen Sensors, Gravity Feed

Screening, Cyclones, Gravity, Leaching & Adsorption, Carbon Management, Flotation, Elution and Electrowinning, Carbon Regeneration, Gravity Concentrate Processing and Goldroom Smelting. The results of this survey are still being processed by the Gold Technology Group. Hence, the results presented the preliminary analysis of the survey data.

This paper focuses on presenting the key data collected in the gold gravity recovery, leaching and adsorption and carbon management benchmark survey in 2021 and comparing it where applicable to the 2012 and 1999 surveys.

SURVEY METHOD

Separate survey questionnaires were created for each area of interest and shared with the participant companies via the P420 website (http://www.goldknowledge.com). Personnel from sponsor companies completed the questionnaires online. Some site personnel also completed spreadsheet questionnaires that were subsequently provided to the P420 project staff who manually transferred the data from the spreadsheets to the online questionnaires.

Survey Questions

The questions for each survey were developed from questions used in industry benchmarking surveys that were conducted during previous P420 projects, new questions that were based on feedback from various sponsors as well as areas of interest for the project researchers.

SURVEY RESULTS

Throughput and Gold Production

Data from the 1999, 2012 and 2021 surveys have been tabulated in Excel. Figure 1 shows the General database is primarily populated with entries from the 1999 and 2012 Surveys, with only 28 of the total 139 (20 %) entries corresponding to the current 2021 Survey. However, in the 1999 survey, only two countries were represented with that number peaking at 14 in the following 2012 Survey.

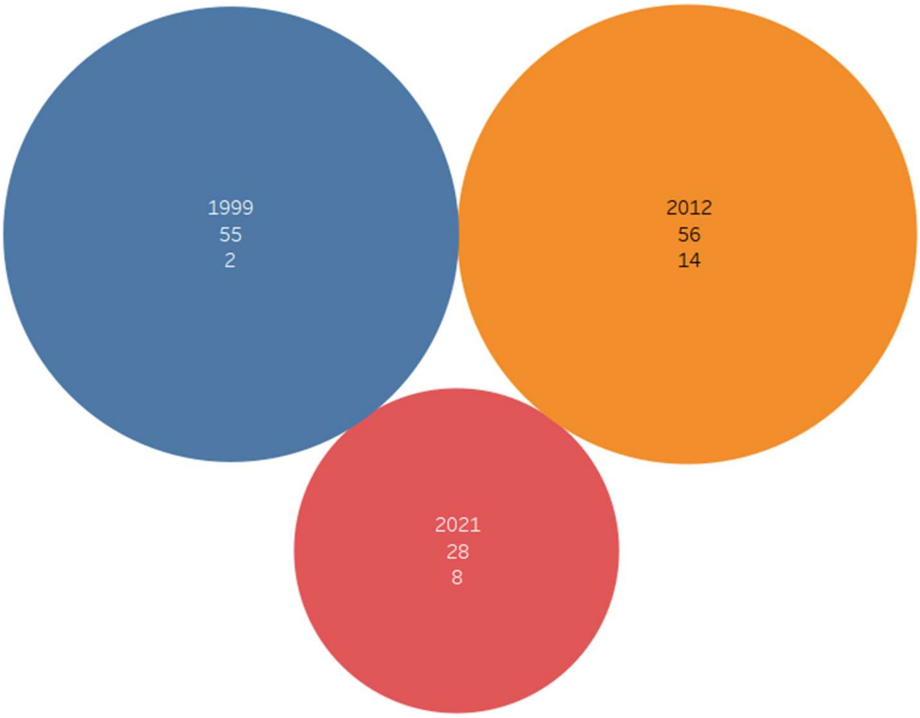


Figure 1: General Survey Database - Year/Number of Entries/Number of Countries [1]

Figure 2 shows the countries participating in each round of the surveys. With only eight countries participating in the 2021 Survey, it is hoped that additional entries will be made from more sites across the world before the end of the Amira P420H project.

Figure 2: Countries Participating in the General Surveys, by Year [1]

Figure 3 shows the annual throughput for sites participating in each of the surveys. This means sites with survey entries for each period will have the unique opportunity to track changes and progress over time. Interestingly, the survey shows an increasing throughput trend of the participant operations.

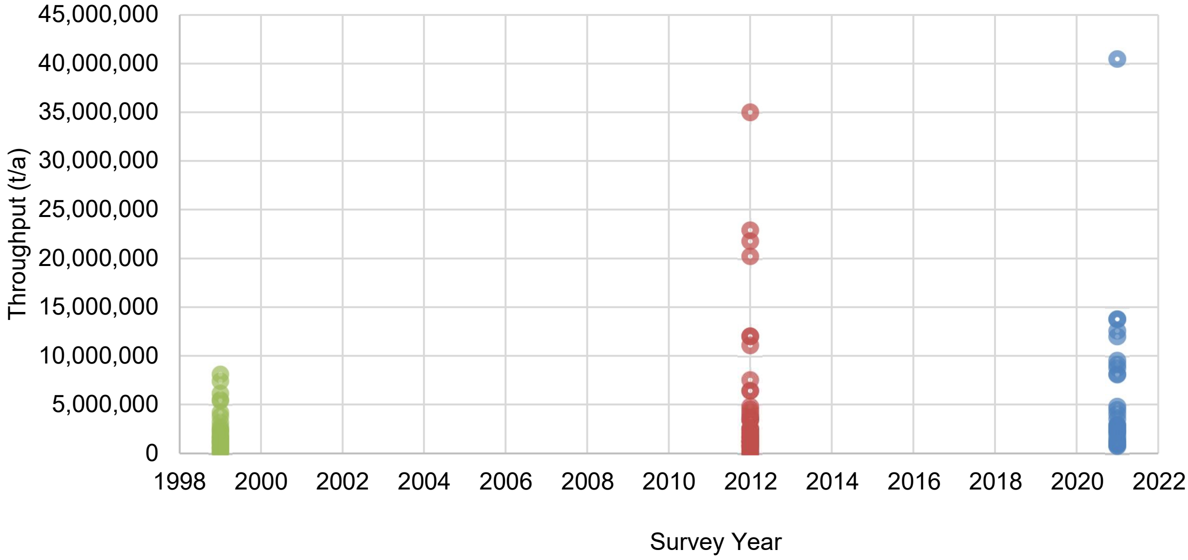


Figure 3: Countries Participating in the General Surveys, by Year

Figure 4 shows the gold production for the years 2012 and 2021. The survey also indicated an increasing trend following the throughput.

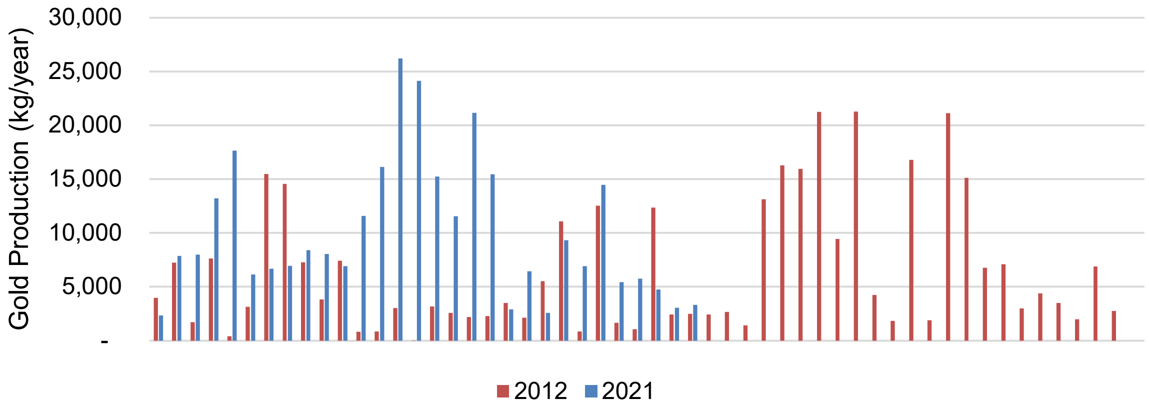


Figure 4: Gold Production in kg/year on the Participant Sites, by Year [1]

Gold Gravity Recovery Circuits

Table 1 shows the proportion of gold recovered by the gravity circuit in the sites surveyed in 2012 and 2021. The average percentage of the total gold recovered from gravity circuits indicated an increase from 2012 to 2021.

Table 1: Gravity Gold Recovery

Gold Gravity Recovery (% of total gold recovery)	2012 Survey	2021 Survey
Average	25	39
Median	15	35
Minimum	1	15
Maximum	80	72

The gravity recovery survey of 2021 covered 37 gravity circuits located in 21 different sites. The type of comminution circuit in which the gravity circuits are installed is compared in Figure 5.

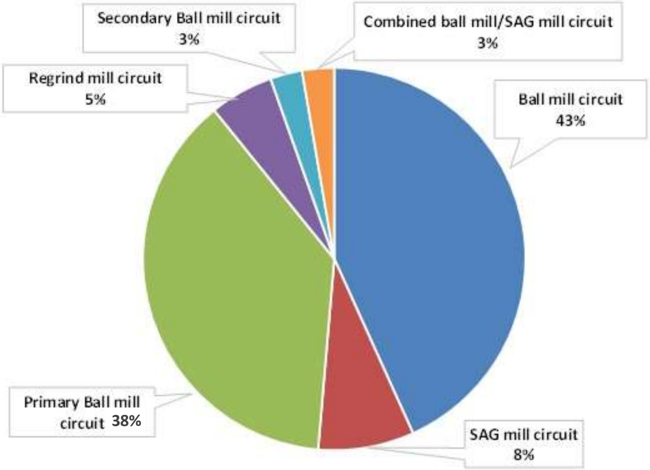


Figure 5: Location of Gravity Circuit

The feed streams for the surveyed gravity circuits are compared in Figure 6**Error! Reference source not found.**. The majority of the gravity circuits in the surveyed plants are treating the cyclone underflow stream.

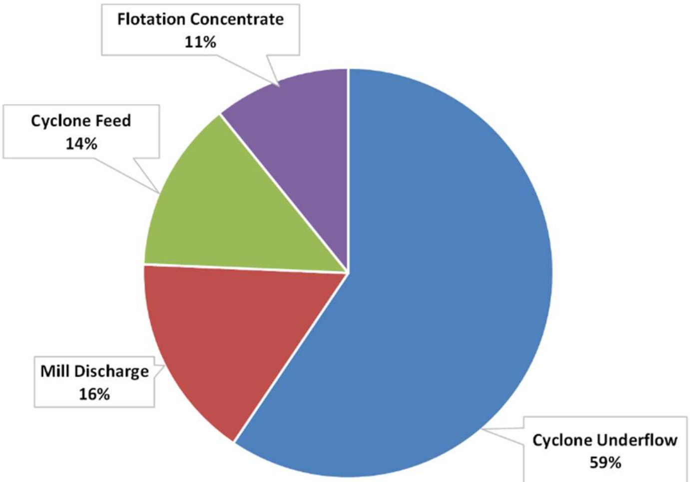


Figure 6: Primary Gravity Recovery – Feed Stream

Figure 7 compares the average proportion of the feed stream reporting to the gravity circuit for the different feed sources. Gravity circuits receiving flotation concentrate treated the entire stream and circuits receiving cyclone underflow processed the lowest proportion of the feed stream.

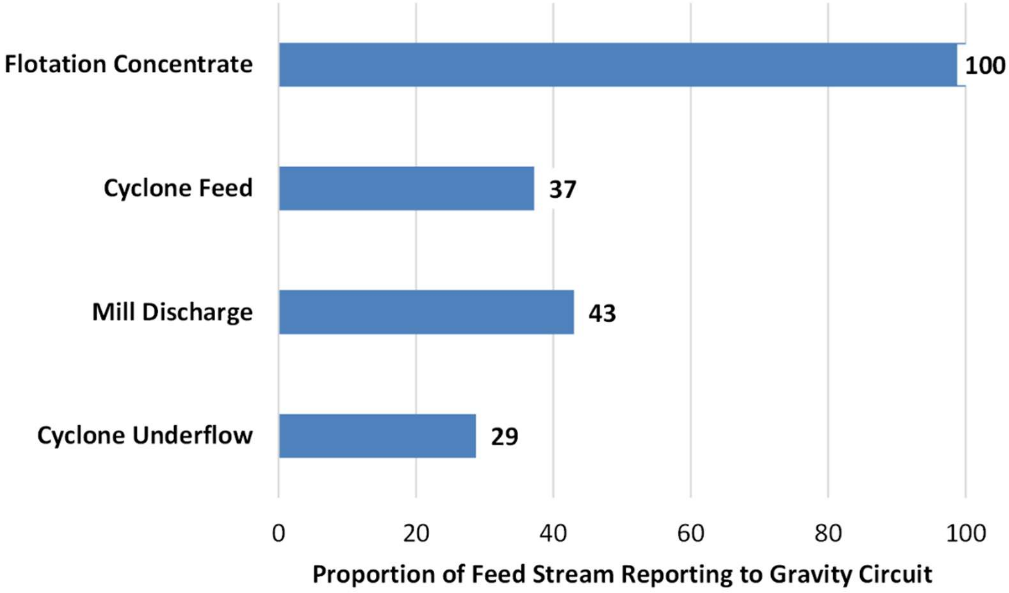


Figure 7: Proportion of Feed Stream Reporting to Gravity

Figure 8 shows the distribution for the number of concentrators installed in each gravity circuit. Nearly two-thirds of the gravity circuits have one concentrator installed and 40% have two concentrators installed.

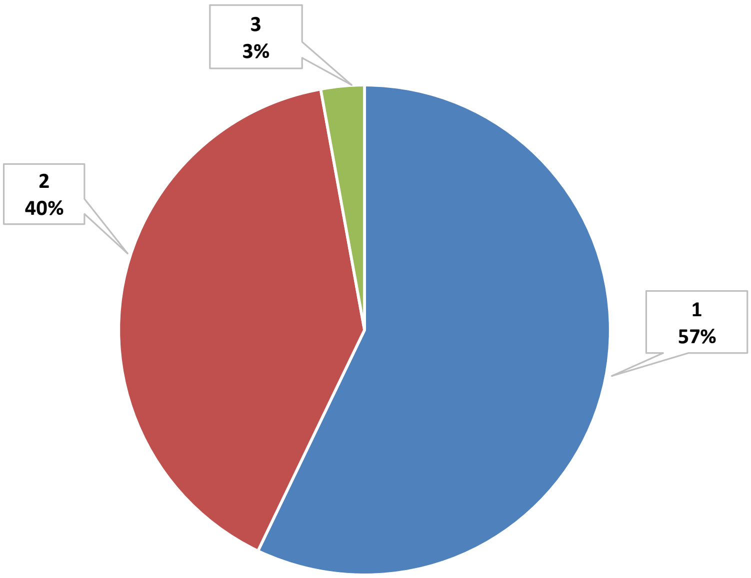


Figure 8: Number of Installed Concentrators in the Circuit

Figure 9 shows the distribution of the operating configuration for gravity circuits with more than one concentrator. Approximately two-thirds of the gravity circuits operate multiple concentrators in parallel.

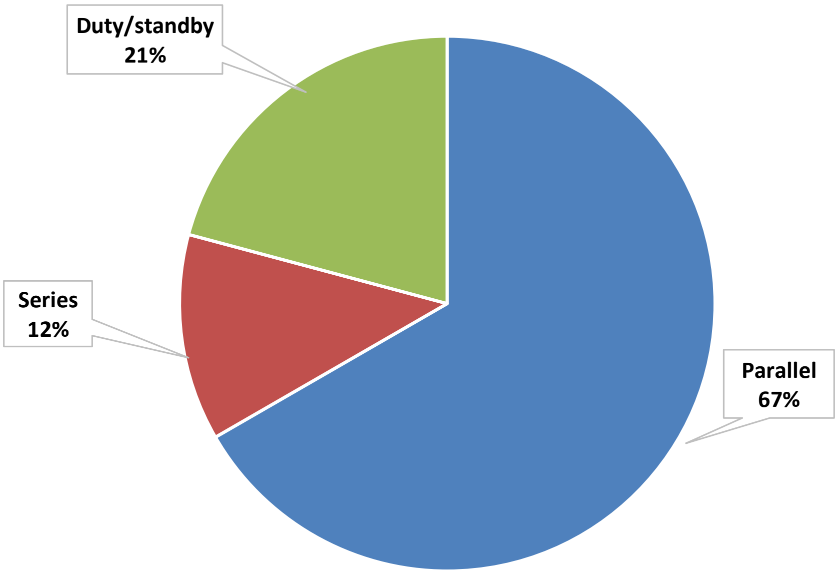


Figure 9: Operating Configuration

Figure 10 shows the distribution of the concentrator feed stream and manufacturer. Knelson concentrators are used to process cyclone underflow, cyclone feed and mill discharge streams. Falcon concentrators are used to process flotation concentrate and cyclone underflow streams.

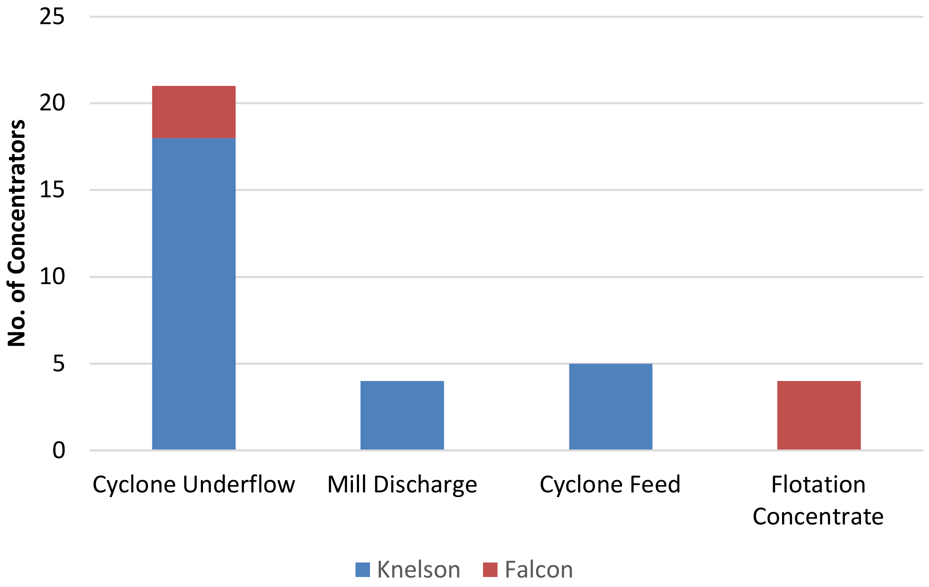


Figure 10: Distribution of Concentrator Feed Stream and Manufacturer

Figure 11 compares the average G-force of the concentrator bowls for the different concentrator manufacturers and feed streams. The Falcon concentrators treating flotation concentrates operated with the highest average G-force and Knelson concentrators processing cyclone feed had the lowest average G-force.

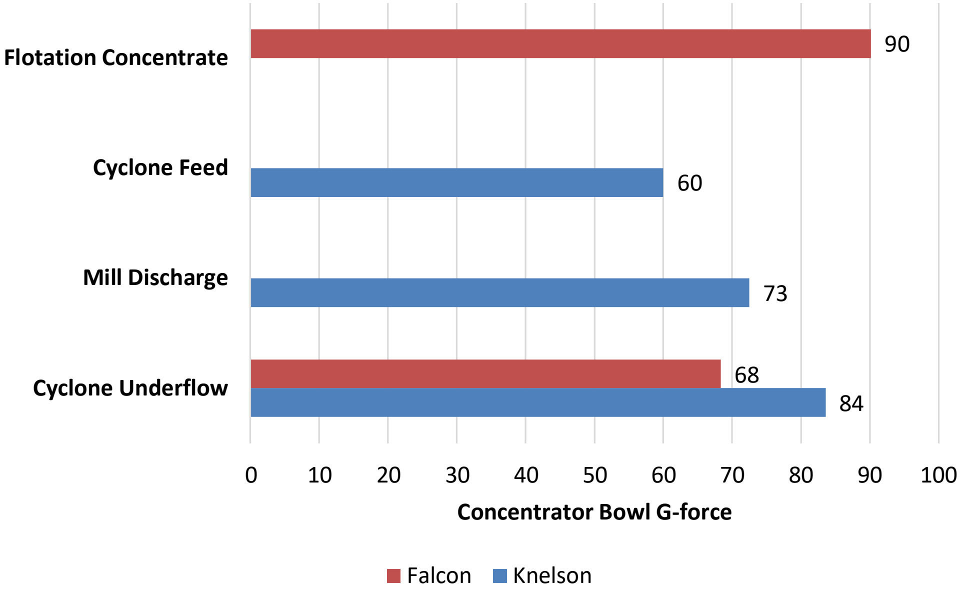


Figure 11: Average Concentrator Bowl G-force Comparison

The distribution of the concentrator cycle times is shown in Figure 12. Except for one site operating with a two-hour cycle time, all of the sites operate the concentrators with cycle times of less than 1 hour.

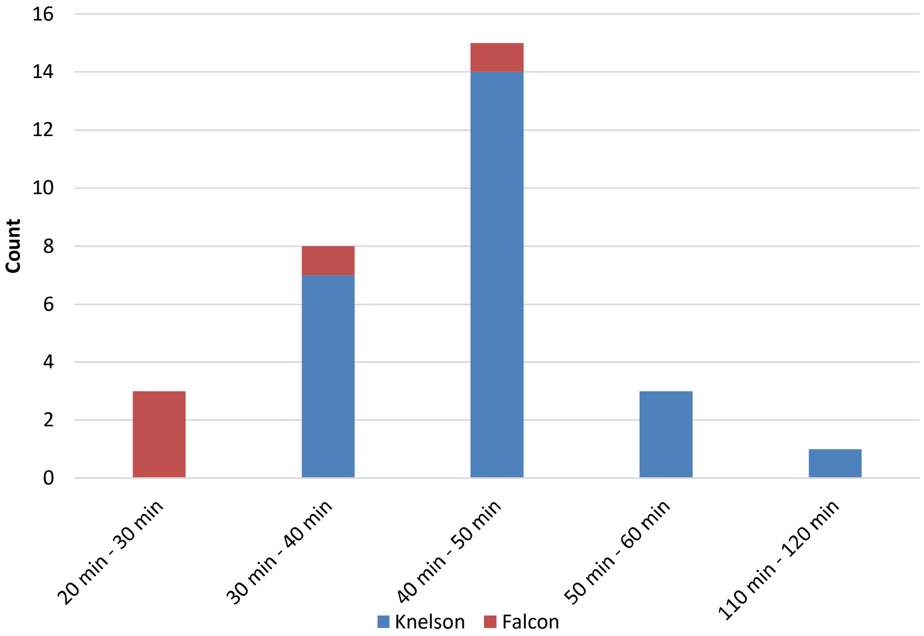


Figure 12: Concentrator Cycle Times

The frequency of bowl inspection for wear is compared in Figure 13. Monthly and weekly inspections make up the majority, accounting for more than two-thirds of the total.

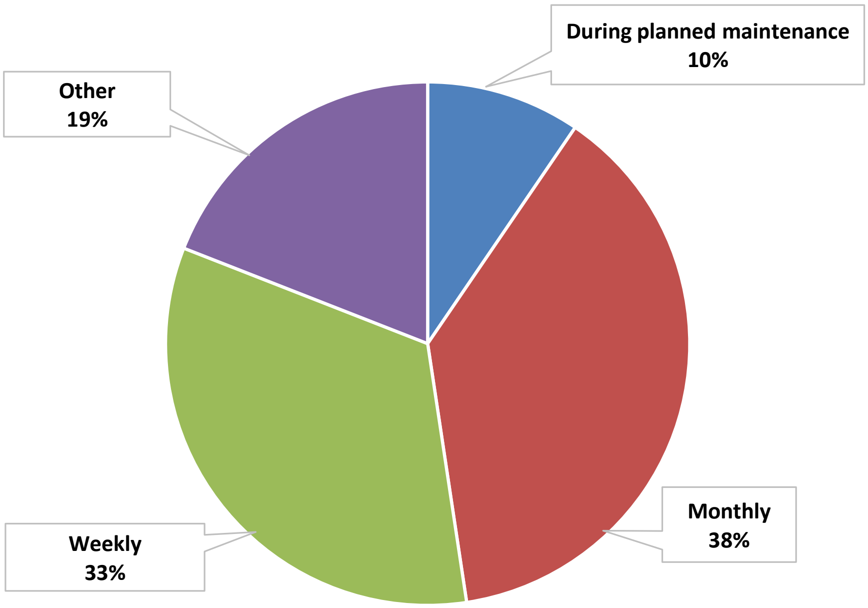


Figure 13: Frequency of Bowl Inspection for Wear

The average concentrator bowl life varied between 3 and 36 months. The frequency distribution for the reported bowl life is shown in Figure 14.

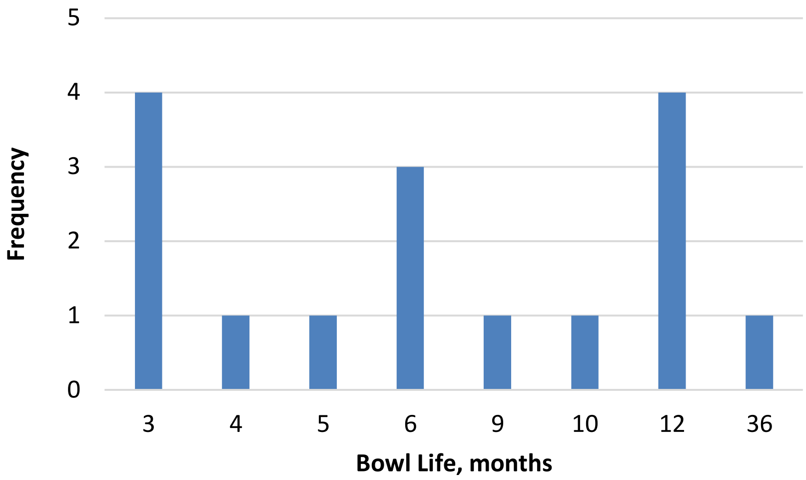


Figure 14: Average Bowl Life

Gold Leaching and Adsorption Circuits

Leaching

The number per site and throughput of the leach circuits varies significantly, from only a single circuit up to three, and from ~250 to +2500 dry tph. Of the 34 responses, CIL is the most common configuration at 56 %, followed by CIP at 35 % and Carousel at 9 %.

As many sites transition to processing more complex ores, including those which contain oxygen consumers, the use of ore-type dependent strategies to ensure sufficient oxygen is present in the leach circuit (and cyanide consumption is minimised) are being more commonly employed. Four sites reported having a dedicated pre-oxidation tank, with dissolved oxygen (DO) levels between 9 ppm and 20 ppm. Seven sites reported adding lead nitrate in the leach circuit. The addition of lead nitrate ranged from 0.03 kg/t to 4.3 kg/t of feed.

Cyanide consumption was tracked in 1999 (Figure 15), 2012 (Figure 16) and 2021 (Table 2) Benchmarking surveys. The trend shows a decrease in cyanide consumption with the automation of cyanide addition in the surveys of 1999 and 2012.

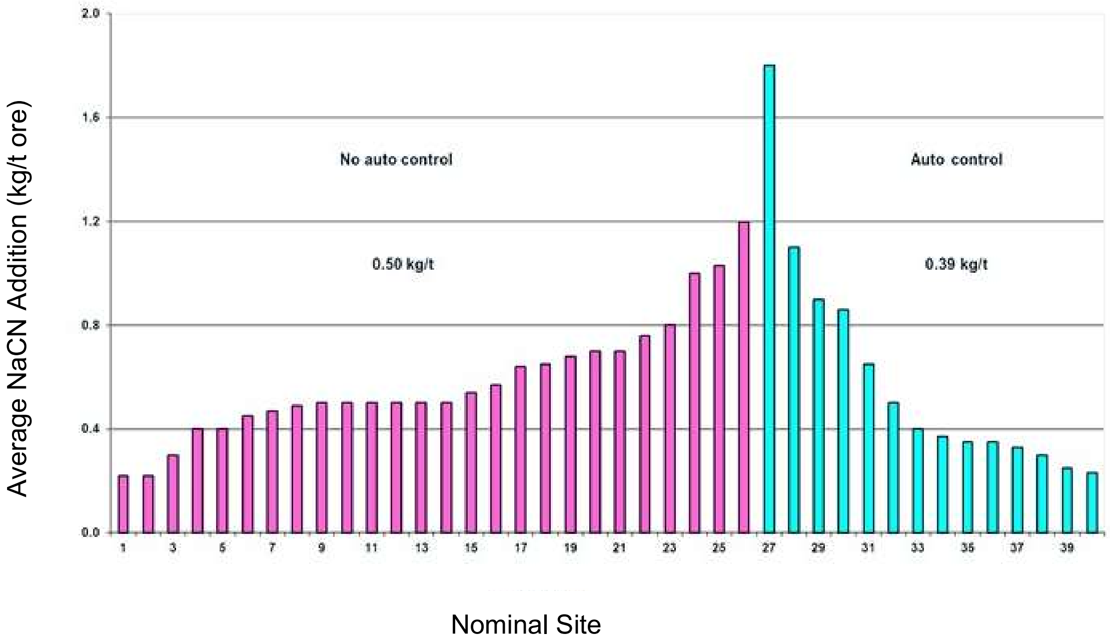


Figure15: Cyanide consumption – P420A Benchmarking survey 1999 [2]

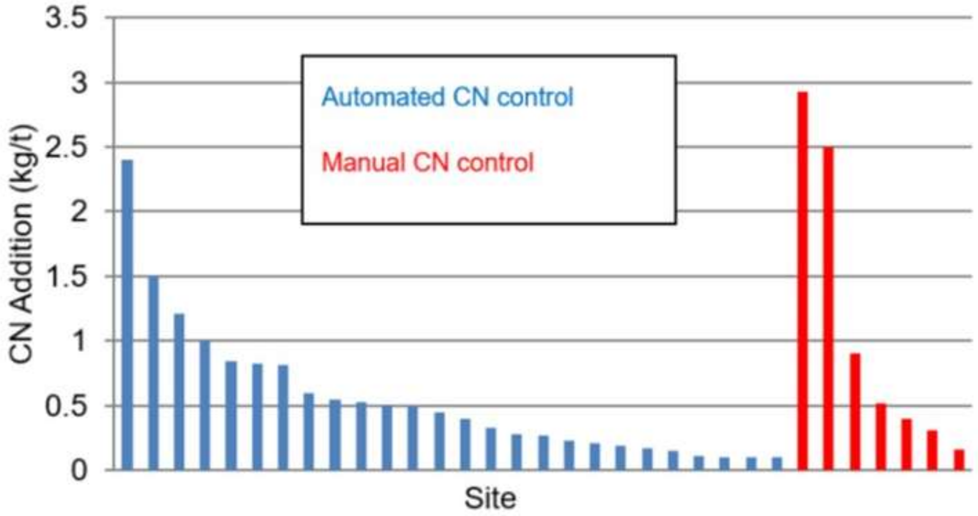


Figure 16: Cyanide consumption – P420D Benchmarking Survey 2012 [3]

The comparisons for cyanide consumption in whole-ore and concentrate leach in the 2021 dataset are given in Table 2.

Table 2: Cyanide Consumption

Cyanide consumption	All Sites Overall Consumption (kg NaCN/t)	Whole Ores Circuits (kg NaCN/t)	Flotation Concentration Circuits (kg NaCN/t)
Average	1.85	0.59	8.77
Median	0.3	0.25	6.8
Minimum	0.12	0.12	0.23
Maximum	17	6.8	17

While it is expected that cyanide consumption for concentrate leaches will be higher than for whole-ore leaches, there are still a surprising number of high cyanide consumers in the whole-ore group. The Gold Technology Group suggests that most of the simple, free-milling ore leaches could be targeting ~0.2 kg NaCN/t so the high numbers seen may be indicative of the data set representing more complex ores (e.g. soluble copper) or lack of cyanide optimisation.

Most sites responding to the 2021 survey are only adding cyanide in a single location, while 36 % are adding cyanide in at least two points within the leach circuit. In theory, multiple addition points should allow for lower cyanide concentrations in the first leach tank with the ability to add cyanide further down the circuit, when and where it is needed for leaching, leading to both lower cyanide consumption and lower WAD/free NaCN in the tails stream, reducing the cost of cyanide destruction where the site has limits on the tailings cyanide concentration.

The cyanide concentration in the leach tanks, according to 2021, is summarised in Table 3.

Table 3: Cyanide Concentration

Minimum Cyanide Concentration	NaCN first tank	NaCN last tank
Average	220	85
Median	120	50
Minimum	10	10
Maximum	1800	600

Six sites reported very low titratable NaCN values of 10 – 40 ppm in the first leach tank and one site reported a very high value of 1800 ppm These results require validation/clarification. The WAD cyanide is monitored at 75% of the responding sites.

In the 2021 survey, all the responding sites reported using automated measurement devices (and manual titration checks), with potentiometric sensors being the most common, having installations at 66 % of sites.

Carbon management

In the 1999 and 2014 surveys, the typical carbon residence time (calculated as = active carbon circuit inventory (t) / elution frequency (t/day) x elution column capacity (t)) was reported as 10-14 days. In the survey of 2021, it was found the average carbon residence time of 16 days, as shown in Table 4.

Table 4: Carbon Residence Time 2021

Residence time	Days
Average	16
Median	13.6
Minimum	3.2
Maximum	70

The maximum residence time of 70 days was observed in a site reporting 2 elutions per week with a significantly high carbon inventory. The minimum carbon residence time was achieved in a site with a high elution frequency and 2 parallel elution columns.

Table 5 shows the typical carbon inventory, in tonnes, at the surveyed circuits in 2012 and 2021.

Table 5: Carbon Inventory

Carbon Inventory (t)	2012 Survey	2021 Survey
Average	83	131
Median	45	113
Minimum	3	25
Maximum	400	368

Longer residence time can lead to very inactive carbon and high gold lock-up in the circuit. Sites with residence times much longer than average generally have a larger inventory than the design criteria. A higher inventory may simply be due to an excess of carbon in the circuit but is more commonly the result of a low carbon activity.

Sites with gold-robbing carbon, or poor carbon activity associated with flotation reagents, may intentionally operate with very low residence times to keep the carbon more active (and loading targets low) to improve circuit performance. Shorter residence times (minimum value of ~3 days noted in each survey) are indicative of low carbon inventory/higher elution frequency and can lead to increased carbon consumption, low gold loading values and higher elution, electrowinning and regeneration costs.

In the previous benchmarking surveys, the median carbon consumption was between 20-30 g/t of ore processed. The 2021 data set (Table 6) has a couple of concentrate leach circuits which have incredibly high carbon consumption which is likely due to the calculation reflecting the very small proportion of the mass treated in the leach circuit (typically only 5-15 % of the mill feed).

Table 6: Carbon Consumption

Carbon Consumption (g C/t of ore)	2012 Survey	2021 Survey
Average	52	139
Median	27	29
Minimum	1	2
Maximum	730	2284

If all reported carbon inventory is taken into consideration, the summary data in Table 6 shows the average would be 139 g/t. The median value of 29 g/t is on the high side of what was considered average from the past two surveys.

Fine carbon is problematic as it can pass through interstage screens and instead of being transferred up the circuit, it will follow the pulp co-currently. Fine carbon, defined as carbon at less than 1.7 mm, can also report to tails carrying appreciable amounts of gold, resulting in gold loss and assay spikes which are challenging during metallurgical accounting. The Gold Technology Group recommends less than 10 % of the carbon inventory should be less than 1.7 mm. In the 2021 benchmark survey, the median bottom size of the carbon fines is reported as 1.18 mm, having a gold grade of 92 g/t.

The Gold Technology Group compile a database of laboratory carbon sizing tests to determine the typical content of fine carbon content, as -1.7+1.18 mm fraction, in circuit carbons (Figure 17). It shows an average of 18 % of fines in the samples tested. This fine carbon (with a bottom size of 1.18 mm) risks wearing finer than the ~800 µm safety screen aperture and being lost to tails.

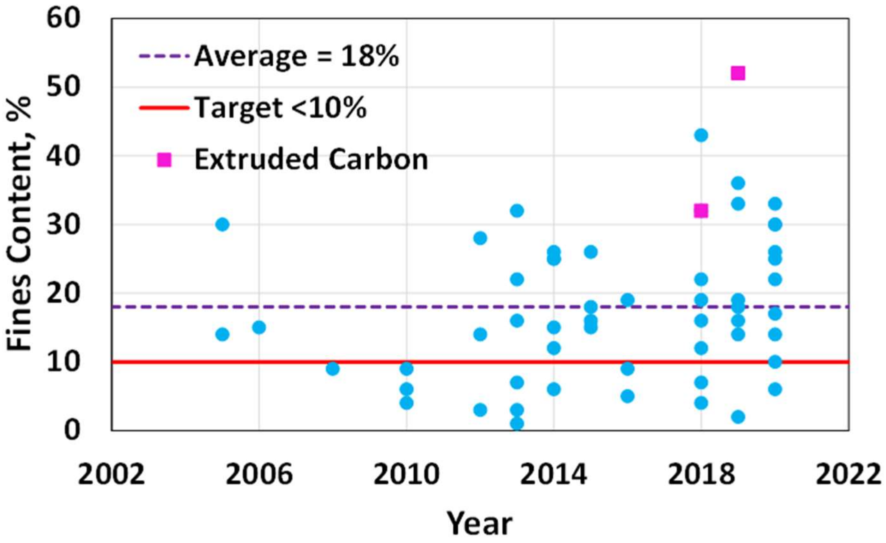


Figure 17: Carbon fines, as – 1.7 mm, content from the Gold Technology Group database (2000-2020) [4]

Solution losses

Since the early days of carbon adsorption circuits, the typical target soluble gold loss for many sites has been 0.01 ppm. In 1999, a wide range of solutions losses was reported with an average of ~0.02 ppm. In 2021, some very high solution losses (~0.3-0.4 ppm) were reported but some very low solution losses were also noted (0.001 ppm). It can be noted that there is concern that these low values may not be accurate as they are likely below the detection limit of the assay method employed. Table 7 shows the average solution loss in 2021 in comparison with 2012. It is still above 0.02 ppm but the median value is closer to the 0.01 ppm target.

Table 7: Solution Losses

Solution loss of Au (ppm)	2021 Survey	2012 Survey
Average	0.023	0.017
Median	0.010	0.011
Minimum	0.001	0.002
Maximum	0.320	0.080

One way to target decreased solution loss is to improve carbon activity and optimise the carbon inventory and distribution throughout the adsorption circuit. Although carbon meters being commercially available to measure carbon concentration in the tanks for many years, only two of the sites reported using the devices in a total of three circuits, with two of those meters only being used to indicate carbon loss through the final interstage screen. All other sites are still relying on manual carbon concentration measurements. Automated carbon measurement has the benefit of not only freeing up the operators' time for more critical duties but also means they may not be required on top of the leach tanks as frequently, minimising HCN exposure.

CONCLUSIONS

This paper summarises the preliminary findings of the Amira P420 G 2021 benchmark surveys and compares the data collected with the surveys from 2012 and 1999. The 2021 survey had a smaller participation of sites, in comparison to the 2012 and 1999. The surveys identified a general increase in throughput and gold production in the sites surveyed. The proportion of gold recovery via gravity also increased in the period observed. In leaching circuits, a significant reduction of cyanide consumption is related to the automation of cyanide addition. The automation of carbon concentration measurement has been reported in two of the sites surveyed. However, the carbon movement sequence has not been reported to be linked to the automated carbon measurement yet. In the Amira P420H project, more work is required to link the data to the previous surveys and correlate survey data to identify the best practices for the operation of gravity and leaching circuits.

ACKNOWLEDGMENT

The authors acknowledge the financial support of the Amira P420H sponsors (AngloGold Ashanti, Australian Gold Reagents, FLSmidth, Gekko Systems, Gold Fields, Kemix, Newcrest Mining, Newmont Corporation, Northern Star Resources, Orica, SGS, B2Gold), Corem, Curtin University.

REFERENCES

1. McGrath, T., Bax, A.R. Process Survey Benchmarking, Part 1 of 2. 2022, WA School of Mines: Minerals, Energy and Chemical Engineering, Curtin University: Perth, Australia.

2. Barbetti, K., O'Leary, S., Staunton, W. P420A Module 2 - A Review of the Amira P420A Gold Industry Survey.2000, A. J. Parker CRC for Hydrometalurgy.

3. Eksteen, J. Progress Report P420D Gold Processing Tec. 2013, A. J. Parker CRC for Hydrometalurgy.

4. McGrath, T., Process Survey Benchmarking, Part 2 of 2. 2022, WA School of Mines: Minerals, Energy and Chemical Engineering, Curtin University: Perth, Australia.

STRIPPING AND ELECTROWINNING OPTIMIZATION IN GOLD AND SILVER PROCESS

By

Göksel Kaygisiz Şendur

Zenit Madencilik San. Ve Tic. A.Ş, Turkey

Presenter and Corresponding author

Göksel Kaygisiz Şendur
g.kaygisiz@zenitmadencilik.com

ABSTRACT

Since the commissioning of the CIL Processing Plant in 2017, Zenit Madencilik has been carrying out surveys in the stripping and electrowinning unit in order to increase the overall Au-Ag recovery, stripping and electrowinning efficiency and reduce the operating cost. For this purpose, the optimization studies on the stripping and electrowinning unit were commenced in 2019, and the outcomes were evaluated after the adequate database was generated.

Since the commissioning of the stripping and electrowinning unit, data collection was started. The barren and pregnant solution samples were collected hourly from the heat exchanger, stripping column and electrowinning cells. During the stripping stage the loaded and barren carbon were sampled and analyzed for Au and Ag.

According to the data collected between 2017-2019, the operating time of the stripping unit was longer and the electrowinning efficiency was lower. Therefore, it was decided that the optimization in the stripping and electrowinning was essential.

The parameters such as operating time, solution flow rate, caustic and cyanide concentrations, stripping vessel pressure and temperature, distance between anode-cathode plates in the electrowinning cells and the cleaning period of the heat exchanger were investigated individually. In consequence, the operating time for stripping was reduced by approximately 33% and the stripping efficiency was increased by 3%.

Key words: *Elution, Stripping, Electrowinning, Optimization, Au-Ag Recovery.*

INTRODUCTION

Stripping of gold and silver from carbon is affected by means of two main processes (AARL and ZADRA). The main rate-enhancing parameter in the stripping of gold and silver is temperature. A secondary rate-enhancing parameter is ionic strength. (1) Stripping is usually conducted within a solution containing sodium cyanide (0.1 – 0.5%) and caustic (1 – 3%) (J.C. Monette, E. J. Fuller, 1992).

Besides these, research on the effect of the process parameters of the Zadra elution process is being conducted in order to optimize the process parameters to minimize energy consumption and lower the operating and design costs of the elution process. Such a move will advance the efficiency of the Zadra elution process (Van Deventer et al., 2003).

Since gold loading on carbon is an equilibrium-driven response, the more effective the stripping, the better the recovery from solution. Maximizing recovery from the activated charcoal is essential to the maximum recovery in the absorption circuit (Michael Drozd, Annual Canadian Mineral Processors Conference, 2008).

Effective stripping depends on optimizing the stripping conditions and maximizing the recovery in electrowinning. Since stripping is an equilibrium response just like loading, less gold is removed as the result grade returning from electrowinning goes up (Michael Drozd, Annual Canadian Mineral Processors Conference, 2008).

In stripping processes, the voltage of the electrolysis cells should be between 2.2 and 2.4 for gold and silver metal. At voltage values above this, other metals in the ore also cling to the cathode plates, causing pollution in the casting (Urbanic, J.E., Jula, R.J., Faulkner, W.D., 1985).

MATERIAL AND METHODS

Material

In the Kızıltepe gold and silver mine owned by Zenit Madencilik, gold and silver recovery is carried out by loading activated carbon. A series of stripping processes related to the efficiency of gold and silver recovery loaded on activated carbon were followed and many parameters were examined.

Method

From the initial examination due to the accumulation of deposits between the heater resistances, there was no solution transfer between the resistances and overheating was detected in the heater tubes. Therefore, the actual temperatures could not be increased. The heaters were disassembled, cleaned between the resistances and reassembled.

Limit temperatures and actual temperatures were provided to read the same values. In this way, the stripping temperature was increased up to the design value of 148°C. In order for the stripping efficiency to be high, the unloaded solution temperature entering the stripping column should be at the maximum level, according to the studies. The accumulation of deposits on the probes of the temperature clocks was largely prevented, and the optimum temperature for stripping was determined. The solution temperature entering the stripping column was increased from 120°C to 135- 140°C by following the temperatures of the heaters.

In the Kızıltepe gold and silver mine, a solution of 12000ppm cyanide and 2-3% caustic was prepared in the unloaded solution tank. The water used while preparing the stripping solution was provided as fresh water. However, the presence of dissolved minerals such as Ca, Mg, Cl in the fresh water used in the field has a corrosive effect on the electrolysis cell plates. The water in which the stripping solution will be prepared has a conductivity degree close to the properties of filtered and pure water, and its corrosive effects on the cell plates will be reduced.

It is thought that the electrical charges on the anode and cathode plates are not constant in terms of not being able to discharge the metal charge. When a load of around 4 volts is given into the cell, only 2.2-2.4 volts current is generated on the plates. Coating the inner surface of the cells to provide insulation will bring a solution.

RESULTS AND DISCUSSION

This study is about increasing the efficiency of the stripping unit in gold and silver mines. In the studies carried out in Balıkesir Kızıltepe gold and silver mine, research was carried out on all active points during stripping.

In the optimization of the stripping unit, firstly, studies related to temperature were carried out, and the graphic related to these studies is given below.

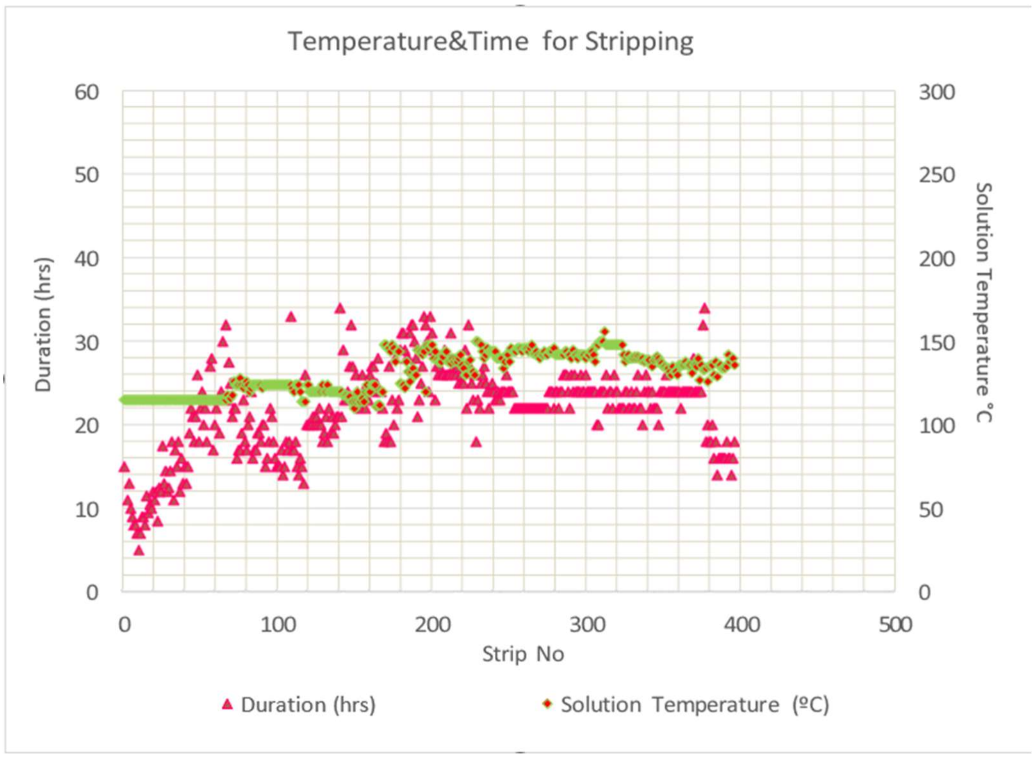


Figure 1 Stripping Duration and Temperature

In Figure 1, the stripping time shortens with the increase in the temperature of the solution entering the pressurized stripping column. First of all, studies were carried out to shorten the stripping time by optimizing this parameter, and the stripping time was shortened.

For each stripping process, a solution of 12000ppm cyanide and 2-3% caustic was prepared in the unloaded solution tank. In the first stripping processes, it was ensured that the cyanide concentration was not reduced below 1000ppm throughout the process. This led to an increase in cyanide consumption. After optimizing the temperature and pressure, studies were started to determine the cyanide consumption, and during the stripping period, cyanide was not added to the unloaded solution tank when the cyanide concentration fell below 1000ppm. Thus, the cyanide consumption in the stripping process is reduced. In addition, the low cyanide concentration throughout the process did not cause any negative effects on metal recovery efficiency or stripping time.

After optimizing the temperature and chemical concentrations, the effects of gold and silver recovery were investigated by performing low voltage electrolysis. The voltage value read on the plates in the cells is measured between 2.2-2.4 V. In this study, the electrolysis cell rectifier voltage was reduced to 2.0 V and the voltage values of the cells were followed during the stripping time. Reducing the EW cell voltage did not have a positive effect on the stripping time or metal recovery efficiency, and when the stripping results were examined, it was seen that the empty carbon results were high. Therefore, the electrolysis cell voltage should be between 2.2 and 2.4 V for gold and silver ore. The results are shown in Table 1.

Table 1 Electrolysis Cell Voltage During Stripping

Time (hrs)	Electrowinning Cell-1 Voltage		Electrowinning Cell-2 Voltage		EW-1 Au(ppm)	EW-2 Ag(ppm)	EW-1 Au(ppm)	EW-2 Ag(ppm)
	Entrance	Exit	Entrance	Exit	Entrance		Exit	
1.00	1.5	1.5	1.5	1.4	77	1130	71	1570
2.00	1.5	1.5	1.5	1.5	70	845	51	592
3.00	1.4	1.1	1.5	1.5	49	240	32	73
4.00	1.4	1.4	1.5	1.5	72	98	13	26
5.00	1.3	1.3	1.5	1.4	32	37	12	17
6.00	1.4	1.5	1.4	1.2	20	18	10	6
7.00	1.1	1.0	1.3	1.1	18	6.8	8.8	5.4
8.00	1.2	1.2	1.1	1.2	16	1.6	8.4	2.2
9.00	1.1	1.2	1.2	1.3	14	1.1	8.1	1.8
10.00	1.1	1.2	1.5	1.6	9.8	0.4	6.9	0.6
11.00	1.7	1.7	1.7	1.7	7.4	0.1	5.8	0.4

Loaded Carbon Au (ppm)	Loaded Carbon Ag (ppm)	Barren Carbon Au (ppm)	Barren Carbon Ag (ppm)
1050	12100	205	3250

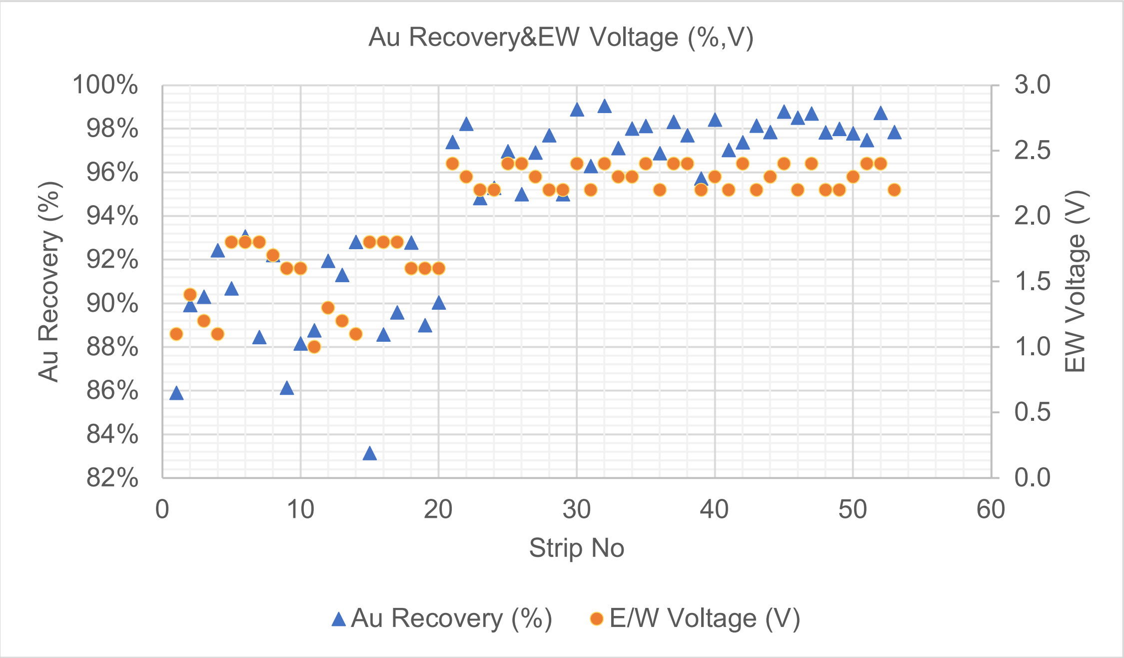


Figure 2 Au Recovery and EW Voltage

When looking at the figure 2, relationship between voltage and gold recovery, the gold recovery decreases as the EW voltage value decreases.

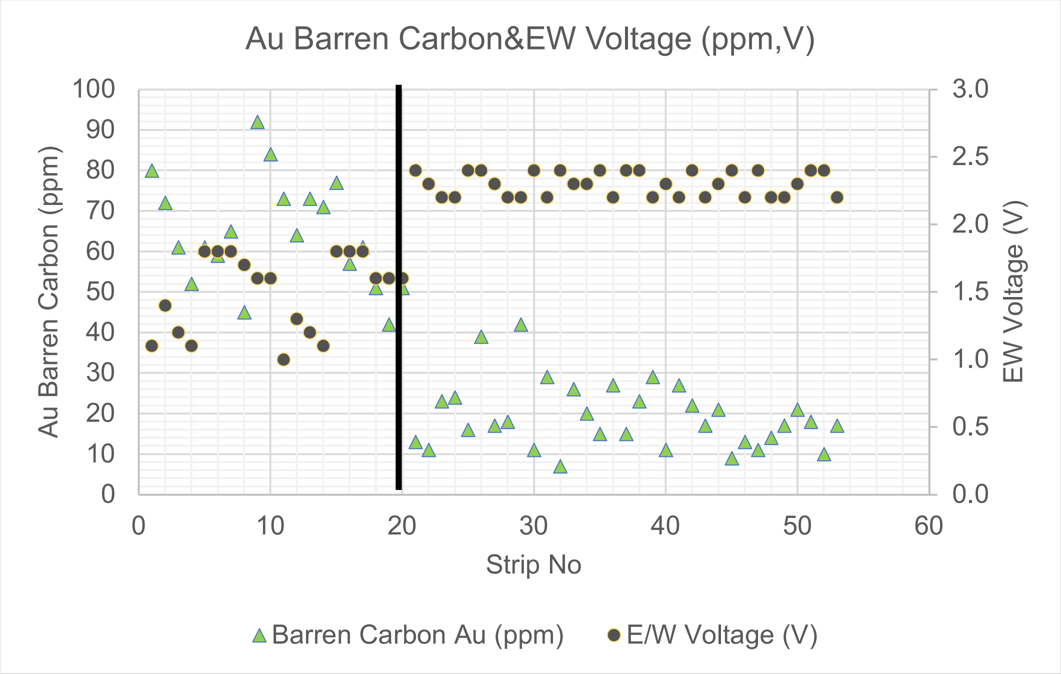


Figure 3 Au Barren Carbon and EW Voltage

When looking at the figure 3, relationship between voltage and Au barren carbon, the EW voltage value decreases, barren carbon for Au increases.

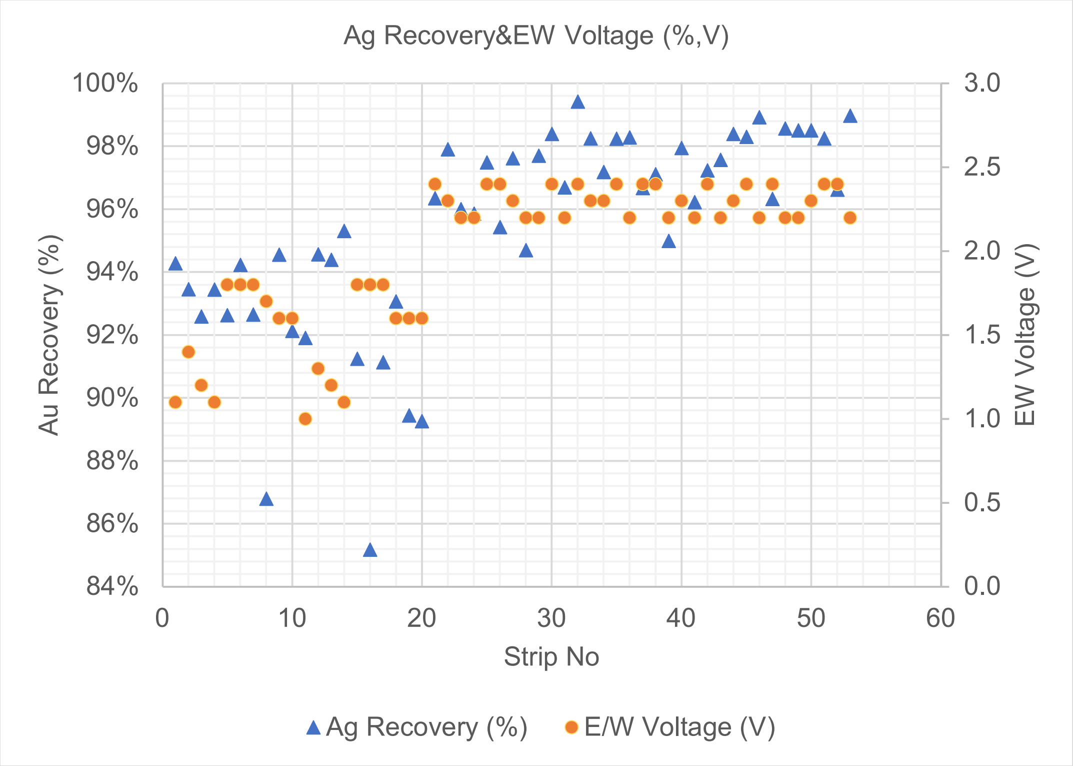


Figure 4 Ag Recovery and EW Voltage

When looking at the figure 4, relationship between voltage and silver recovery, the silver recovery decreases as the EW voltage value decreases.

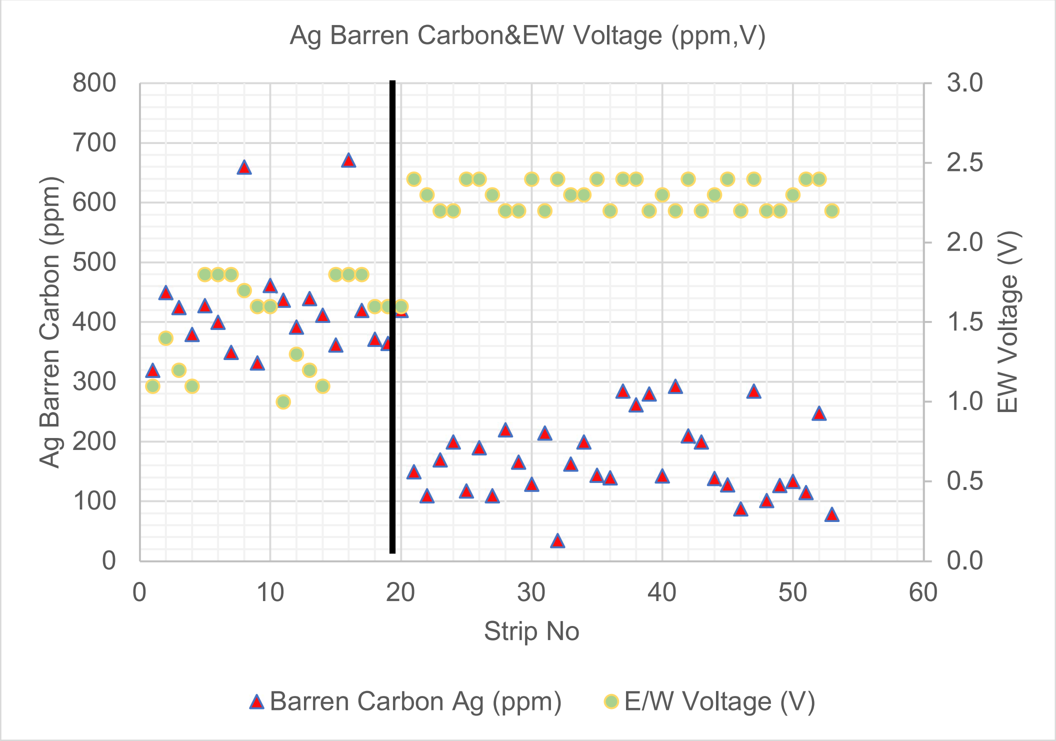


Figure 5 Ag Carbon and EW Voltage

When looking at the figure 5, relationship between voltage and Ag barren carbon, the EW voltage value decreases, barren carbon for Ag increases.

CONCLUSION

Many different tests were carried out by changing one variable determined in the stripping process and keeping other variables constant. As a result of these tests, it has been clearly revealed that all parameters (temperature, flow rate, solution concentration, electrolysis voltage etc.) in the stripping unit have an effect on stripping. When all variables and parameters are examined, the shortening of the stripping time;

- Decreased stripping duration and decreased power consumption.
  Equipment operating times (pump, heaters..) decreased from 24-26 hours to 14-16 hours.
- Decreased chemical consumption, reduced blockage/accumulation in the unit
  The cleaning times of the unit decreased as the precipitation material decreased with the reduction of the caustic concentration.
- Initial stripping solution concentrations should be prepared as 12000ppm NaCN and 2-3% NaOH
- Decreased chemical consumption, increased lifetime of the spares.
  The operation life of the anode cathode is extended.
- Decreased chemical consumption, decreased the operation cost
  Cyanide consumption decreased by approximately 50%.
  Caustic consumption decreased by approximately 40%.
- The temperature of the stripping solution entering the pressure tank should be 135-140°C.
- In cases where the difference between the limit temperatures and the actual temperatures of the heaters exceeds 10 degrees, the heaters should be immediately taken into the cleaning operation without being damaged.
- Electrolysis cells depend on the volt value between the plates being between 2.2-2.4V.

The detailed table of the results is given below.

Table 2 Stripping Optimization Results

Test No	Stripping Duration (hrs)	Loaded Carbon Au (ppm)	Loaded Carbon Ag (ppm)	Barren Carbon Au (ppm)	Barren Carbon Ag (ppm)	Au Recovery (%)	Ag Recovery (%)	Solution Temperature (ºC)	EW Voltage (V)
1	16	900	12600	30	830	93.30%	93.40%	134	2.2
2	14	1095	13000	32	500	94.50%	96.20%	126	2.2
3	16	1185	13350	25	550	88.40%	94.80%	135	2.3
4	16	940	13300	42	475	95.50%	96.40%	136	2.3
5	14	1005	13600	51	866	94.90%	93.60%	136	2.2
6	16	975	13250	31	525	92.60%	94.30%	130	2.2
7	16	870	10600	41	475	95.30%	95.50%	138	2.3
8	14	1030	12600	26	430	89.60%	92.60%	129	2.3
9	16	1310	12900	25	400	94,00%	93.90%	136	2.2
10	16	1250	12100	30	420	83.80%	91.70%	136	2.2
11	16	1603	12715	20	272	97.90%	98.20%	134	2.3
12	16	1123	12490	15	350	79.70%	91.50%	134	2.4
13	16	1147	13160	28	520	82.80%	92.70%	134	2.2
14	16	1400	13600	27	443	98.10%	96.70%	135	2.2
15	16	686	11575	15	1060	97.80%	90.80%	142	2.2
16	16	820	12700	39	534	95.20%	95.80%	138	2.2
17	14	738	11025	32	600	93.60%	90.50%	140	2.3
18	16	965	13250	15	410	98.40%	93.10%	140	2.2
19	16	1065	11900	21	470	98.00%	96.10%	136	2.2
20	16	1093	12500	15	480	98.60%	96.20%	146	2.2
21	16	1200	13300	25	380	92.70%	94.90%	146	2.3
22	16	1180	11350	18	445	98.50%	96.10%	146	2.3

REFERENCES

1. J.C. Monette, E. J. Fuller, SME Annual Meeting (1992) “Plant experiences with the use of a novel chemistry to strip gold and silver from carbon”,

2. Van Deventer, J. S. J., & Liebenberg, S. P. (2003). “The role of cyanide in the multicomponent elution of cyanide complexes from activated carbon. The European Journal of Mineral Processing and Environmental Protection”, 3(3), 1303-1868.

3. Michael Drozd, Annual Canadian Mineral Processors Conference, (2008) “Activated carbon’s use in gold adsorption and recovery”, 491-493.

4. Michael Drozd, Annual Canadian Mineral Processors Conference, (2008) “Activated carbon’s use in gold adsorption and recovery”, 491-493.

5. Urbanic, J.E., Jula, R.J., Faulkner, W.D., 1985. Regeneration of activated carbon used for recovery of gold. Miner. Metall. Process., November, 193 – 198.

METSO OUTOTEC FLOWBOTTOM, FIRST REFERENCE EXPERIENCES AND LOOK BACK TO DEVELOPMENT

By

[1]Tuomas Hirsi, [2]Lauri Mäenpää, [3]Jari Peltola

[1]Metso Outotec Finland Oy, Finland
[2]Metso Outotec Finland Oy, Finland
[3]Metso Outotec Finland Oy, Finland

Presenter and Corresponding Author

Tuomas Hirsi
Tuomas.Hirsi@mogroup.com

ABSTRACT

Gold Cyanide Leaching agitators have been fairly standard for recent decades with only incremental changes. Good mixing is an important factor in gold cyanide leaching as the reaction kinetics are often controlled by mass transfer. In order for the leaching to proceed effectively, the solids should maximize the contact area with the solution, thus requiring complete solids suspension. This occurs when there is no solid settling nor solids bed formation at the bottom of the tank with all solids suspended. In optimal CIL configuration the agitation is optimized to achieve this solids suspension and required gas dispersion with lowest possible power consumption. Metso Outotec has developed new solution to optimize energy consumption and carbon attrition.

In first FlowBottom reference site Metso Outotec set out to solve a problem, where processing of high-grade gold concentrate resulted in recurring solids settling in the leaching tanks at the customer site. Inadequate suspension of solids lead to reduced capacity, more difficult adjustment of the slurry density and downtime of the tanks increased considerably.

Investigation of a solution started with authentic solids tested at research center at Pori, Finland and the tests confirmed the sanding issues with the existing agitation configuration providing reference point. An improved agitation solution was tested and tailored for the leaching tanks, which consisted of dual OKTOP®3300 impellers, FlowBottom element, and a SandSense measuring system for agitation optimization. The performance of this configuration was validated at the site and it was confirmed that the solution was able to completely suspend the solids without need for agitation power increase.

After more than 7 months of the upgrade with FlowBottom and SandSense, the tanks have not experienced downtime due to solids settling and utilization of their full effective volume has meant that the capacity of the tanks has increased up to 16%. In this work we will explain the functionality of FlowBottom, SandSense and achieved benefits in Svartliden and estimate what kind of further benefits it can provide in CIL process.

Keywords: CIL, CIP, OKTOP, FlowBottom, SandSense, Gold, Sanding, Solid suspension, Energy savings

INTRODUCTION

Agitators used in Gold Cyanide Leaching, in both CIL (Carbon in Leach) and CIP (Carbon in Pulp) reactors have been fairly standard for recent decades. There has been some incremental changes in the design of the agitators, mainly to bring down cost and improve the structural integrity of the agitators. However, the used mixing power (kW/m^3) and the generic layout of the agitators has been quite standard varying from 1 kW/m^3 in smaller CIL reactors to below 0.05 kW/m^3 in larger reactors. The reactor size normally varies between 10 to 15 meters of diameter, but smaller and larger reactors are also in use. A vertically operated agitator is generally used with a standard layout of two impellers. The diameter to height ratio with a tank is usually 1:1 or 1:1.1. Used impeller type is typically two-level hydrofoil agitator, where upper impeller might be upwards or downwards pumping. The Metso Outotec FlowBottom solution for more efficient CIL processing was developed and published first time in 2015. The FlowBottom is a bottom structure that enables mixing of CIL reactor with one impeller and considerably lower power requirement compared to conventional solutions. (1, 2).

The development of the FlowBottom was carried out with co-operation of CFD testing and laboratory testing. CFD modelling has also been shown outside of Metso Outotec research to provide good basis for development of CIL reactors agitation. The modelling and testing were to validate that FlowBottom can provide estimated solid suspension results, but also has no negative effect on the gas dispersion. Also, the effect of reduced power to carbon attrition was tested in laboratory environment. In Figure 1 it is shown the result of modelling tank operation with flat bottom tank and tank with FlowBottom(1, 2). In Figure 1 it can be seen significant flow rate increase and better suspension in FlowBottom than in flat bottom reactor in the same conditions.

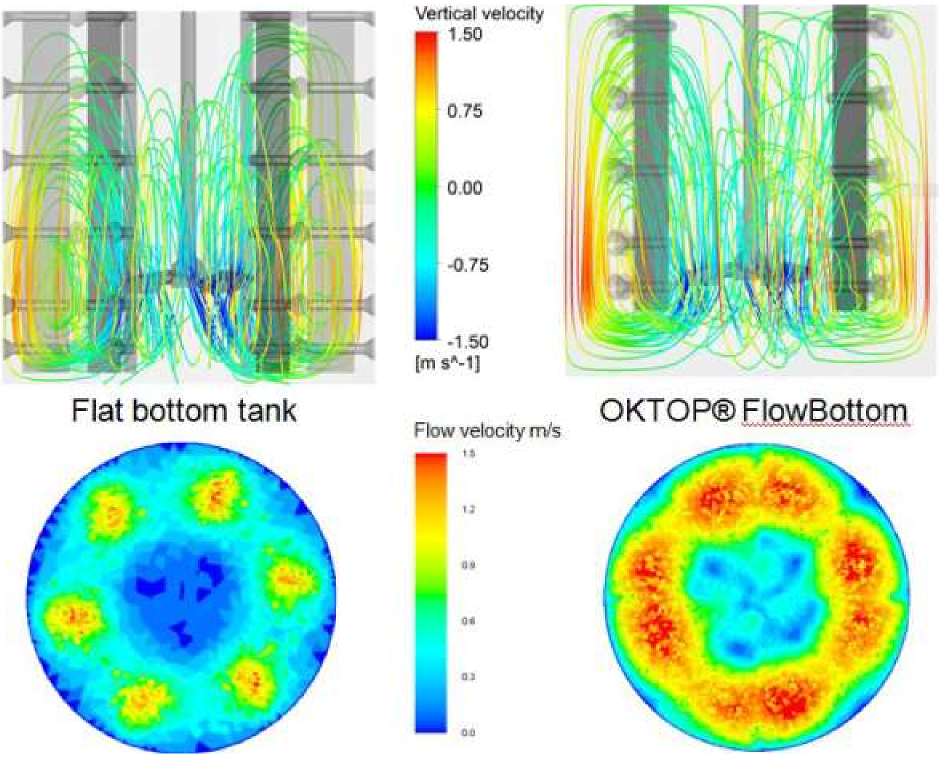


Figure 1. CFD comparison showing vertical fluid velocities and fluid velocities at 85 mm distance from the tank bottom with flat bottom and OKTOP® FlowBottom structures at the same impeller rotation speed. The cylindrical reactor has a diameter and solution surface height of 8.5 m and is agitated with an OKTOP®3200 hydrofoil impeller. (3)

SVARTLIDEN REFERENCE, PROBLEM DESCRIPTION

The leaching circuit in Svartliden plant is composed of two pcs of leaching reactors and five pcs of carbon in leach reactors. These tanks had been experiencing quite serious sanding issues with solid particles accumulating inside the tank. During longer periods of operation, the tanks required cleaning to remove solids accumulation on the bottom. This accumulation leads to loss of operational volume inside the tank and with measurements it was estimated that 16% of the volume of each tank was lost due this issue. Sanding indicates that the agitation inside the tanks is insufficient.

Good mixing in the CIL reactor is essential to provide efficient mass transfer, gas dispersion and oxygen utilization, but to also provide good flow from one reactor to other. In the CIL reactor solids are transferred through intermediate screens that reside at the top section of the tank. If the solid suspension level in the tank is not uniform, larger particles do not rise to the top of the tank and are not transferred to the next reactor. This will cause in long run sanding on the reactor and process problems.

This problem could have been solved with replacing agitators with more power to ensure solids suspension and adequate mixing. However, increase in the motor power was not allowed due to the structural limitations set by the current agitator support structure and tank. This required other solution to be found for the problem.

PROPOSED METSO OUTOTEC SOLUTION

To solve the problem Metso Outotec proposed use of Metso Outotec FlowBottom. This novel new technology did not have any existing references but had been thoroughly tested in our research center in various scales. In this case the benefit of the FlowBottom is that it does not need any additional mixing power but can provide big increase on the solid suspension properties of the agitation.

Laboratory testing of the proposed solution for Svartliden was carried out in the Pori Research Center in Finland. The testing was done with actual material from the site and included solid suspension testing with current agitation in the laboratory scale as well as with FlowBottom solution in the laboratory scale. In addition to the solid suspension test the effect on the carbon attrition was tested. The proposed solution was also calculated with CFD model (Figure 2) to see how the mixing would perform in the industrial case. The final results of testing were that in full suspension conditions the FlowBottom could lead to 65.3% reduction in power consumption and 60.3% reduction in carbon attrition taken place in CIL reactor.

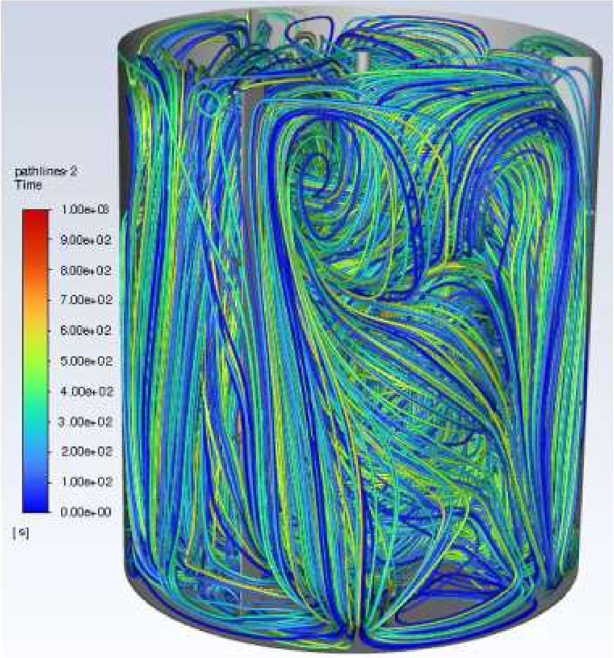


Figure 2. Flow path lines in the reactor with CFD modelling and FlowBottom in place.

The final selected agitation is combination of 2 pcs of Metso Outotec OKTOP 3300 impellers with FlowBottom. Due this being the first reference for Metso Outotec FlowBottom, and the above 1:1.1 aspect ratio, it was decided that 1 impeller to not be used here and 2 impeller levels was utilized.

PROVING THE SOLUTION

Testing campaign

The important factors to be tested during the test campaign were:

- Complete solids suspension with no sanding occurring.
- Uniform suspension of the particles so that there is no difference in solids % or particle size distribution in different heights on the tank.
- Electrical consumption of the agitator when flow bottom is used.

The testing campaign on the site was performed in co-operation with the Svartliden mine personnel. The testing campaign lasted for one year with the main target of proving that FlowBottom works in varying conditions on site during long operational period. In addition to the operational testing there were 2 visits from personnel of Metso Outotec Research center from Pori. During these visits more focused testing on the product development was done.

Temperature measurements

During the operation of the sanded tanks the site personnel noticed it was very straightforward to measure the sanding inside the tanks by measuring the temperature of the wall with handheld IR-camera. This showed quite clearly where the solid bed started inside the tank.

When a solid bed forms at the bottom of the reactor it can act as a thermal insulator compared to freely moving slurry for which convective heat transfer can distribute the heat much more efficiently. Outside the tank this can be seen as a higher wall temperature at sections where the slurry can move.

Figure 3 shows the measured wall temperature profiles before and after the agitator upgrade. A clear transition can be noticed in the prior upgrade-profile at approximate height of 1.4 m from the bottom. This level gives a good indicator of the solid bed height near the wall. On the other hand, the profile after the upgrade shows only a minor fluctuation around average temperature, which is a sign of effective heat transfer even at the bottom of tank, and therefore absence of a settled bed.

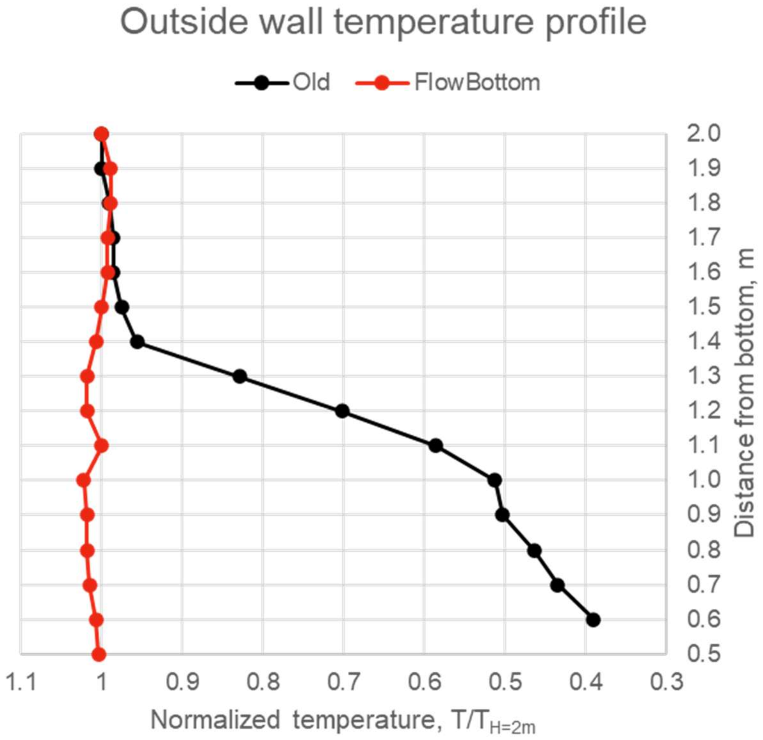


Figure 3. Normalized temperature (T/TH=2 m) with respect to measurement height.

According to the experience of the site personnel, the settled bed takes a “bowl”-shaped form due to the flow pattern created by the agitator. Considering this shape and the height of the bed near the wall, an estimate for the volume occupied by the bed can be calculated. Assuming the height of bed at the center to be about half of the height at the wall, the volume of this disc with a concave side is approximately 16% of the effective volume of the reactor. (Figure 4).

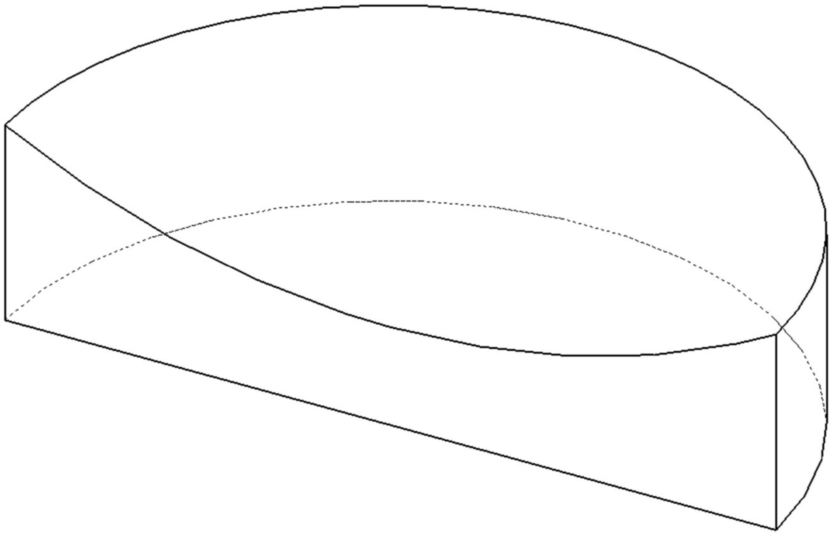
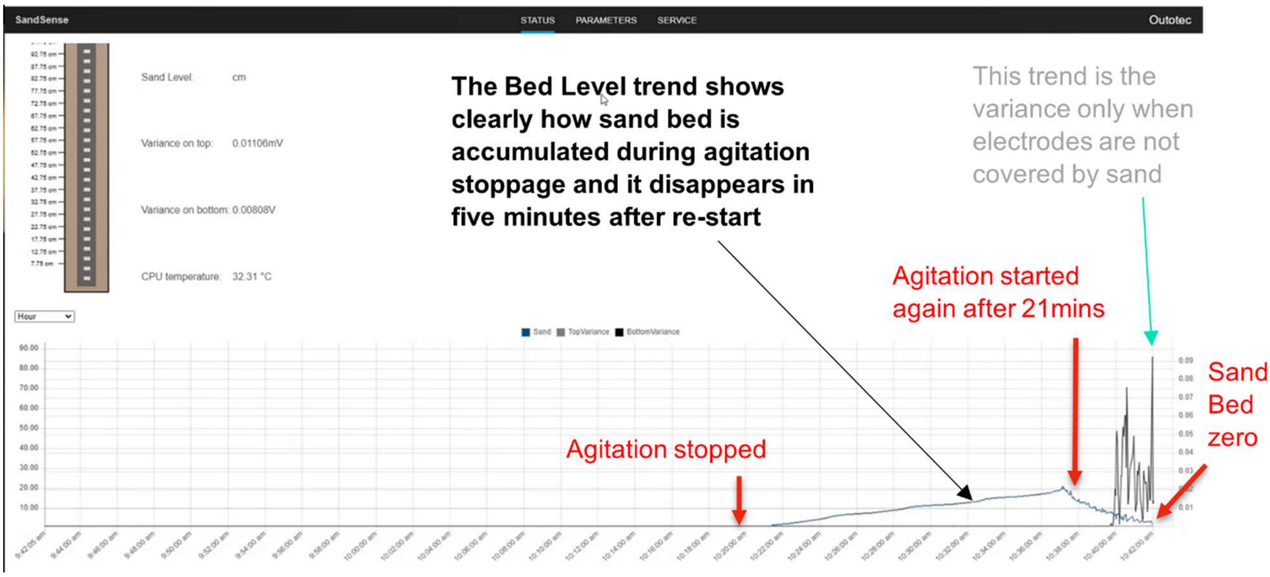


Figure 4. Illustration of the solid bed inside the tank.

SandSense

The operation was continuously followed by Metso Outotec SandSense instrument. This instrument is a proprietary industrial scale method to follow solids accumulation on the bottom of the tank in real time and is located on the side of the tank which has been shown in laboratory test to be first place to sand in both conventional and FlowBottom agitation. SandSense instrument is based on electrical resistance measurement. SandSense can measure up to 1 meter from the bottom of the tank with high accuracy.

There was no sand bed accumulated during normal run with agitator speed from 50% to 100%. To ensure the reliability of the instrument, a "stop-and-go" test was carried out. The agitator was stopped and settling was allowed to occur. This was detected by SandSense as the reading increased slowly when particles accumulated to the bottom of the reactor (Figure 5). The disappearance of the bed was also clearly visible in the trend when the agitation was restarted. Between the solid bed and the liquid there was a clear difference in conductivity in which the measurement is based on. The SandSenses were equipped with remote connection possibility and commissioning & testing was done on remote mode with live process operation, illustrating the flexibility and repeatability of this testing method. (4)



Sand Level: cm

Variance on top: 0.01106mV

Variance on bottom: 0.00808V

CPU temperature: 32.31 °C

The Bed Level trend shows clearly how sand bed is accumulated during agitation stoppage and it disappears in five minutes after re-start

This trend is the variance only when electrodes are not covered by sand

Agitation started again after 21mins

Agitation stopped

Sand Bed zero

Figure 5. M The trend covering a "stop-and-go" test. View from the SandSense Status page.

Particle size tests

During the trial period, the uniformity of the solid distribution was tested by collecting slurry samples at different depths of the reactor. This was particularly important for prolonging the operating time as the slurry outflow was via overflow and necessitated that also the coarse particles should reach the outlet.

The samples were collected with a dip sampler that allowed the slurry samples to be collected at different depths. The device consists of a spring-loaded cap for the container, which can be opened by pulling the string attached to the cap. With the help of extension pipes the sampler can be guided to the desired depth and operated from the top of the tank. Photos of the device are shown in Figure 6. After collecting the samples, the slurry was filtrated and washed and shipped to Pori research center, where the particle size distribution was determined by laser diffraction method.

Figure 6. Dip sampler used for the sample collection.

Results of the particle size analyses are shown in Figure 7. As seen from the figure the trends are almost horizontal, meaning that there is little difference between the samples at different depths. This was an important finding, especially for the coarsest fraction as it meant that no accumulation was expected even after a longer operation time. The results of high degree slurry uniformity were also supported by the fact that the plant was able to operate the reactors without any mixing related issues for the entire trial period.

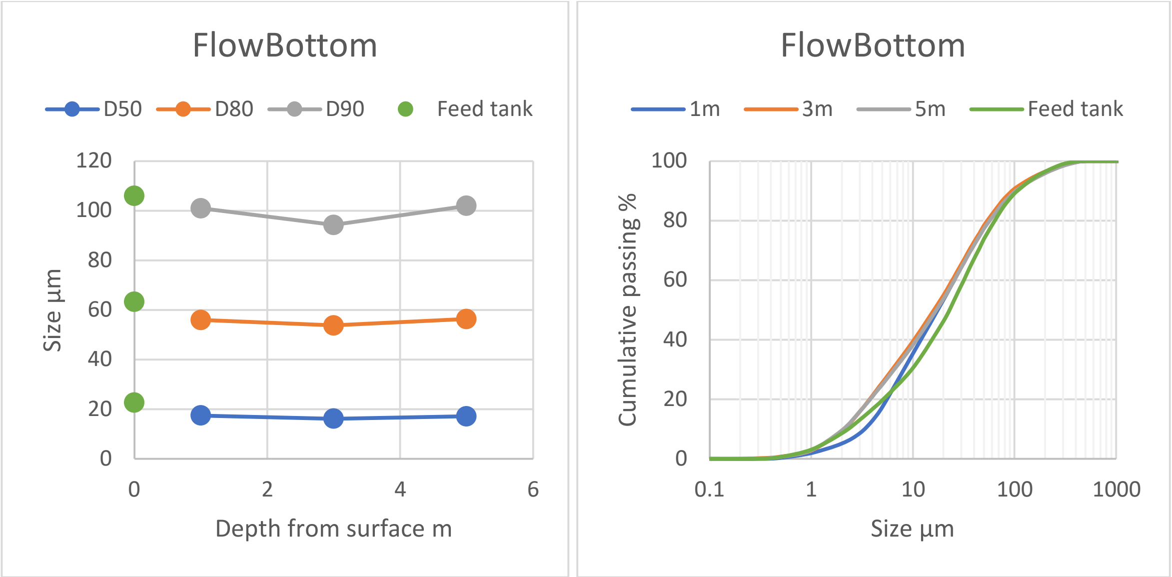


Figure 7. PSD results for the upgraded reactor. The feed tank sample was collected from an upstream tank to provide a reference for the solids.

Electrical consumption

Total electrical consumption was observed from the frequency converter on site when operating the equipment. From this it is possible to further calculate by taking in accordance the electrical losses what is the absorbed power of the agitator when utilizing it with FlowBottom and whether there are differences to the scaled-up laboratory values. Testing of the electrical consumption was done with multiple different rotational speeds to check the whole operational range of the agitator.

Agreement with the laboratory values was on a good level and supported the finding that the upgraded agitator would run at a lower power draw compared to old installation. After the electricity consumption tests, the agitation was adjusted to adequate levels of agitation ensuring solid suspension and good surface movement. This agitation rate was later confirmed with PSD samples (see previous chapter) to provide near uniform solids distribution. With the mixing speed confirmed the longer testing period started, during which agitation rate was kept nearly constant.

At time of testing the power consumption resulted approximately to 29.9% power reduction compared to the original. Considering the upgraded agitation system was able to resolve the heavy sanding issue and still reduce the electricity consumption that highlights the impact of FlowBottom. Annually, this means a power saving of approximately 21 MWh, which also corresponds to a reduction of CO_2 in the range of 4.2-8.4 t/year (200-400 kg/MWh) depending on the energy source, if the electricity is produced with fossil fuels. (6)

Further work

The benefits of lowering the mixing power are not limited to electricity consumption. Reduction of energy input also has a positive impact on the reduction of carbon attrition in the adsorption tanks. This in turn can provide additional savings from reduced carbon consumption, and more importantly gold losses via carbon fines.

Gold losses due to carbon attrition happen when carbon particles, loaded with gold, are broken into a smaller size than the interstage screen size allowing them to pass to the tailings. Carbon fines also have a high specific surface area making them efficient in adsorbing and increasing their gold content. The savings potential for gold loss reduction can be a significant part for the total savings in agitation optimization. Focusing on minimizing the losses at the final stages of the recovery also means the efforts for extracting the gold in previous processing stages are not wasted. (3, 5)

Proving reduction of carbon attrition with lower mixing energy is straight forward in batch tests at laboratory scale. However, quantification of carbon attrition reduction is very difficult in a continuous large-scale process, as it requires very stable operating conditions on long testing periods and is not only affected by the mixing at the adsorption circuit. The reduced mixing intensity also brings benefits to maintenance. Lower energy utilized for agitation leads to less wear on the agitators and longer operation time. This can be considerably saving in cases were feed material has high wear properties. Maintenance benefits can only be observed over longer time and will be a purpose of further work.

CONCLUSIONS

Site validation tests, as well as the smoothly went long trial period, confirmed the successful agitator upgrade. The main problem of sanding was resolved with a tailored mixing solution that provided also near-uniform slurry homogeneity at reduced power draw. The disappearance of the settled bed meant that the reactor could utilize its full capacity, increasing the effective volume up to 16% and reducing the operational downtime. The increase of the effective volume compared to the old installation effectively means the increased volume is one extra tank compared to the old installation and this will lead to improved gold recovery.

Testing was performed with multiple methods to ensure that all the aspects of required agitation were taken in consideration. SandSense was working as designed and it was tested for both – LT1 & LT 2 – reactors with good results and further illustrates the benefit of using SandSense instrument in optimizing and monitoring process performance.

Now that the solid suspension performance of the FlowBottom has been proven further work is still required to show the additional benefits. The savings on operational costs via decreased gold and carbon losses and reduced maintenance needs can bring even further benefits. This work will continue further as more operational time with FlowBottom is accrued.

ACKNOWLEDGMENTS

Metso Outotec would like to thank you Dragon Mining staff and management for excellent co-operation during the testing campaign and important insights to the analyze of the results and functionality of the equipment.

REFERENCES

1. Latva-Kokko, M, Hirsi, T., Ritasalo, T. & Tiihonen, J. (2015), Improving the process performance of gold cyanide
leaching reactors, World Gold Conference 2015 – Proceedings, The Southern African Institute of Mining
and Metallurgy, pp. 183-193

2. Divyamaan Wadnerkar, Vishnu K. Pareek and Ranjeet P. Utikar, CFD Modelling of Flow and Solids Distribution in Carbon-in-Leach Tank, Metals Journal 2015

3. M. Latva-Kokko, J. Kourunen and K. Lillkung, New Ways To Reduce Carbon-Related Gold Losses In Cil And Cip Processes, Worl Gold 2017.

4. Kourunen, J. (2014), Imaging of mixing in selected industrial processes using electrical resistance tomography, Academic Dissertation

5. M. Latva-Kokko, J. Kourunen and K. Lillkung, (2017), New ways to reduce carbon-related gold losses in CIL and CIP processes, COM2017 | The conference of metallurgist hosting World Gold & Nickel Cobalt

6. Carbon dioxide emissions factor (ourworldindata.org) https://ourworldindata.org/grapher/carbon-dioxide-emissions-factor, accessed 24.03.2023.

SOLVING SCALE PROBLEM IN PROCESS TANKS WITH SWIRL FLOW AGITATION

By

[1]Bon Nguyen, [1]Lachlan Graham, [1]Michael Hurley, [1]Greg Short, [1] Shen Long, [1]Dean Harris, [1]Yuqing Feng,
[1]Jie Wu, [2]Andrew Nash

[1]CSIRO Mineral Resources
Building 121, Bayview Avenue/Research Way, Clayton, Victoria 3168, Australia

[2] Agnico Eagle Australia – Fosterville Mine
McCormicks Rd, Fosterville, VIC, 3557, Australia

Presenter and Corresponding Author

Bon Nguyen
Bon.Nguyen@csiro.au

ABSTRACT

Scale deposition is an on-going problem in many hydrometallurgical processes. Examples include neutralization processes for acidic tailings treatment in the gold, zinc and nickel industries, precipitation in the alumina Bayer process and crystallisation in the lithium carbonate process. Scaling leads to blockages in pipes, in slurry transfer launders and jamming of valves. Scale deposition on flow instruments can also lead to difficulties in monitoring and controlling the plant operation. Scale in tanks and on impellers can lead to loss of effective tank volume, inhibition of heat transfer, and more severely, can lead to tank stoppage due to bogging of the agitator. In some large-scale tanks, scale formation leads to deformation of the tank shells and bending of the agitator blades and shaft. Clean-up of scale lumps is not only time consuming, but it can also pose significant OH&S risk, for example, from the fall of large lumps of scale which can cause human injury and damage to equipment.

To address the challenges caused by the scale problem in the hydrometallurgical process, CSIRO Fluids laboratory has been undertaking research and development since early 2010 to solve this problem via design innovation, in close collaboration with the technical staff at alumina and gold processing plants. This research program has led to the successful application of CSIRO SWIRLFLOW® for solving scale problem in large-scale tanks. In this paper we will present two case studies demonstrating how to reduce the scale problem in the mixing tanks in a gold neutralisation tank and in alumina precipitation tanks. We will also present our latest laboratory scale modelling method involving a Gypsum scale system mimicking scale formation in plant scale tanks.

Keywords:
Scale, Tanks, Agitation, Swirl Flow/SWIRLFLOW ®. Crystallisation, Neutralisation.

INTRODUCTION

The unwanted deposition of solids on the tank wall or tank internal structure is normally referred as scale, and is an ongoing issue for hydrometallurgical processes distinct from settling solids. It has been well described by the operators at a number of refinery processes including gold and iron ore refineries that scale formed on their mixing tanks has caused many operational issues. Particularly, scale build-up on the walls of precipitation tanks has been noted to result in problems such as increased power consumption, tank deformation due to the additional weight, and sedimentation reducing the active working volume of the tanks[1]. If a significant amount of the active volume of the tank is occupied by scale, the residence time distribution (RTD) may also be compromised, which can affect the quality of the final product. These issues increase running costs, damage the structural integrity of the tank and often lead to increased down time, as the tanks need to be taken offline regularly for cleaning. Figure 1 shows precipitation tanks fitted with draft tube agitators. The volume of these tanks is typically in the range of 3500 – 5000 m^3.

The focus of this paper is on gaining further understanding of scale formation using physical modelling techniques, as well as exploring alternatives for scale reduction, such as using SWIRLFLOW® technology instead of conventional mixing systems. The hypothesis is that SWIRLFLOW® generates a much higher tangential or 'swirl' velocity component near the wall, which is believed to be one of the key factors in reducing the scale growth. For the SWIRLFLOW® technology to function properly, all existing structure in the conventional mixing tank must be removed. This effectively eliminates all of the potential flow 'dead zone' regions, which have a low velocity, and therefore tend to have increased rates of scale formation. A common example of this low velocity region is behind the baffles in conventional agitated tanks.

Beyond the success in alumina precipitation tanks, SWIRLFLOW® offers a better alternative to the conventional agitator design used in tanks in the neutralisation circuit at the Agnico Eagle Australia, Fosterville Biox gold refinery plant. Fosterville Biox plant is located about 130 km North-Northwest from Melbourne, Australia. A conventional agitator design with an axial flow impeller and baffles was used in the neutralisation tanks at Fosterville. The scale formation in the neutralisation tank occurs when lime is added to adjust the pH to approximately 7, before the waste can be send to tailings storage. The issue with the conventional mixing tank design is that the mixing action in the upper section of the tank is gentle and low velocity. Therefore, when lime was added into the tank, it created a super saturated condition in the upper part of the tank. This caused the neutralisation reaction to take place quickly, which led to scale forming rapidly, creating a fillet in the top section of the tank.

Figure 2 shows an example of this typical scale formation in the upper section of tank, which grew over a period ranging from 6 weeks to 3 months. The scale also grew in other parts of the tank, such as the bottom fillet and behind the baffles, where the velocity is expected to be low, and eventually covered the tank wall. Among many concerns, during cleaning the safety concern is, the scale could fall from the top of the tank which may cause serious injury to the people who work below, e.g. driving the forklift in and out of the tank to remove the scale. Scale formed on the mixing tanks was observed at a number of different refineries however to further understand the scale behaviour and formation, this scale needs to be reproduced in the laboratory as described in the experimental set up.

a) Precipitation tanks

b) Draft tube with scale formed
(Before caustic clean)

Figure 1: Precipitation tanks at an alumina refinery

a) Neutralisation tanks b) conventional mixing tank

Figure 2: Conventional mixing tank in the gold refinery neutralisation circuit

EXPERIMENTAL SET UP

CSIRO's approach was that in order to tackle the scale issue, it was essential to firstly understand the scale formation. Therefore, the important first step was to be able to reproduce the scale in the laboratory. A series of geometrically scaled down mixing tanks of 2 L capacity were set up in the CSIRO laboratory to include the draft-tube tank, conventional mixing tank and SWIRLFLOW® tank. Figure 3 (a) shows the modelling tank with a draft tube, which was included as part of the research program since CSIRO has extensive experience and data on the draft tube and the SWIRLFLOW® precipitation tanks from the alumina plant. It was important for the physical modelling to be able to reproduce the scale phenomena that was observed at the refinery.

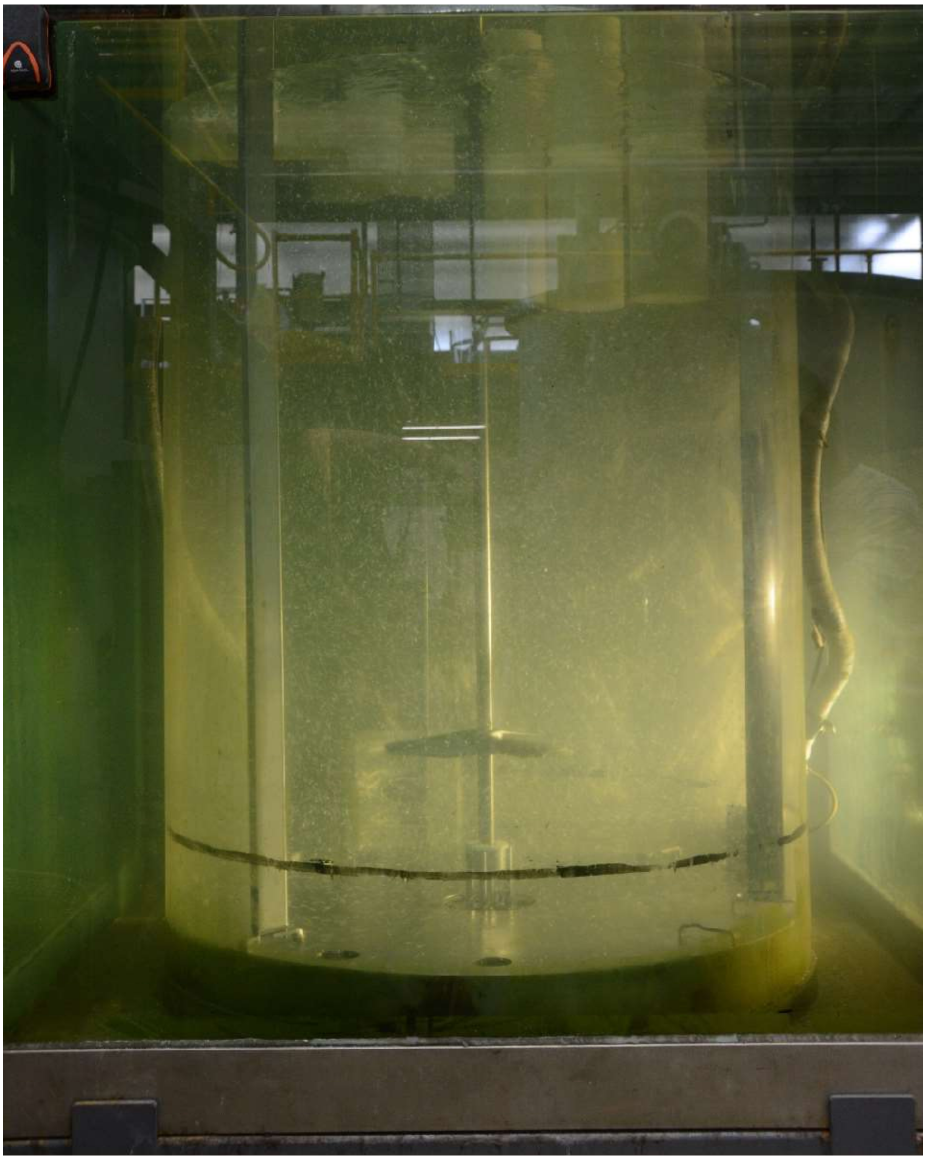
Similarly, a scaled down conventional mixing tank was set up, as shown in Figure 3 (b), to understand, and reproduce, the scale issue in the neutralisation tank at Fosterville gold refinery. A mixing tank was also set up with SWIRLFOW®, as shown in Figure 3 (c), so that the scale formation under different agitation conditions could be determined. All of the agitators were run at nominally the same power, so that a fair comparison could be made between the different designs. At the initial stage of the research, rather than quantifying the scale, the assessment of the scale growth in the laboratory was by visual observation and photos.

a) Draft-tube b) Conventional mixing tank c) SWIRLFLOW ®

Figure 3: Laboratory modelling mixing tanks, 2000 mL; a) draft tube (alumina precipitation), b) Conventional mixing tank (neutralisation at Biox gold refinery) and c) SWIRLFLOW ®

When Fosterville agreed to proceed with the SWIRLFLOW® implementation, the plant trial results of the scale thickness measured from a rod that was deliberately submerged into a fixed location of a mixing tank were also available for comparison. The ultimate objective was to reduce scale in the neutralisation tank; however, it was also important to carry out due diligence to assure that the SWIRLFLOW ® was not compromising the tank's suspension performance. A larger scaled down modelling tank was set up to include a conventional mixing design, which was then replaced by the SWIRLFLOW® configuration, so that their performance could be assessed under the same operating conditions. The modelling mixing tank consists of a 1 m diameter, 1.5 m high transparent acrylic tank installed in an outer square glass tank (for optical correction) as shown in Figure 4. The agitator shaft was equipped with an Ono Sokki SS101 torque transducer and speed detector. The motor was equipped with a Danfoss variable speed drive to allow the agitator speed to be varied.

The slurry used in the suspension tests was made up of water and sand particles of similar particle size distribution (PSD) and solids concentration (~100 g/L) as that used in the Fosterville neutralisation tank. The tests were conducted at ambient conditions. The torque and sedimentation bed height were recorded at a range of fixed speeds during the tests. The test rig, agitator speed and other operating parameters for the conventional mixing tank are detailed in Table 1.



a) Conventional mixing tank.

b) Swirl flow mixing tank.

Figure 4: Mixing tank diameter 1 m; a) convention mixing tank, b) SWIRLFLOW® mixing tank

Table 1: Operating conditions & geometry of scaled down from plant data

Parameter	Full-scale (Conventional)	Full-scale (SWIRLFLOW®)	Laboratory (Conventional)	Laboratory (SWIRLFLOW®)
Tank diameter, T [m]	5.1	5.1	1.00	1.00
Liquid level, H [m]	5.20	5.20	1.00	1.00
Tank volume, Vol [m^3]	106.60	106.60	0.80	0.80
No. of Baffles	4	0	4	0
Impeller diameter, [m]	1.91	1.39	0.38	0.291
Bottom clearance, [m]	1.55	3.18	0.30	0.62
Motor power capacity, [kW]	11.00	11.0	-	-
Speed, N [rpm]	56	56	270	270
Tip velocity, [m/s]	5.59	4.08	5.42	3.85
Solids concentration, [g/L]	106	106	106	106
Liquid density, [kg/m^3]	1000.00	1000.00	1000.00	
Solids density, [kg/m^3]	2400.00	2400.00	2400.00	1000.00
Slurry density, [kg/m^3]	1061.83	1061.83	1061.83	1061.83
Particle maximum size [mm]	0.40	0.40	1.40	1.40

Computational Fluid Dynamics (CFD) was also conducted on the neutralisation tank at full-scale, for both configurations, the conventional and SWIRLFLOW® tanks, to predict the wall velocities, as it is anticipated that a higher wall velocity will lead to lower scale deposition onto the tank wall.

Whilst conducting a trial on the plant, CSIRO also established a scale testing rig to closely monitor and make assessment on the scale formation by comparing the results from laboratory to the plant. Figure 5 shows a test rig consisting of a twin tanks system, which have a volume capacity of 50 L each. They are nominally identical and are configured as a geometrically scaled-down version of the full-scale neutralisation tank. The twin tanks set up allows the test program to better understand the scale behaviour between the test results from the plant trial and laboratory results. One tank was used as a benchmark of the SWIRLFLOW® implementation at Fosterville, whilst the other tank was used to test potential design improvements.

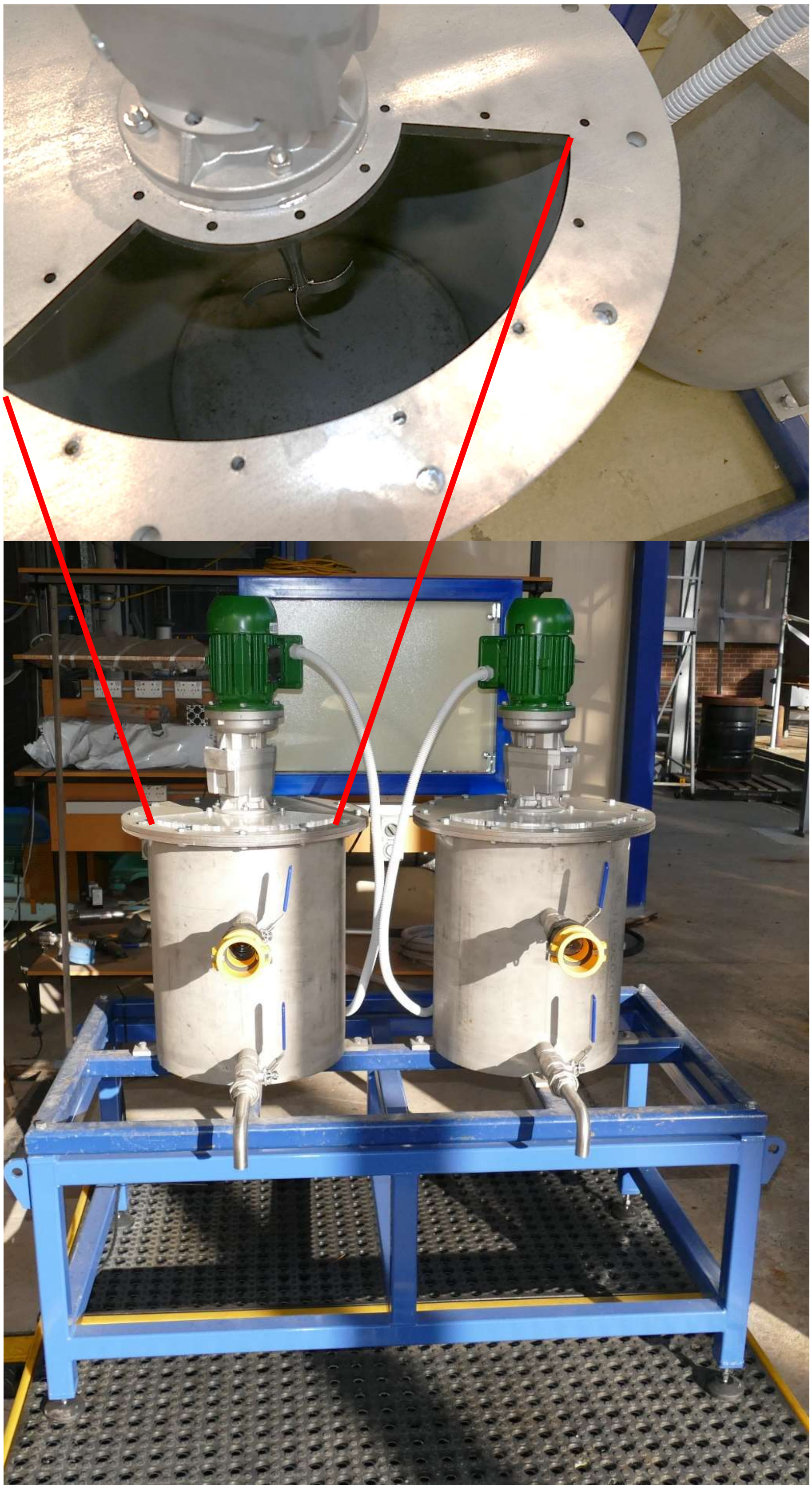


Figure 5: Mixing tank, 50 L, physical modelling to optimise the scale reduction with SWIRLFLOW ®

RESULTS

The scale growth in the lab-scale mixing tank was successfully reproduced in the laboratory to be identical profile as that observed at the plant. Figure 6 shows the results of scale growth in mixing tanks with different agitation configurations; a) conventional mixing tank, b) draft-tube tank and c) SWIRLFLOW® tank. The conditions such as the volume, type of chemical, concentration of chemical and procedures such as when and how the chemical were added to these tanks to grow scale were kept the same. This was carefully controlled to establish the scale growth from the laboratory to be similar to what was observed on the plant. From the scale growth modelling, it was found that the SWIRLFLOW ® tank has a much lower scale growth rate compared to the conventional and the draft tube tanks.

a) Conventional mixing tank

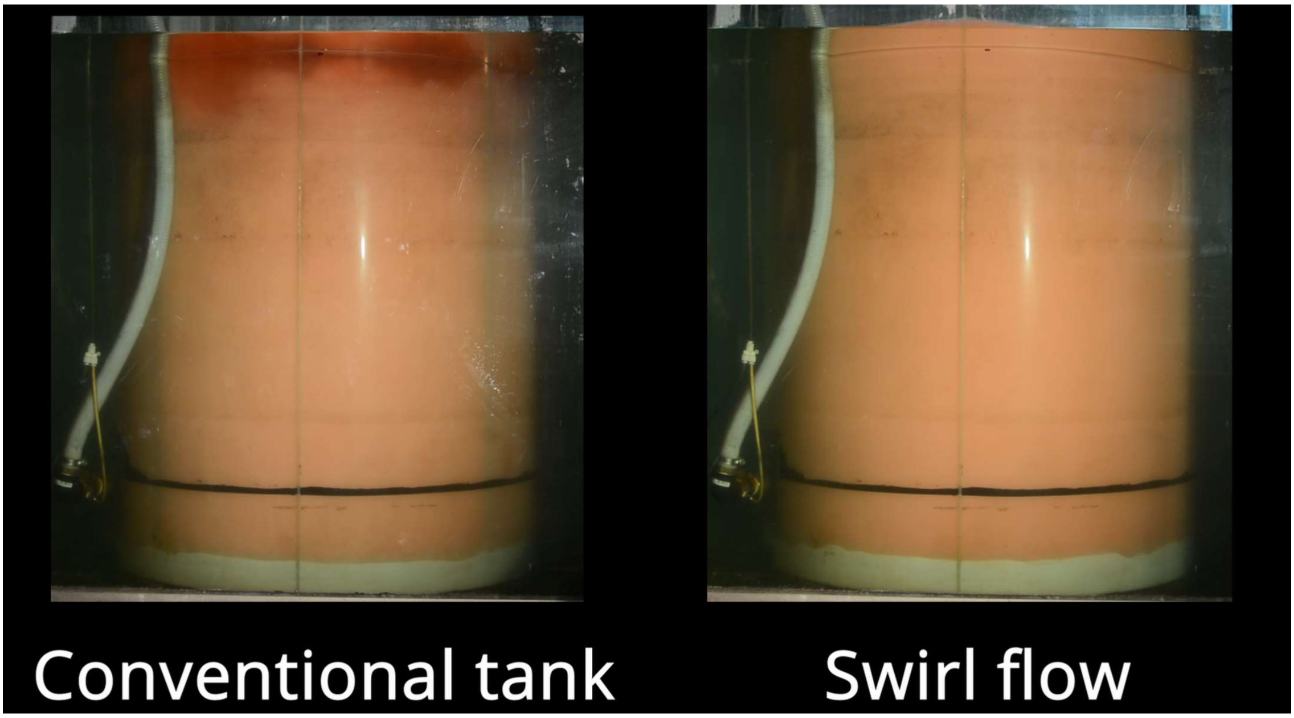
b) Draft tube tank

c) Swirl flow mixing tank

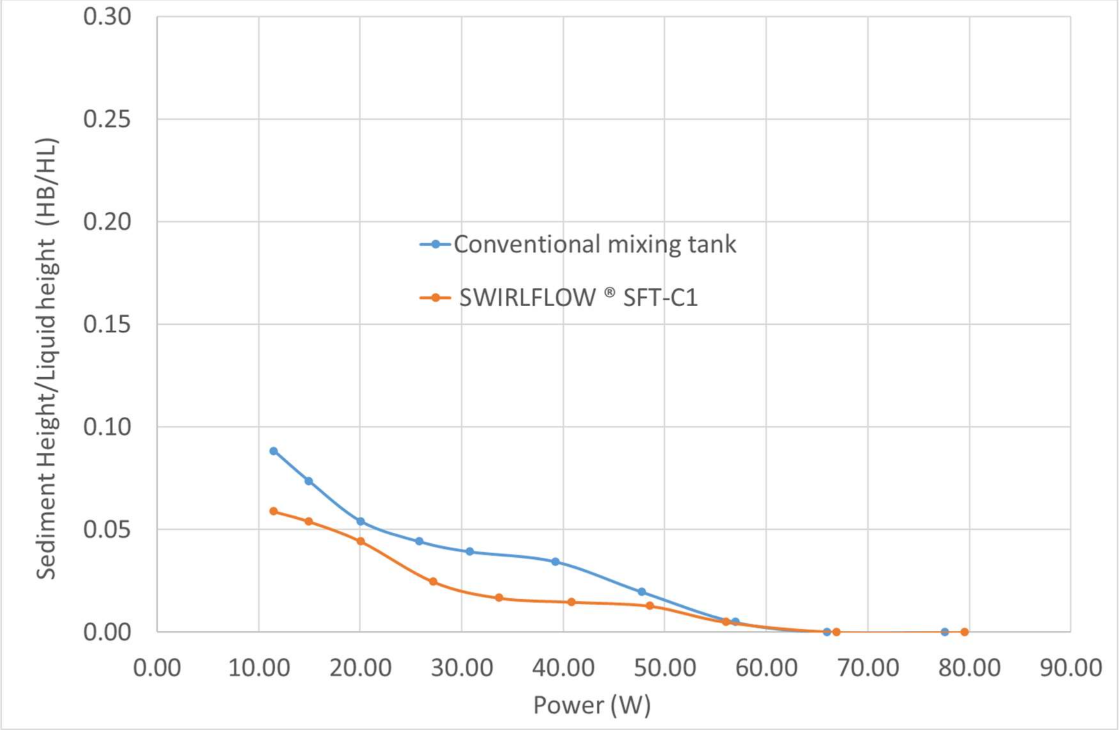
Figure 6: Scale reproduced in the laboratory

While was also found that the SWIRLFLOW® had a lower scale deposition rate compared to the conventional mixing tank, it was also essential to compare the mixing and suspension performance between the two cases to establish if the process performance would be detrimentally affected by the change in agitator design.

Figure 7 (a) shows the contrast of the mixing performance between the two designs. The conventional mixing tank (on LHS) appears to have slightly less homogenous mixture compared to the SWIRLFLOW® mixing tank. This can be seen by the unmixed dye beneath the liquid surface for the conventional design, whereas the SWIRLFLOW® distributes solids all the way up to the liquid surface. Figure 7 (b) shows that the power required by the SWIRLFLOW® to keep the solids in suspension was slightly lower than the conventional mixing tank. The SWIRLFLOW® consumes less power to achieve the same level of solids suspension and mixing, which means that it is more efficient.



a) Mixing performance based on visual observation



b) Suspension performance (sedimentation bed height versus the power consumption)

Figure 7: The mixing and suspension performance between the conventional mixing tank and the SWIRLFLOW ® mixing tank.

The results from laboratory tests all indicated that the SWIRLFLOW® was able to maintain or offer better performance in terms of mixing and suspension, while also significantly reducing the formation of scale. This effectively means the tank can be kept in operation for a much longer period of time before cleaning is required.

CFD simulations were conducted on the full-scale neutralisation tank, on both the conventional and the SWIRLFLOW® mixing configurations. Figure 8 shows a comparison between the velocities of the SWIRLFLOW® and the conventional mixing tank at the same power consumption. As the SWIRLFLOW® has all internal structures removed to eliminate the dead zones, it is unsurprising that CFD demonstrated it to have much higher tangential velocities near the tank wall. This is believed to be a key parameter in reducing the scale growth rate.

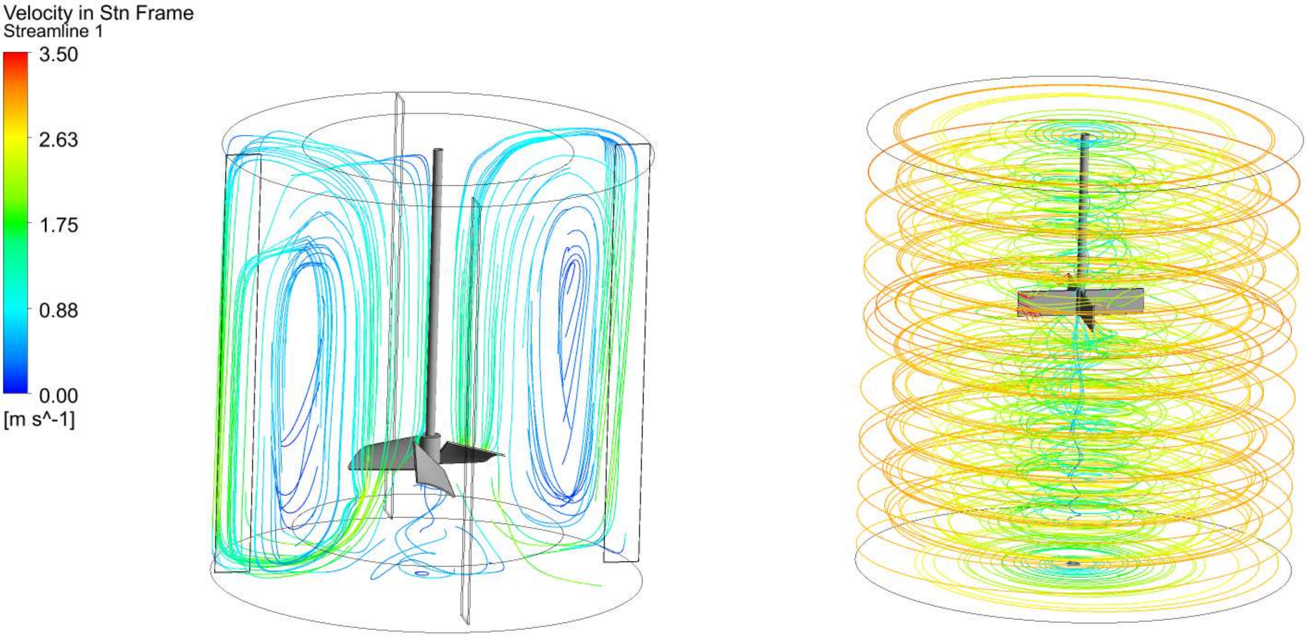


Figure 8: CFD results. Conventional mixing tank (Left) vs SWIRLFLOW® mixing tank (Right)

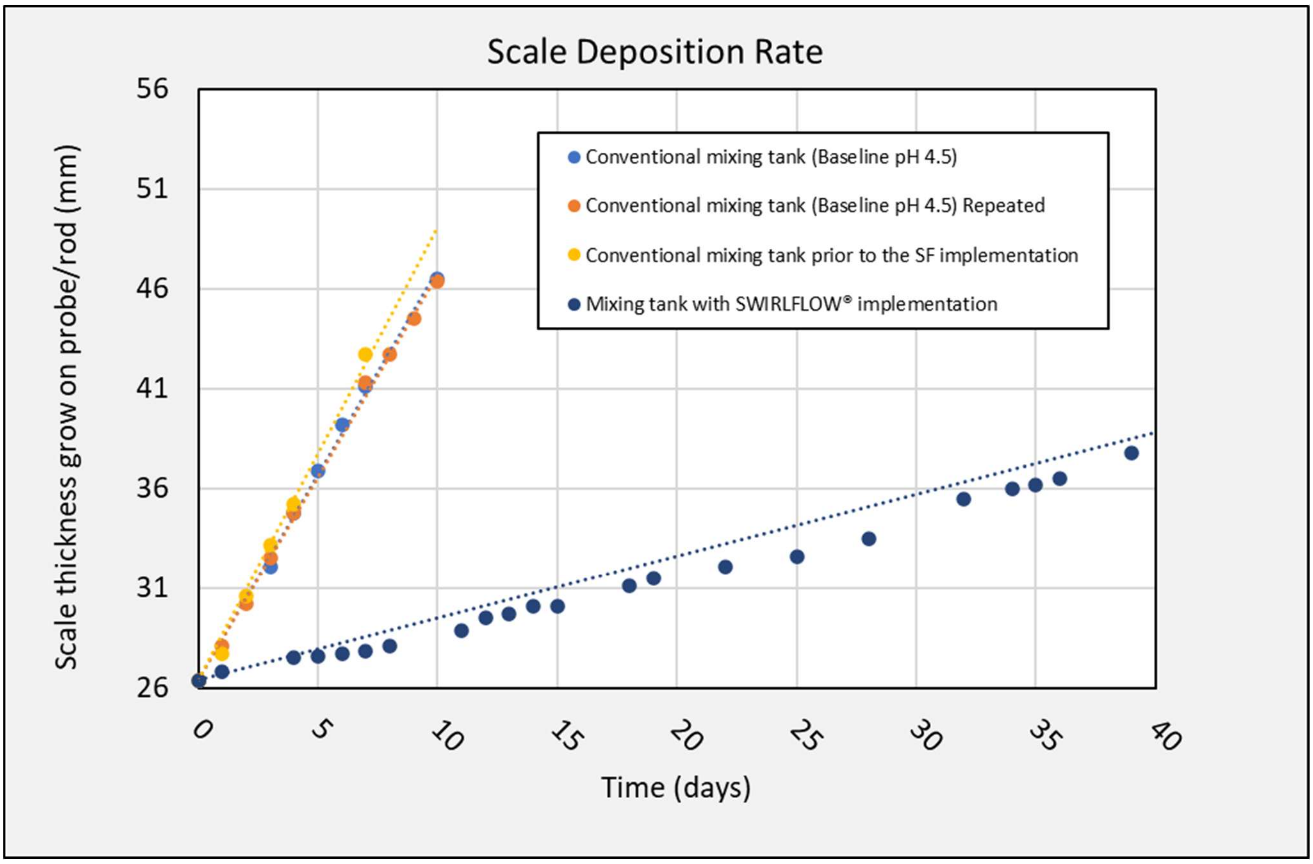
TRIAL RESULTS FROM PLANT

After the retrofitting of the SWIRLFLOW® agitator in the Fosterville Neutralisation tank, measurements of the scale thickness were made. There were compared with similar measurements made with the conventional mixing system in the same tank, prior to the SWIRLFLOW® implementation. Figure 9 shows the scale in conventional neutralisation tank in comparing to SWIRLFLOW® tank with a similar length of online time. For both agitation configurations, a probe/rod as shown in Figure 10 (a) was set up so that it was carefully submerged at the same location and periodically taken out the tank for the scale thickness measurement over time. Figure 10 shows the comparison of scale growth in the neutralisation tank on plant for both the conventional and SWIRLFLOW ® mixing tank. The results show the SWIRLFLOW ® takes approximately ~7-9 times longer than the conventional neutralisation tank to grow to a similar scale thickness compared to the conventional agitator.

a) Typical scale in conventional tank b) Small scale in SWIRLFLOW® tank

Figure 9 Contrast of the scale in a) convention mixing tank and b) SWIRLFLOW®

a) Probe/rod that scale grows on



b) Scale thickness measured from the probe coupon

Figure 10: Scale thickness comparison between the conventional mixing tank versus SWIRLFLOW®

DISCUSSIONS

Through many years of experience from working with alumina refineries, CSIRO has gained good understanding about the scale deposition in draft-tube precipitation and conventional mixing tanks in the full-scale plant and appreciates that scale is an ongoing issue for the Bayer process. The scale is not just limited to the alumina industry, it's a generic problem across other industries. In seeking further understanding, the CSIRO team decided to grow scale in the lab-scale mixing tank, effectively to reproduce the scale behaviour that is observed in the plant. Obviously, the time taken to grow the scale in the laboratory was on the order of days, which is much shorter than the time in which significant scale growth occurs on the plant, which normally could take weeks, months or even longer. Due to this reason, the scale characteristics from the laboratory may not be the same to those on the plant, however, an important feature of the scale grown in the laboratory was that it was similar in profile to what is observed on plant.

In particular, a visual assessment was made, which concluded that the scale profile in the laboratory was similar to the scale observed in the draft tube tank and the SWIRLFLOW® tank in the precipitation circuit at the alumina refinery. It was also established that the scale growth in the laboratory for both the conventional mixing tank and the tank with the SWIRLFLOW® was similar to what was observed in the neutralisation circuit at the Fosterville gold refinery. The scale growth was successfully reproduced in the laboratory and matched with scale patterns that observed on the plant means the technique, chemicals and procedures used are qualitatively able to reproduce the plant scale behaviour. This technique could therefore potentially be used to model and to solve scale issues in other mixing tanks, regardless of the industry, or application that the mixing tank is being used for.

The scale growth rate in the SWIRLFLOW® was significantly lower than in the conventional mixing tank. The plant trial demonstrated that the scale formed in the SWIRLFLOW® tank over a similar period to the conventional mixing tank was not only much thinner, but it was also more uniform. This reduces lumpy and overhanging scale formations at the top of the tank, which could fall and become a safety hazard during the cleaning progress.

The results from experimental tests, CFD and full-scale trial aligned with each other. This means that the potential of the current CSIRO modelling techniques are not just limited to utilising the SWIRLFLOW® as a practical solution to scale issue in mixing tanks, but it also opens a new opportunity for future technology development to solve scale issues in other processes and equipment.

CONCLUSIONS

The SWIRLFLOW® technology developed by CSIRO was proven in both laboratory testing and plant trial results to successfully reduce the scale formation on the tank wall for tanks in the precipitation circuit in an alumina refinery, as well as in a gold neutralisation tank. This resulted in keeping the tank online for much longer, which helps in reducing maintenance costs.
The laboratory test results, CFD and full-scale trial results support the hypothesis that the removal of baffles and other flow dead zones, along with increased wall velocity led to significant scale reduction.

The scale growth patterns observed on the plant were successfully and accurately reproduced in the CSIRO facility for the draft tube tank, conventional mixing tank and tank with SWIRLFLOW® agitation.
Therefore, regardless of which industry a mixing tank is from, as long as the problem is related to scale formation, the SWIRLFLOW ® technology offers a better alternative to conventional mixing systems.

The plant implementations demonstrated a significant reduction of the scale in the neutralisation tank at Fosterville with SWIRLFLOW®. This means the scale modelling technique described here can be utilised to model the scale growth in the existing mixing tanks however more work is needed to achieve the objective of the scale reduction.

ACKNOWLEDGEMENTS

The authors would like to thank QAL and Agnico Eagle Australia – Fosterville Mine for their ongoing support, sharing trial results and collaborations in help to model and solve this complex issue. We also acknowledge the contribution of CEM for supplying full scale SWIRLFLOW® impellers.

REFERENCES

1. B. Nguyen, S. Mills, A. Nash, M. Davood, D. Harris, K. Mohanarangam, W. Bruckard, L. Graham and J. Wu, Swirl Flow Agitation for a BIOX Neutralization Process, World Gold conference
2. Nyombolo, B M, Neale, J W, and van Staden, P J, 2000. Neutralization of bioleach liquors, *The Journal of The South African Institute of Mining and Metallurgy*, 100(7):403-408.
3. Wu, J, Wang, S, Nguyen, B, Marjavaara, D, Eriksson, O, 2016. Improved Mixing in a Magnetite Iron Ore Tank via Swirl Flow: Lab-Scale and Full-Scale Studies, Chemical Engineering & Technology, 39(3):505-514.
4. Wu, J, Nguyen, B, Graham, L, Connor, T, Marjavaara, D, Eriksson, O, Jia, F, and Coleman, C, 2018. Slurry Tank Swirl Flow Agitation, Alumina 2018, in Proceedings The 11th AQW International Conference, pp175-179 (AQW (Inc.)).
5. Wu, J, Nguyen, B, Graham, L, 2010. Energy Efficient High Solids Loading Agitation for Mineral Industry, Canadian Journal of Chemical Engineering, 88(2):287-294
6. Wu, J, G. Lane, G, Livk, I, Nguyen, B, Graham, L, Stegink, D, Davis, T, 2012. Swirl flow agitation for scale suppression International Journal of Mineral Processing, 112–113 (Special Issue Communition 2009):19–29

DRIVING STAGEWISE DEVELOPMENT OF GOLD PROJECTS WITH THE JAMESON CONCENTRATOR AND ALBION PROCESS™

By

Paul Bullock, Rebecca McKechnie, Kelvin Roche, Glenn Stieper and Stanko Nikolic

Glencore Technology

Presenter and Corresponding Author

Laurie McDonell
laurie.McDonnell@glencore.com.au

ABSTRACT

Extending the life of a depleted gold mine can be as simple as treating and recovering gold from neighbouring sulphide deposits and tailings. However, gold is often finely disseminated in iron sulphides within the mineral matrix, making dissolution via direct cyanidation ineffective. Processing this refractory ore yields poor gold recovery, therefore pre-treatment is required to concentrate and oxidise these iron sulphide minerals prior to cyanidation. This includes flotation to concentrate the sulphide mineral fraction, followed by sulphide oxidation and then gold dissolution and recovery. Glencore Technology works closely with resource development organisations to evaluate a stagewise implementation of technologies to treat their sulphide ore reserves. This involvement begins early with collection of laboratory data to support the design and installation of a Jameson Concentrator™, followed by ultrafine grinding with the IsaMill™, and finally construction of an Albion Process™ plant utilising our OxiLeach™ Reactors (OLR). The Albion Process™ combines ultrafine grinding and oxidative leaching at atmospheric pressure, a proven process for sulphide mineral oxidation. Stagewise plant implementation allows for staggering equipment design and delivery schedules to expedite the path to establishing revenue. This is achieved by aiming for production of a saleable grade of flotation concentrate in the first stage of plant operation, with on-site dore production occurring after commissioning of subsequent plant stages. On-site sulphide oxidation allows the processing of a lower grade, higher recovery concentrate without the transport costs and penalties associated with concentrate trading terms. The design and delivery of these downstream stages can occur in parallel to construction of the Jameson Concentrator™ flotation circuit. This approach inherently carries higher risks and rewards. This paper reviews the application of GT technologies in the context of stagewise implementation, including compliance with environmental, social and governance (ESG) criteria and impacts on project execution.

Keywords: Gold, Jameson, IsaMill, Albion, Testwork, Staged Implementation, Production

INTRODUCTION

As reserves of easily recoverable oxide gold ores are depleted, operations are increasingly looking to extend the life of mine by treating and recovering gold hosted in neighbouring sulphide deposits and tailings. In these ore bodies, gold is often finely disseminated within the sulphide matrix, making it unamenable to direct cyanidation and resulting in poor gold recovery. For such projects, the question exists as to how to best treat sulphide ores; a decision which is influenced by multiple factors including offtake opportunities, legislation and existing infrastructure. Whilst many sites have elected to generate a saleable con, higher transport costs and penalties associated with concentrate trading terms are often incurred, and in some countries there are restrictions on the sale of intermediate products, leaving operations with no choice but to generate a final product.

Generating a final product from sulphide ores requires pre-treatment to render the finely disseminated gold amenable to cyanide leaching. This includes a flotation stage to generate a higher-grade concentrate, followed by sulphide oxidation. Stagewise plant implementation allows for a progressive approach to the design, installation, and commissioning of these stages of production, resulting in staggered delivery schedules, earlier product generation and an expedited path to establishing revenue. The first stage of plant operation aims to produce a saleable grade of flotation concentrate to provide a source of revenue until on-site doré (final product) production is achieved after subsequent plant stages are commissioned.

Glencore Technology (GT) works closely with resource development organisations to create bespoke flowsheet solutions to achieve maximum economic and environmental benefits. GT is the proprietor of several specialised technologies suited to stagewise implementation and has extensive expertise evaluating their amenability to treat unique refractory ores. This paper reviews the application of GT technologies in the context of stagewise implementation, including compliance with environmental, social and governance (ESG) criteria and impacts on project execution.

STAGEWISE IMPLEMENTATION – FROM ORE TO DORÉ

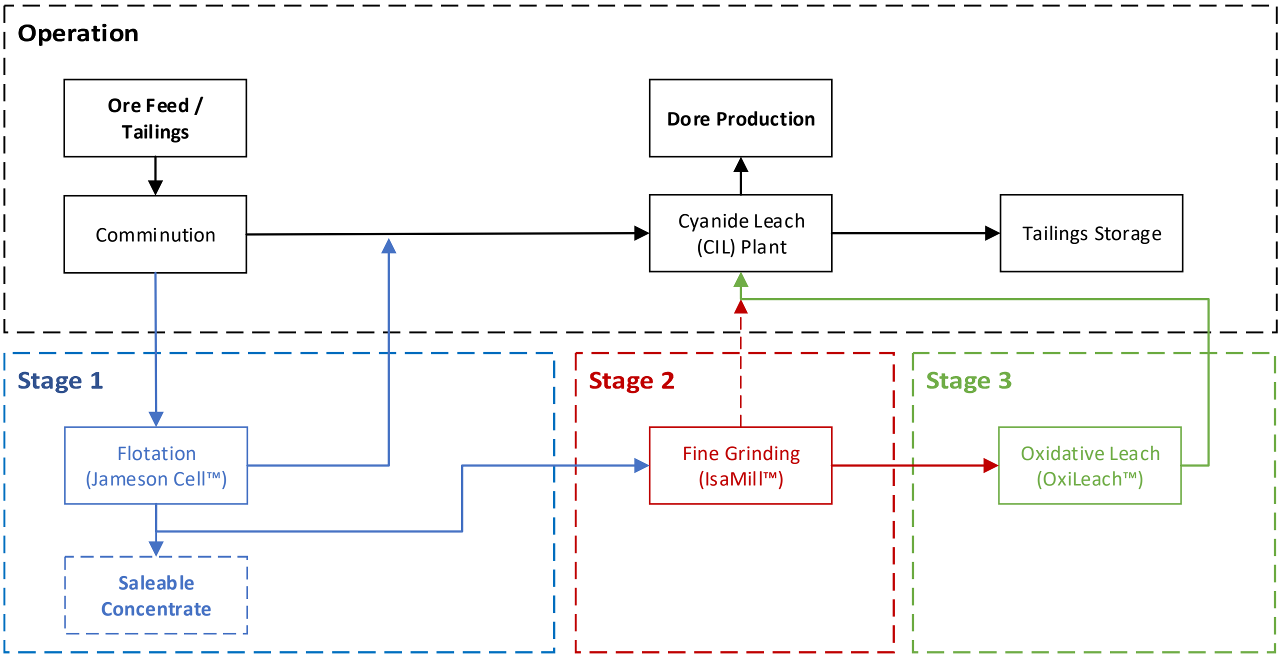


Figure 1: Stagewise Implementation Using GT Technologies

GT's approach to treating refractory gold ores includes three stages (**Error! Reference source not found.**). The first stage is the building and commissioning of a flotation circuit to produce a saleable concentrate. Stage 2 adds a fine grinding mill to achieve better mineral liberation and improved recoveries. Stage 3 involves the building and commissioning of an oxidative treatment plant to enhance downstream cyanide leaching recoveries. A stagewise approach to implementation allows for expedited revenue generation upfront using well understood mineral processing technology while the downstream plant is fully tested, designed and commissioned.

This section describes GT technologies and processes and their role in stagewise implementation, initially leveraging a saleable concentrate to create capital to build a full leach plant for the production of dore on site.

Stage 1: Flotation and the Jameson Concentrator

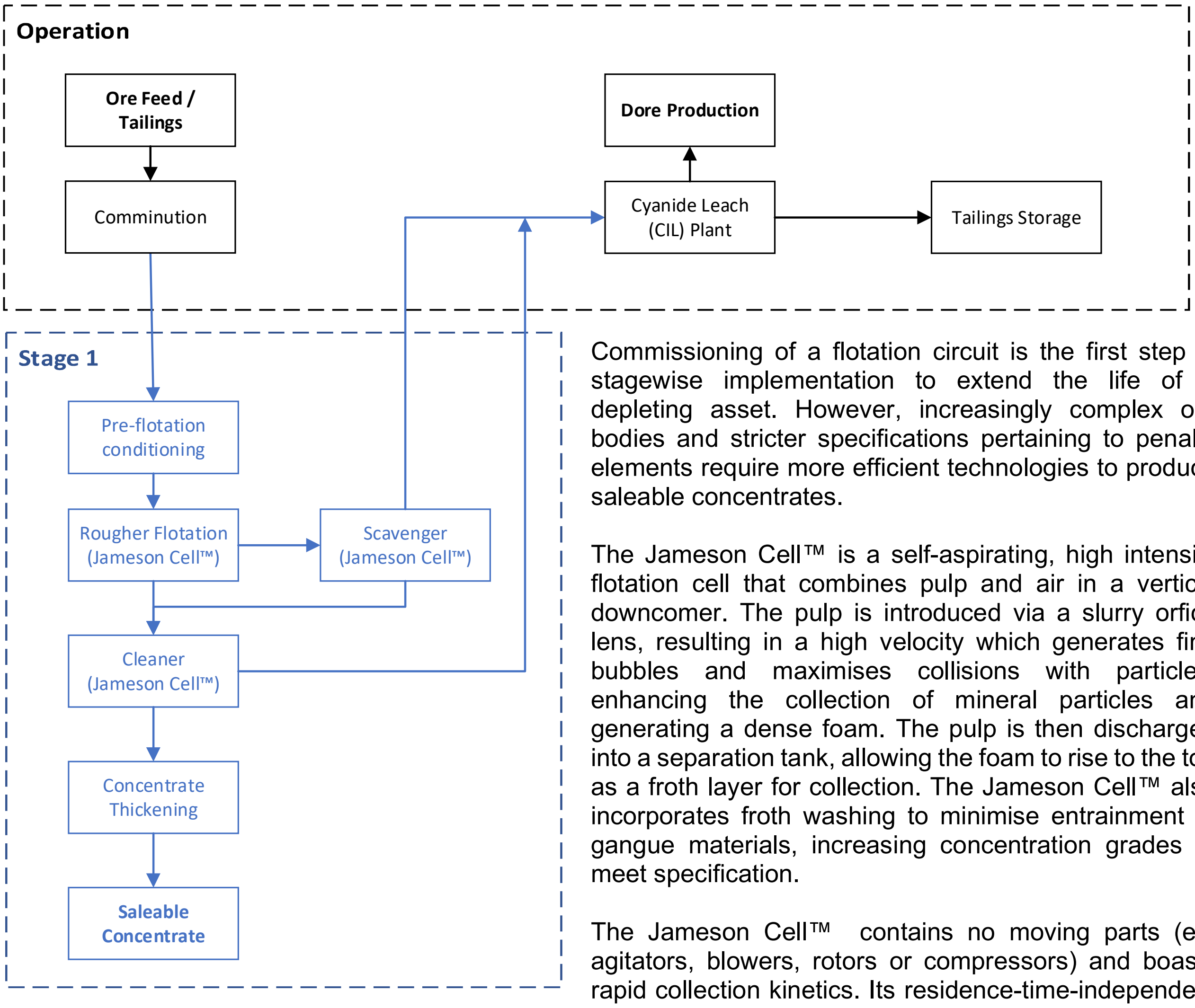


Figure 2: Stage 1 Jameson Concentrator

Commissioning of a flotation circuit is the first step in stagewise implementation to extend the life of a depleting asset. However, increasingly complex ore bodies and stricter specifications pertaining to penalty elements require more efficient technologies to produce saleable concentrates.

The Jameson Cell™ is a self-aspirating, high intensity flotation cell that combines pulp and air in a vertical downcomer. The pulp is introduced via a slurry office lens, resulting in a high velocity which generates fine bubbles and maximises collisions with particles, enhancing the collection of mineral particles and generating a dense foam. The pulp is then discharged into a separation tank, allowing the foam to rise to the top as a froth layer for collection. The Jameson Cell™ also incorporates froth washing to minimise entrainment of gangue materials, increasing concentration grades to meet specification.

The Jameson Cell™ contains no moving parts (e.g agitators, blowers, rotors or compressors) and boasts rapid collection kinetics. Its residence-time-independent design allows for a significantly smaller circuit footprint (up to 50% reduction compared to conventional flotation cells) (Example 1).

Example 1: ESG benefits of Jameson Cell Concentrator

GT was commissioned to design of a 7 million tonnes-per-annum lead-zinc concentrator for a mineralogically complex ore project. Conventional flotation would have required 63 tank cells. Replacement with Jameson Cells lowered the cell requirement to 19 and reduced the flotation circuit footprint by 50%. Additionally, reduced earthworks, concrete, and structural steel combined with only using 30-60% of the energy of a conventional flotation circuit resulted in significant CAPEX and OPEX savings.

Jameson Cell™ technology was originally installed at the head of conventional cleaning circuits to improve cleaning capacity, minimise deportment of deleterious elements, and generate a high grade final concentrate. There are over 500 Jameson Cell™ installations across coal, precious and base metals, potash, and oil sands, demonstrating its reliability for new circuits and upgrading existing ones. The advantages Jameson Cells™ have over conventional flotation units has resulted in the introduction of a full Jameson Concentrator™ layout in several operations. The Jameson Concentrator™ is easily constructed by an EPCM to go live quickly, allowing quick turn arounds and commencing cash flow as early as possible. Implementing a Jameson Concentrator™ circuit rapidly produces a high-grade saleable concentrate and raises capital to implement Stages 2 and 3.

The flexibility of the Jameson Cell™ underpins efficient stagewise implementation, due to its ability to operate at numerous points along the grade recovery curve. When an operation progresses from generating a saleable concentrate to producing dore on site, the Jameson flotation circuit can be reconfigured to maximise recovery. The operation of the Jameson cells makes it possible to change the function of the plant to meet this future stage of processing where a bulk concentrate of lower grade is required, either by operating at a different point on the grade recovery curve or configuration of the circuit. Cleaners are able to be switched to treat the scavenger concentrate only or could be utilised at the head of the circuit to achieve a higher overall plant throughput.

Figure 2 shows a typical concentrator consisting of a rougher, scavenger and a cleaner to produce a final saleable concentrate grade. This configuration is to ensure the ability to produce a high grade / saleable concetrate. If mineralogy and liberation issues exist further regrinding may be required to assist in achieving the desired results. In certain applications and during treatment of transition material it may be possible to produce a final concentrate grade at an acceptable recovery from a single Jameson Cell™ or a simple rougher scavenger Jameson Cell™ circuit.

Stage 2: Introduction of Fine Grinding

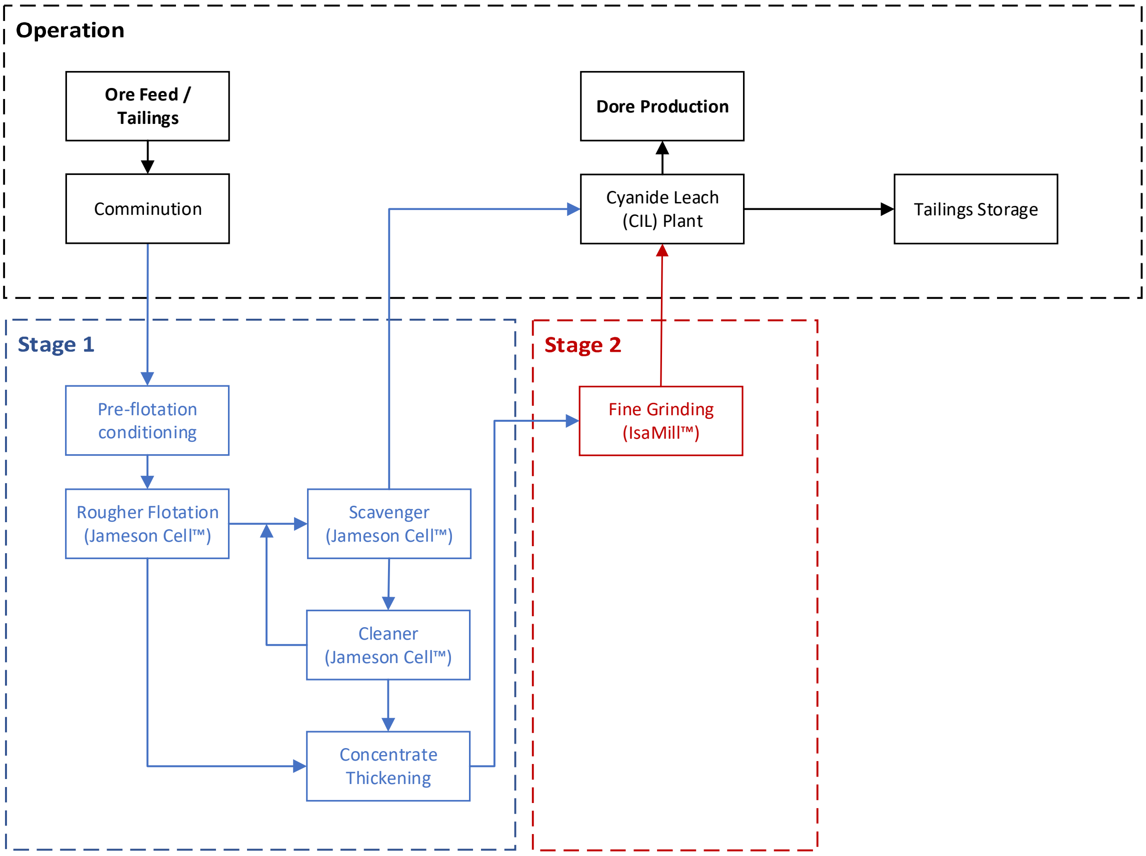


Figure 3: Stage 2 Ultrafine Grinding and Jameson Concentrator Modifications

As the operation progresses from selling concentrate to producing all of the gold as dore on site then the introduciton of a fine grinding stage may provide an ecomonic interim solution (Figure 3). For some projects, fine grinding alone may result in sufficient liberation of the gold for recovery in traditional downstream processes (Example 2). Many conventional mills are inefficient in fine grinding duties due to prolonged milling times, resulting in low throughputs and high power consumption. Ultra Fine Grinding (UFG) mills like the IsaMill™ overcome the limitations of conventional mills by using rotating stirrers within a stationary mill shell, offering an energy efficient means of achieving ultrafine grind sizes.These sizes achieve better liberation of gold encapsulated within the sulphide matrix.

The IsaMill™ is a well-established, highly efficient fine grinding mill capable of achieving the tight size distribution required to enhance the rate of downstream chemical reactions. This key parameter is tracked in the testwork by measuring the coarse size index (CSI) which is a ratio of the p98 to p80. This is achieved by a plug-flow design that incorporates multiple grinding chambers.

Example 2: Ultrafine Grinding (UFG) using the IsaMill as a replacement for roasting – environmental and economic benefits

A refractory gold processing operation was investigating opportunities to eliminate their existing roaster. Pressure oxidation, biological oxidation and ultrafine grinding were investigated as potential alternative processes. Economic analysis indicated that UFG had the highest Net Present Value return of the alternative process options. The inclusion of an IsaMill™ for UFG of the refractory gold concentrate to 10µm prior to cyanidation resulted in an increase in recovery from 75% to 92%. The replacement of roasting with UFG via IsaMill™ also eliminated harmful air emissions whilst maintaining production rates.

As the central shaft rotates, the grinding discs and grinding media are agitated, causing the media to centrifuge out along the disc face towards the shell liner. The media is then redirected back towards the mill shaft. Slurry fed to the mill must pass through each grinding chamber before exiting and is unable to short circuit the mill. The slurry discharges through a product seperator with a closer spacing between the final and rotor discs to centrifuge coarser particles toward the shell, at which point the rotor directs the material back into the grinding zones (Figure 4).

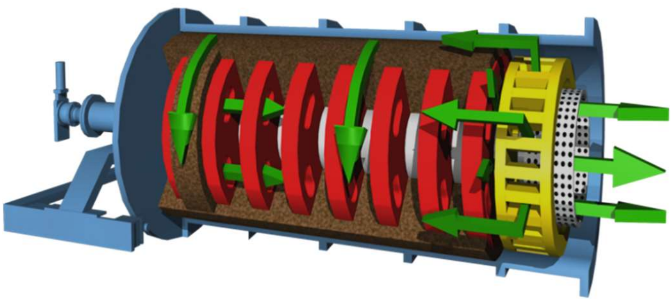


Figure 4: Slurry and media flow through the IsaMill™

The design of the IsaMill allows for efficient particle flow and grinding, retention of grinding media, and minimises the potential for particles to short circuit. This achieves the tight size distribution necessary for efficient oxidation and high gold recoveries in downstream cyanide leaching. Additionally, the horizontal design of the IsaMill™ allows for a single floor design and offers greater power intensity compared to alternative fine grinding mill options. GT has accredited laboratories perform designated IsaMill™ testwork on every client's sample to generate an IsaMill™ signature plot for specific grind energy. Tests are performed in a continuous mode to replicate full-scale performance, resulting in a 1:1 direct scale up from testwork to full-scale.

Stage 3: Albion Process and downstream processes

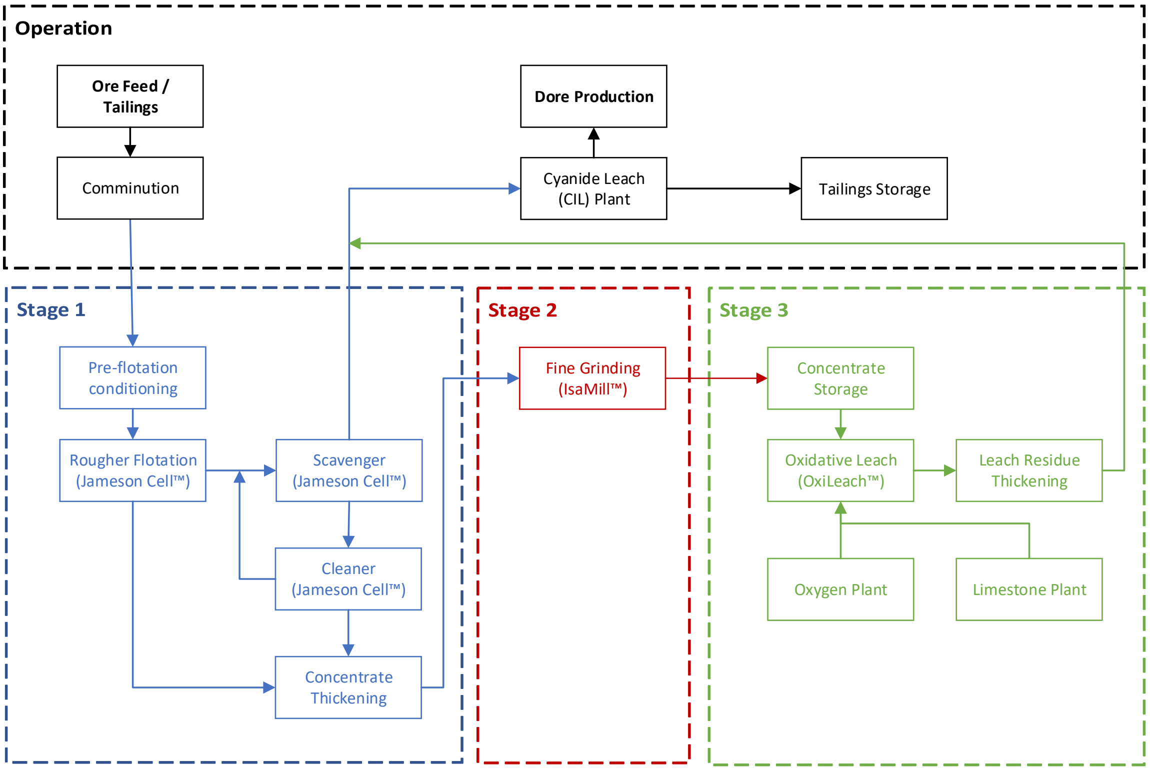


Figure 5: Stage 3 Albion Process full implementation

If the gold is extremely refractory and is not amenable to extraction after UFG alone, oxidative treatment may be required. The Albion Process is a globally patented technology developed by Glencore Technology in 1994. It consists of two stages: (i) ultrafine grinding in the IsaMill™ and (ii) oxidative leaching. Therfore the oxidative leaching step may be added after UFG in a project requiring stagewise implementation.

During the first stage, feed material undergoes UFG in an IsaMill™ to increase surface area and minimize potential passivation of the mineral surface during leaching. The high degree of residual strain within the crystal lattice coupled with increased surface area results in a high defect density within individual mineral grains and increased reactivity of minerals to oxidation. Therefore, less extreme conditions are required to effectively oxidise sulphide minerals, enabling leaching to occur at atmospheric pressure. This renders autoclaves unnecessary, leading to a safer, simpler, and more energy efficient plant.

During the second stage, oxidative leaching occurs in the OxiLeach™ Reactor System. This system utilises Glencore Technology's patented HyperSparge™ to inject oxygen at supersonic velocities. The high shear and increased particle contact achieved by the HyperSparge™ offers more efficient oxygen mass transfer in solution and maximises oxygen utilisation, thereby ensuring the process is not limited by oxygen mass diffusion. The hypersparge achieves oxygen utilisation of 80% or higher, compared to 50% as exhibited by other sparging systems (Voigt, Mallah and Hourn, 2017).

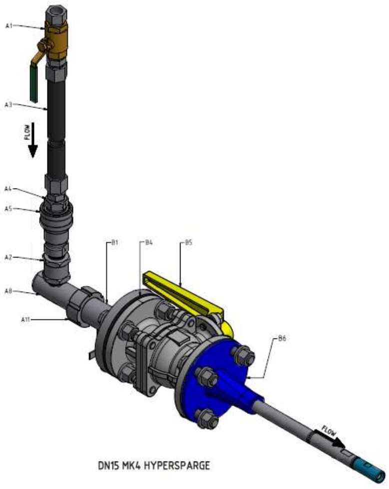


Figure 6: HyperSparger™ supersonic oxygen injection

Figure 7: OxiLeach™ Reactor

The leaching process occurs autothermally at atmospheric pressure (with the sulphide in feed targetted above 10%w/w) and, in the case of most gold bearing concentrates, under mildly acidic condidtions (approximately pH 5.5). Paradoxically, despite the term 'leaching', precious metals remain within the solid residue to be subjected to cyanidation or an equivalent gold extraction process while the liquour reports to treatment/tailings.

The Albion Process™ has been commercialised for both precious and base metals, with seven plants constructed and reported on extensively (Hourn & Turner, 2010; Hourn & Turner, 2012; Voigt, Hourn, & Mallah, 2016; Senshenko, Aksenoz, Vasiliev, & Seredkin, 2016). Compared to other oxidative leach processes, it is low cost, technically straightforward, effective, and safe. It eliminates the need for a large technical workforce, and due to its simple layout and atmospheric conditions, it avoids the myriad safety and process issues associated with other approaches. The Albion Process™ has also been shown to have up to one third lower operating costs compared to other oxidative leaching processes in part due to lower consumption of limestone and oxygen (24% less), water (58% less) and power (50% less) compared to traditional oxidation methods (McNeice, Marzoughi, Kim, & Ghahreman, 2021; Aylmore, 2012).

Example 3: Effectiveness and ESG benefits of the Albion Process ™ for treating refractory gold ores

The Ararat processing plant owned by GeoPro Mining (GPM) in Armenia is Glencore Technology's flagship gold processing operation. This plant was commissioned in January 2014 and produces 120,000 oz of gold per annum. The Albion process plant consists of a M3000 IsaMill™ and nine OxiLeach Leach Reactors. The introduction of the Albion plant to the Ararat flowsheet increased gold recoveries to 92%.

The plant is highly robust, able to operate with high variability in feed sulphide content and still achieve high gold recoveries above 95%. This is of particular importance due to the requirement of the plant to treat feed from seven different ore bodies, including three that are high in arsenic. During the process, arsenic is leached into the bulk solution. The addition of limestone (to control the pH to 5.5 within the range for optimal pyrite oxidation) allows for the co-precipitation of iron and arsenic, forming a stable ferroarsenate (predominantly scorodite). Subsequent testing of the Ararat tailings storage facility have confirmed the stability of the arsenic compounds.

STAGEWISE IMPLEMENTATION IN THE REAL WORLD

The following section provides an example of a proposed stagewise implementation of GT technologies for an existing gold operation. The site has less than two years processing remaining for the current oxide ore body. The underlying sulphide deposit consists of pyrite hosted gold, with significant non-sulphide gangue present as silicates and carbonates. Due to the short time requirements for being able to process a sulphide ore, stagewise implementation will see the commissioning of a Jameson rougher-scavenger flotation circuit for the initial generation of a saleable concentrate.

Stage 1: Flotation (Jameson Concentrator)

Feed to the Jameson flotation circuit is characterised as follows:

Throughput	mtpa	1.0
	tph	125
P_{80}	µm	150
Au grade	g/t	1.9
S grade	%	0.8

The subsequent circuit to treat this feed will consist of one Jameson Rougher cell and one scavenger cell, equating to USD 12M capital expenditure and USD 0.9M operating expenditure annually.

The saleable concentrate produced by the Jameson flotation circuit will achieve gold recoveries of 85% gold at a mass pull of 6%, resulting in a gold concentrate grade of ~35 g/t. Using the current gold price of USD 65 per gram, selling this concentrate for 75% of payable contained value will provide USD 79 million per annum in revenue, in addition to the processing of remaining oxides, bringing additional revenue forward whilst remaining design, installation and commissioning is completed for Stages 2 and 3.

Stage 2: Ultrafine grinding (IsaMill™)

Single-stage ultrafine grinding will be achieved through the implementation of a M7,500 IsaMill™ with an installed motor power of 2,200 kW. Feed will be ground to 12 µm, with the available power offering flexibility to manage ore variability and future requirements. This stage equates to an annual operating expenditure of USD 2.5M and a capital expenditure of USD 13.5M. Whilst fine grinding alone provides a 20% uplift in gold recovery (from 20% to 39% after ultrafine grinding), unlike Example 2 earlier the economic benefits of a separate fine grinding stage prior to commissioning an oxidative leaching circuit are not recognised for this particular project, thus Stage 2 and Stage 3 will be implemented concurrently.

Stage 3: Oxidative Leaching (Albion Process™)

Upon commissioning of Stages 2 and 3, the flotation circuit settings will be configured to maximise recovery as opposed to grade. Flotation will achieve 95% gold recovery at a mass pull of 10%, yielding a feed to the Albion circuit as follows:

Throughput	tpa	100,000
	tph	12.5
P_{80}	µm	12
Au grade	g/t	18
S grade	%	9

The concentrate sulphur grade of 9% has been modelled and will allow for autothermal operation of the leaching circuit, negating the need for external heating. The leach residence time will be approximately 48 hours, during which 2.8 tonnes per hour of limestone and 1.5 tonnes per hour of oxygen will be dosed into the reactors. The capital cost of the Albion Process™ leaching circuit will be USD 22.2M, with an annual operating cost of USD 3.8M. After oxidative leaching approximately 96% of gold will be recoverable via cyanide leaching, equating to an uplift in gold recovery of 76% (absolute) attributed to the Albion Process™ and resulting in an annual revenue of USD 112M.

The total combined cost to facilitate the treatment of sulphide ore is USD 36.5, with an annual operating cost of USD 6.8M. With a stagewise approach to implementation costs are staggered over time, with capital cost consisting of stage 1 - 25%, stage 2 - 28% and stage 3 - 47% and the operating cost stage 1 - 12%, stage 2 - 36%and stage 3 - 56%. This highlights one of the key advantages of a staged approach to implementation, starting with lower capital and operating cost technologies to expedite revenue whilst concurrently staggering expenditure over time to improve cash flow and complete the subsequent stages to achieve high gold recoveries and a robust process for future production (Figure 6).

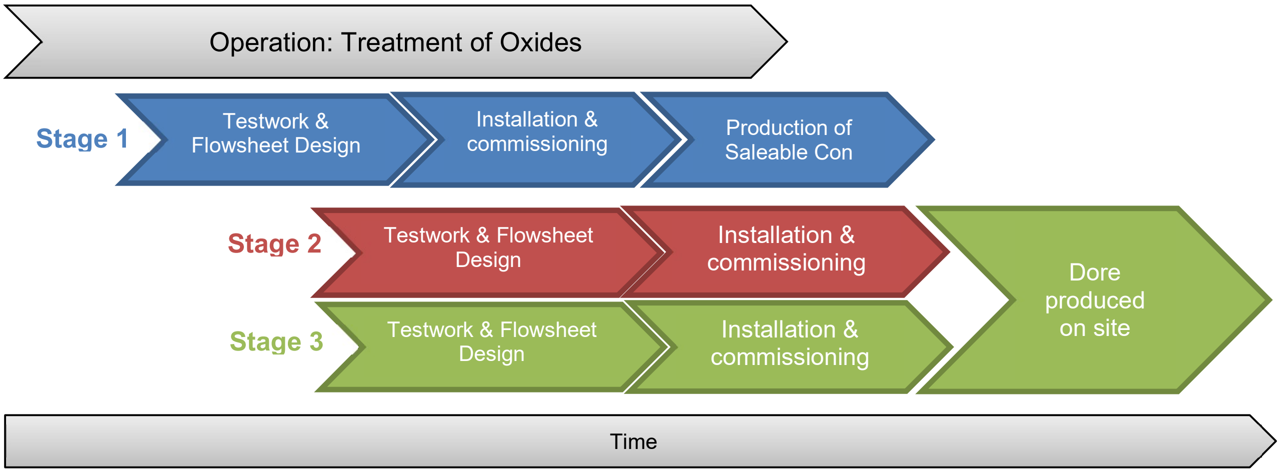


Figure 6: Staggered Implementation of Sulphide Treatment Stages

CONCLUSIONS

A stagewise approach to implementation allows eligible projects to extend the life of their operation by providing the opportunity to treat refractory sulphide materials. This approach ensures that revenue generation can initially commence through the production of a saleable concentrate while the downstream processes that will ultimately treat this concentrate onsite are refined in design and commissioned. The evaluation and comparison of differrent implementation stages should focus on comparing cost:revenue ratios or NPV rather than a sole focus on a designated recovery targets. This must then be supported by good financial modelling to ensure the most beneficial economic case is identified

GT provides several industry-proven technologies and processes that complement each other and facilitate a stagewise approach to implementation. All of these technologies are flexible and simple in their in operation, to compliment this approach to implementation. Applying this this to a real-world scenario for treating 1.0 mtpa of ore through a Jameson Concentrator to produce 100kt of concentrate through the Albion Process, the total cost of the project is USD 36.5, with an annual operating cost of USD 6.8M. A stagewise approach to implementation allows for costs to be staggered over time, with the capital cost consisting of stage 1 - 25%, stage 2 - 28%and stage 3 - 47% and the operating cost stage 1 - 12%, stage 2 - 36%and stage 3 - 56%. Thus for 25% of the capital spend and 12 % of the operating cost it is possible to start the implementation of the flotation stage and generate a saleable concentrate. This generates ~70% of the available gold revenue for the operation and sets up the project to complete the final stages to achieve high gold recoveries and production onsite with a robust plant for future production.

NOVEL CORROSION-RESISTANT AND EROSION-RESISTANT COATING QUALIFIED FOR USE IN PRESSURE OXIDATION SEVERE SERVICE

By

Evelyn Ng

Callidus Process Solutions, Australia

Presenter

Evelyn Ng
evelyn.ng@callidusgroup.com.au

ABSTRACT

Metals and non-metals, which are normally non-flammable in air, can burn very rapidly in an oxygen-enriched environment such as Pressure Oxidation severe service. Accordingly, oxygen systems present unique risk and safety concerns wherein materials selection of equipment used in oxygen systems is critical in mitigating the risk of damage to equipment and injury to personnel due to fire or explosion.

Equipment used in POx is also subject to erosive and corrosive conditions. While titanium is commonly used in hydrometallurgical severe service equipment because of its mechanical strength and corrosion resistance, its use in POx is limited by its high propensity for ignitability and flammability.

Conventional materials that are resistant to ignition or slow to combust include copper and its alloys, nickel-copper alloys, carbon steel and the 300 series of stainless steels. However, these materials often do not have sufficient mechanical properties or corrosion-resistance to be used in POx service where severe service conditions present in the form of high temperature, high pressure, and acidic environment, and also comply with industry guidelines and international standards governing the use of oxygen.

In response to a POx client's problem of high levels of corrosion and erosion on auxiliary discharge equipment to the autoclave, Callidus developed an innovative dual coating BM-1600 that is a corrosion and erosion-resistant system for severe service equipment in POx and HPAL applications.

BM-1600 is fully dense, metallurgically bonded to the surface with a low coefficient of friction ideal for application on isolation equipment such as metal-seated ball valves (MSBVs). BM-1600 was further qualified for use in oxygen-enriched environments following ASTM G124 for temperatures up to 230ºC and pressure up to 3,300kPag in a 100% gaseous oxygen environment. ASTM G124 is an internationally recognized test method that determines the minimum gas pressure and temperature that supports self-sustained burning.

The introduction of this novel coating applied on valve trim extended the life and reliability of the MSBV from less than one month to over a year. Common failure modes observed on MSBVs such as spallation of traditional bond coatings and topcoats of ceramic coating, and erosion and corrosion are eliminated with the application of BM-1600 to the valve trim.

This presentation will discuss case studies featuring Callidus' proprietary BM-1600 coating with results from field trials.

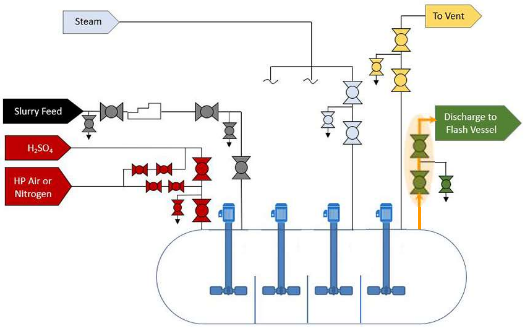
Keywords: Pressure Oxidation, POx, severe service, fused coating, metallurgical bond, sulphuric acid, surface modification, equipment, autoclave, ASTM G124, corrosion, erosion, performance, duplex stainless steel, Alloy 20, high temperature, repair, refurbishment, properties, performance

Innovation at Callidus

An ongoing journey

ALTA 2022 Ni-Co-Cu

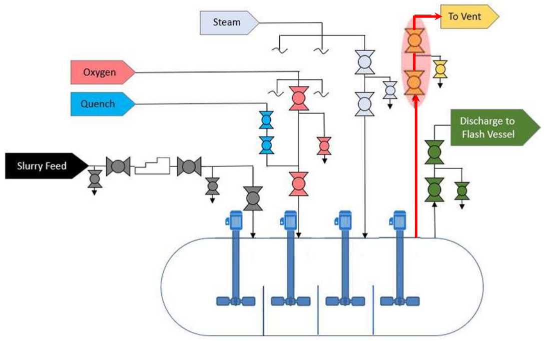
“FM-1500”



HPAL severe service

This year Gold-PM

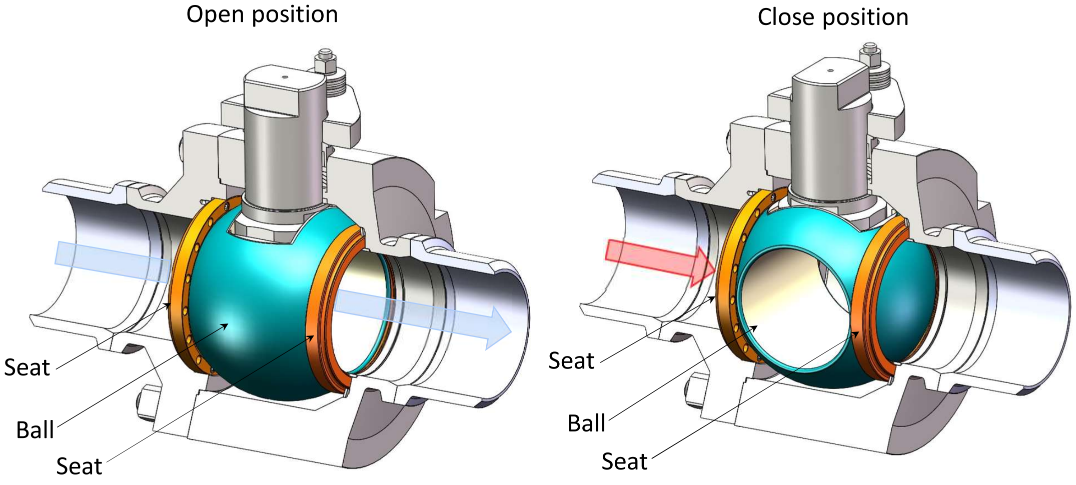
“BM-1600”



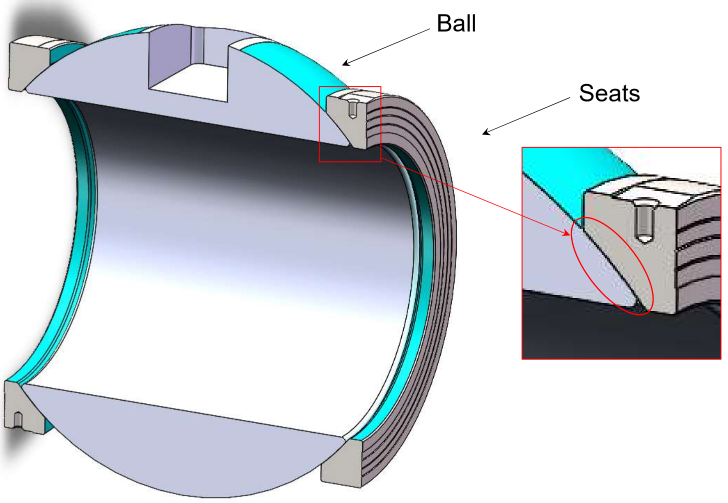
POx severe service

Image ref. https://www.valv.com/product/hpal_pox_autoclave/

Ball valve primer – components & isolation



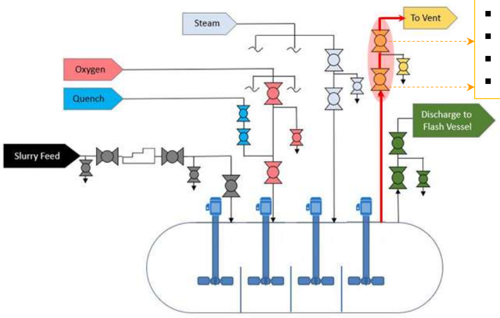
Valve trim and typical coated regions



Vent isolation location

- Ball removed after 4 weeks in-line.
- Super duplex stainless steel substrate + oxide APS coating
- Loss of sealing efficiency:
    - Loss of protective coating from erosion & corrosion
    - Mass corrosion of the substrate (SDSS)

Vent isolation requirements & challenges



Operating conditions:
- Temp. >200ºC
- Press. >3,000kPag
- Oxygen
- Steam
- Sulphuric acid

Requirements:
- Adequate mechanical properties
- MSBV requirements
    - Dimensional accuracy
    - Spherical contact between ball & seats
- Corrosion-resistant
- Erosion-resistant
- Industry compliant
- Safe for use in oxygen service

POx vent isolation challenges:
- Allowable materials qualified for use >200ºC?
- Allowable material geometry >200ºC?

Image ref. https://www.valv.com/product/hpal_pox_autoclave/

What happens with only one coating

A tale of two cities

Oxide coating via APS thermal spray

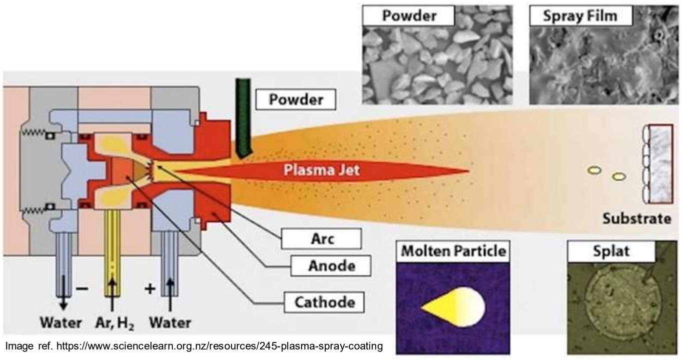
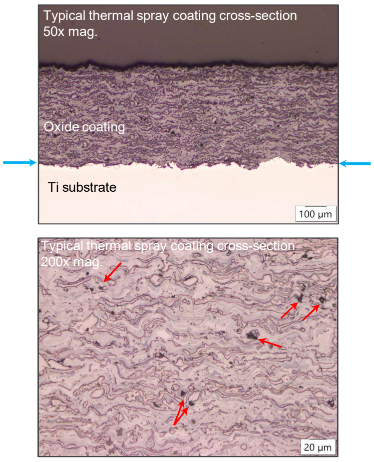


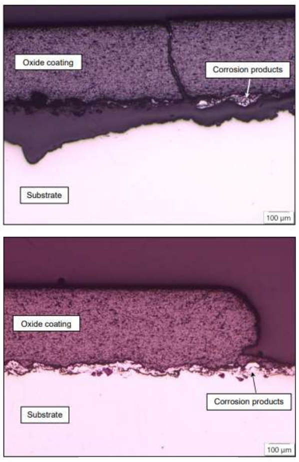
Image ref. https://www.sciencelearn.org.nz/resources/245-plasma-spray-coating

As-applied

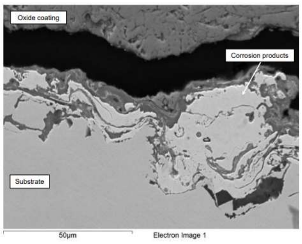
When only ceramic coating is used



Before service



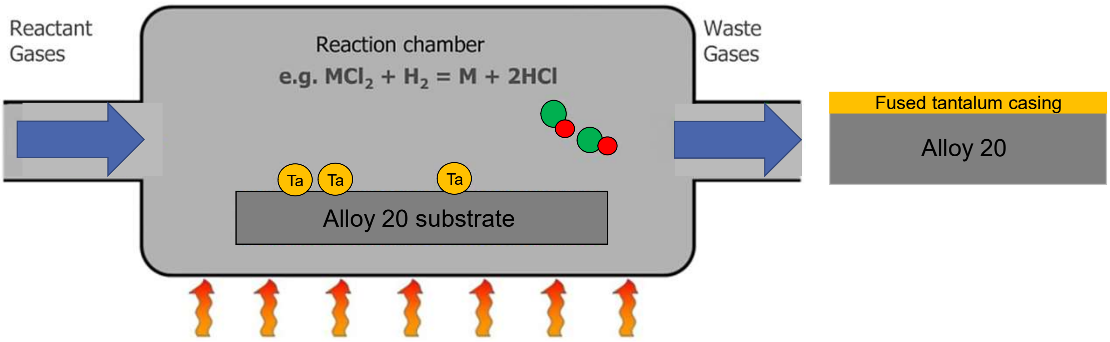
After service



Coating properties:

- Hardness – Yes
- Porosity – Yes
- Adhesion – No

Tantalum bond coat via Chemical Vapour Deposition

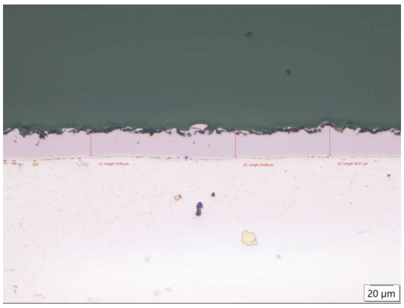


e.g. $MCl_2 + H_2 = M + 2HCl$

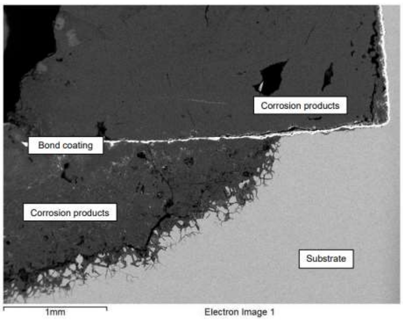
Heated vacuum chamber

Image ref. https://www.youtube.com/watch?v=9XKGVHPXXho

When only tantalum coating is used



Before service



After service

Coating properties:

- Hardness – No
- Porosity – No
- Adhesion – Yes

An innovative two-coating system BM-1600

Alloy 20

- ✓ Mechanical properties
- ✓ Industry compliant

Fused tantalum casing

Alloy 20

- ✓ Mechanical properties
- ✓ Industry compliant
- ✓ Corrosion-resistance from fully dense, fused & chemically inert tantalum casing

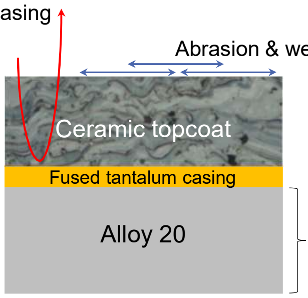
Ceramic topcoat

Fused tantalum casing

Alloy 20

- ✓ Mechanical properties
- ✓ Industry compliant
- ✓ Corrosion-resistance from fully dense, fused & chemically inert tantalum casing
- ✓ Erosion-resistant abrasion & wear resistant ceramic topcoat

BM-1600 in theory



Hot acidic gas & vapour
cannot penetrate fused casing

Abrasion & wear resistant topcoat

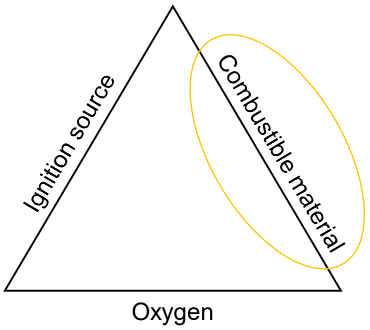
Maintains structural integrity because it is protected from hot acidic gases and fluids

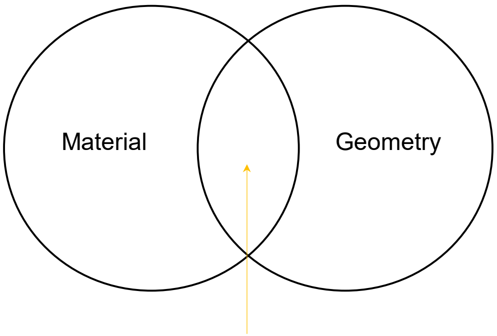
- ☐ Safe for use in oxygen service?

Qualifying for use in POx

And for use in a rocket ship to space

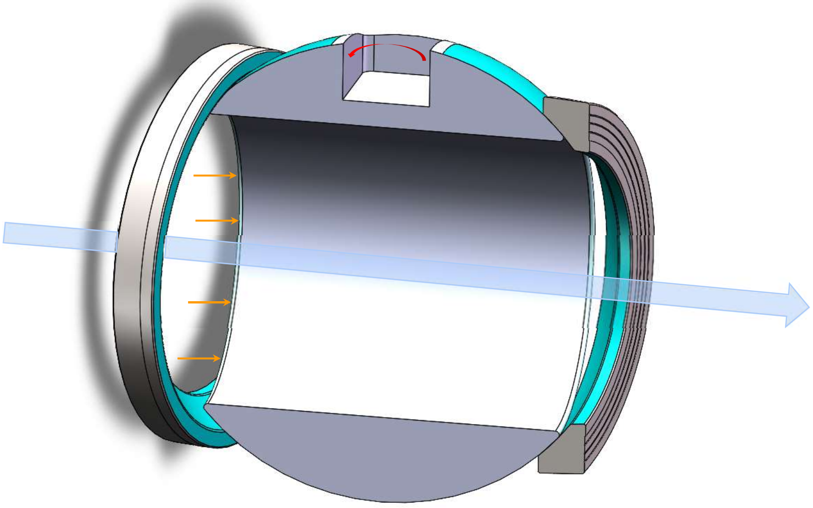
Added challenge: POx service is unique





Selecting material type
for valve trim use in POx

Thinnest geometry on valve trim is the leading edge on the ball



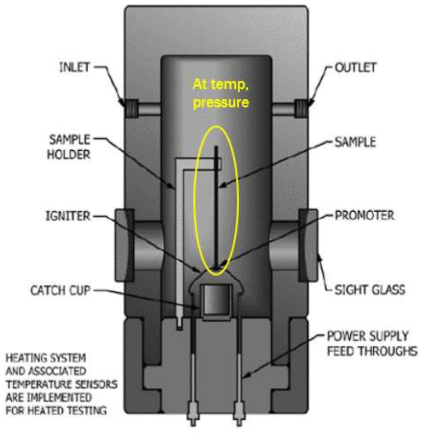
Compatibility tests to evaluate ignition & flammability

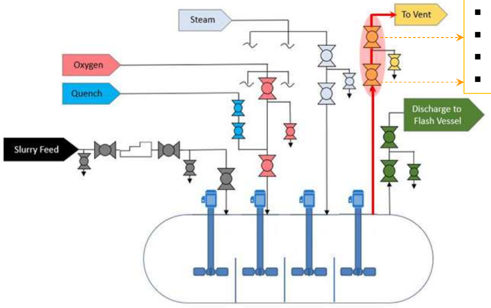
Test name	Test designation
Autogenous Ignition Temperature	ASTM G72 ISO 11114:3 ISO 21010
Heat of Combustion	ASTM D4809
Gaseous Fluid Impact Sensitivity	ASTM G86 ASTM D2512 ISO 21010 NASA Test 13A
Liquid Oxygen Mechanical Impact Testing Sensitivity	ASTM G86 ASTM D2512 ISO 21010 NASA Test 13B
Oxygen Index	ASTM G125
Aging	ASTM G114
Promoted (Heated) Combustion of Metals	ASTM G124

Specifically for testing non-metals & lubricants

Metals compatibility test in oxygen

ASTM G124 to Qualify BM-1600 for POx

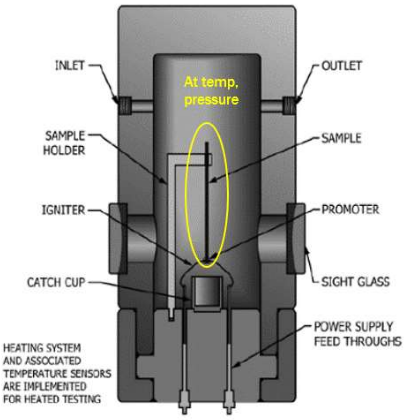


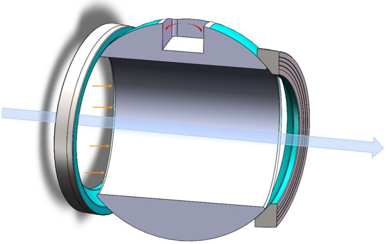


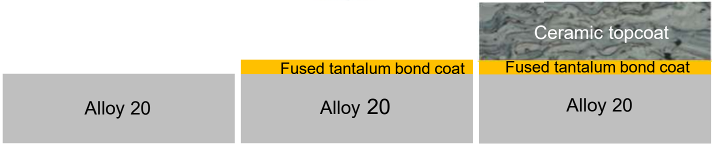
Operating conditions:
- Temp. >200ºC
- Press. >3,000kPag
- Oxygen
- Steam
- Sulphuric acid

ASTM G124 to qualify BM-1600 for POx

- No burn results 5 out of 5 in all configurations







Case studies

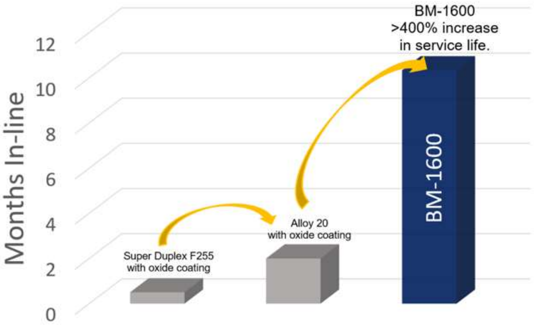
The proof is in the pudding

Case study I

28 days in-line
Super Duplex w/oxide coating
Extensive corrosion on trim.
Loss of sealing capability.

58 days in-line
A20 w/oxide coating
Corrosion initiated at substrate resulting in coating delamination and loss of sealing capability.

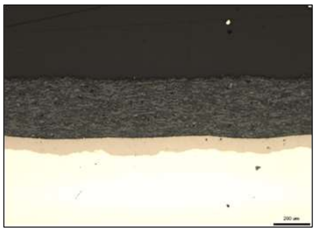
310 days* in-line
BM-1600
Coating intact, minimal to no corrosion observed. Full sealing capability.
* Valve removed for observational purposes.



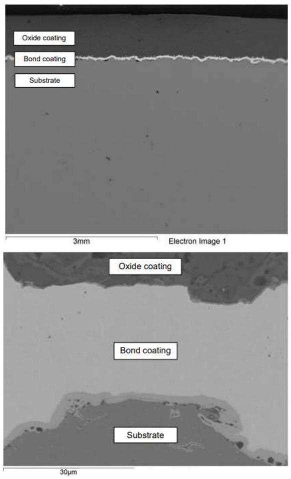
Case study II

		Traditional oxide coating on Super Duplex substrate *4 mo. post-service*	**Callidus BM-1600** *12 mo. post-service*
Downstream	Ball		
	Seat		
Upstream	Ball		
	Seat		

BM-1600 performance in POx severe service



Before service



After service

Coating Property	Tantalum bond coat	Oxide Topcoat
Hardness	Up to 200HV	Up to 900HV
Porosity	100% dense	Up to 3%
Adhesion	Fused Metallurgical bond	Up to 7,000 psi Mechanical bond

---

Summary

- A patent-pending innovative coating system innovated to solve a client’s pain point in the vent isolation location in POx.
- Qualified for use in POx service, beyond standard service parameters.
- Callidus supports the end user to achieve optimum performance in the form of technical, operational, and engineering support to key OEMs to assist with product improvements and development.

PROCESS DEVELOPMENT FOR HYDROMETALLURGICAL RECOVERY OF BASE AND PRECIOUS METALS FROM WASTE PRINTED CIRCUIT BOARDS

By

Huan Li, Elsayed Oraby, Jacques Eksteen,

WA School of Mines: Minerals, Energy and Chemical Engineering,
Curtin University, Australia

Presenter and Corresponding Author

Huan Li
huan.li@curtin.edu.au

ABSTRACT

Base and precious metals contained in waste printed circuit boards (PCBs) have significant economic value, and some of them pose high risks to the environment and human health. Australia generates a relatively small volume of e-waste compared with populous countries and regions (e.g., Europe, China and India) but is among the highest producer of the waste per capita. The lower e-waste volume and the vast geography of Australia bring the opportunity for a decentralised and flexible recycling process. This study aims to develop integrated hydrometallurgical processes for the recovery of base and precious metals from waste PCBs sourced from Western Australia.

The crushed and ground waste PCBs sample contained up to 22.6% copper, 106.8 ppm gold, 170.5 ppm silver and 10.4 ppm palladium, as well as other base metals with varied contents. Different leaching steps and modules were investigated and developed to extract metals selectively. Specifically, ferric sulfuric acid leaching and alkaline glycine leaching were studied respectively for selective base metals extraction, such as copper, zinc, lead, nickel, etc. Ferric sulfuric acid in the presence of thiourea, alkaline cyanide-starved glycine and alkaline glycine-oxidant (non-cyanide) leaching steps were investigated respectively for a second step leaching of precious metals, including gold, silver and palladium. In the case of the alkaline glycine-based leaching process, some base metals that remained in the residue could be further removed in a third step of ferric sulfuric acid leaching.

Based on the results of the multistep leaching, flowsheets with acidic and alkaline routes and a combination of both were proposed. The metal contents in the final residue were reduced to <1% for base metals except for tin and <35 ppm for precious metals. It is hoped that this study can provide insights into the development of fit-for-purpose e-waste recycling processes in Australia.

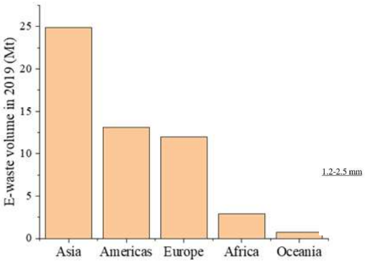
Keywords: e-waste, waste printed circuit board, recycling, glycine, sulfuric acid

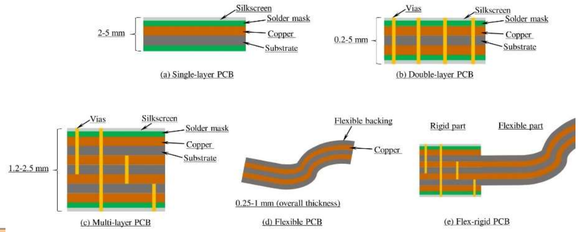
Outline

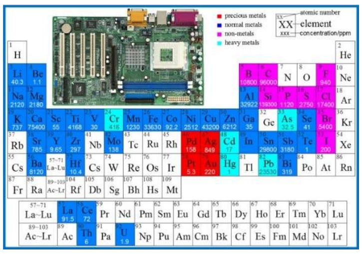
1. “Urban Mining” of e-waste
2. Development of glycine-based process
3. Development of sulfuric acid leaching process
4. Recommendations for future study
5. Acknowledgements

“Urban Mining” of e-waste

- Australia generated 554 kt of e-waste in 2019, ranking 5th worldwide per capita, and only 10.4% of them were documented to be collected and recycled.
- Waste PCBs are complicated e-waste in composition and contents: 40% metals, 30% plastics, 30% ceramics; ~40 types of metals and ~10 types of non-metals; metals locked in plastics & alloys.
- Proximity to the urban area – requiring a safe process.



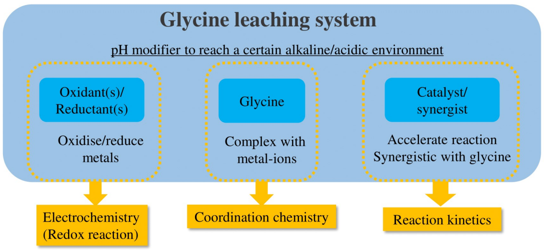




Source: Li, H., Eksteen, J. and Oraby, E., 2018. Resour. Conserv. Recy., 139: 122-139.
Lu, Y. and Xu, Z., 2016. Resour. Conserv. Recy., 113: 28-39

“Urban Mining” of e-waste

- Western Australia’s scenario
  - Low e-waste volume – a pyrometallurgical process may not be highly economical
  - Vast geography – expensive road transport to a centralized location
- Why hydrometallurgy?
  - De-centralized; flexible in scales and modules; capital-friendly to SMEs
- Glycine leaching systems
  - Involving technologies invented at Curtin University and owned by Draslovka a.s.



Glycine leaching system

pH modifier to reach a certain alkaline/acidic environment

Oxidant(s)/ Reductant(s) – Oxidise/reduce metals → Electrochemistry (Redox reaction)

Glycine – Complex with metal-ions → Coordination chemistry

Catalyst/ synergist – Accelerate reaction Synergistic with glycine → Reaction kinetics

Source: Li, H., Eksteen, J. and Oraby, E., 2018. Resour. Conserv. Recy., 139: 122-139.

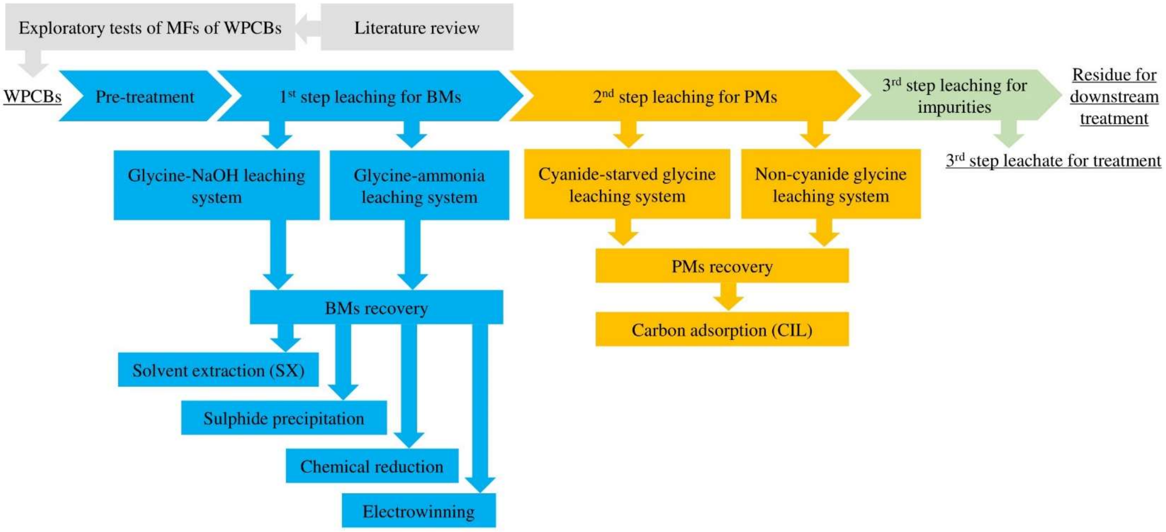
Waste PCBs
⬇
Mechanical-physical processing
⬇
Pyrometallurgy
- Incineration
- Pyrolysis

Hydrometallurgy
- Mineral acid
- Organic acid
- Ammonia
- Cyanidation
- Thio-system

Bio-metallurgy (bioleaching)

Development of glycine-based process

- Schematic diagram of process development



Exploratory tests of MFs of WPCBs ← Literature review

WPCBs → Pre-treatment → 1st step leaching for BMs → 2nd step leaching for PMs → 3rd step leaching for impurities → Residue for downstream treatment

1st step leaching for BMs: Glycine-NaOH leaching system; Glycine-ammonia leaching system → BMs recovery → Solvent extraction (SX); Sulphide precipitation; Chemical reduction; Electrowinning

2nd step leaching for PMs: Cyanide-starved glycine leaching system; Non-cyanide glycine leaching system → PMs recovery → Carbon adsorption (CIL)

3rd step leaching for impurities → 3rd step leachate for treatment

Development of glycine-based process

- Publications from glycine-based process development

Resources, Conservation & Recycling 139 (2018) 122–139

Contents lists available at ScienceDirect

Resources, Conservation & Recycling

journal homepage: www.elsevier.com/locate/resconrec

Review

Hydrometallurgical recovery of metals from waste printed circuit boards (WPCBs): Current status and perspectives – A review

Huan Li[a], Jacques Eksteen[a,*], Elsayed Oraby[a,b]

Keywords: WEEE, Waste print, Hydrometal, Recycling, Review

Waste and Biomass Valorization (2020) 11:3897–3909
https://doi.org/10.1007/s12649-019-00780-0

ORIGINAL PAPER

An Alkaline Glycine-Based Leach Process of Base and Precious Metals from Powdered Waste Printed Circuit Boards

Elsayed A. Oraby[1,2] · Huan Li[1] · Jacobus J. Eksteen[1]

© Springer Nature B.V. 2019

Abstract

Resources, Conservation & Recycling 154 (2020) 104624

Contents lists available at ScienceDirect

Resources, Conservation & Recycling

journal homepage: www.elsevier.com/locate/resconrec

Full length article

Extraction of copper and the co-leaching behaviour of other metals from waste printed circuit boards using alkaline glycine solutions

Huan Li[a], Elsayed Oraby[a,b]

ARTICLE INFO

Keywords: Glycine, Base metals, Extraction, Waste PCB, WEEE

Waste Management 125 (2021) 87–97

Contents lists available at ScienceDirect

Waste Management

journal homepage: www.elsevier.com/locate/wasman

Cyanide consumption minimisation and concomitant toxic effluent minimisation during precious metals extraction from waste printed circuit boards

Huan Li[a], Elsayed Oraby[a,b], Jacques Eksteen[a,*]

[a] Western Australian School of Mines: Minerals, Energy and Chemical Engineering, Curtin University, WA 6102, Australia
[b] Mining and Metallurgical Engineering, Asyut University, Asyut 71515, Egypt

ARTICLE INFO

Article history:
Received 20 September 2020
Revised 29 December 2020
Accepted 17 February 2021

ABSTRACT

Waste printed circuit boards (WPCBs) constitute a hazardous material with up to 40 different metals, including numerous many heavy metals and environmentally harmful metals. Most hydrometallurgical processing approaches use high concentrations of toxic reagents and generate significant amounts of harmful effluents. This research investigates the use of cyanide-starved glycine solution containing no

Hydrometallurgy 199 (2021) 105540

Contents lists available at ScienceDirect

Hydrometallurgy

journal homepage: www.elsevier.com/locate/hydromet

Recovery of copper and the deportment of other base metals from alkaline glycine leachates derived from waste printed circuit boards (WPCBs)

Huan Li[a], Elsayed Oraby[a,b], Jacques Eksteen[a,*]

ARTICLE INFO

Keywords: Waste printed circuit boards, Copper recovery, Glycine, Hydrazine, Sulphide precipitation

Minerals Engineering 181 (2022) 107501

Contents lists available at ScienceDirect

Minerals Engineering

journal homepage: www.elsevier.com/locate/mineng

Extraction of precious metals from waste printed circuit boards using cyanide-free alkaline glycine solution in the presence of an oxidant

Huan Li[a], Elsayed Oraby[a,b], Jacques Eksteen[a,*]

ARTICLE INFO

Keywords: Non-cyanide, Glycine, Precious metals, Waste printed circuit boards, E-waste

Resources, Conservation & Recycling 187 (2022) 106631

Contents lists available at ScienceDirect

Resources, Conservation & Recycling

journal homepage: www.elsevier.com/locate/resconrec

Full length article

Development of an integrated glycine-based process for base and precious metals recovery from waste printed circuit boards

H. Li[a], E.A. Oraby[a,b,*], J.J. Eksteen[a]

[b] Mining and Metallurgical Engineering, Asyut University, Asyut 71515, Egypt

ARTICLE INFO

Keywords: Glycine, Ammonia, Copper, Precious metals, Waste printed circuit boards

ABSTRACT

Hydrometallurgy 216 (2023) 106008

Contents lists available at ScienceDirect

Hydrometallurgy

journal homepage: www.elsevier.com/locate/hydromet

Review

Amino acids as lixiviants for metals extraction from natural and secondary resources with emphasis on glycine: A literature review

Huan Li[a], Zixian Deng[a], Elsayed Oraby[a,b], Jacques Eksteen[a,*]

[a] Western Australian School of Mines: Minerals, Energy and Chemical Engineering, Curtin University, GPO Box U1987, Perth, WA 6845, Australia
[b] Mining and Metallurgical Engineering, Asyut University, Asyut 71515, Egypt

ARTICLE INFO

Keywords: Glycine, Leaching, Recovery, Precious metals, Base metals

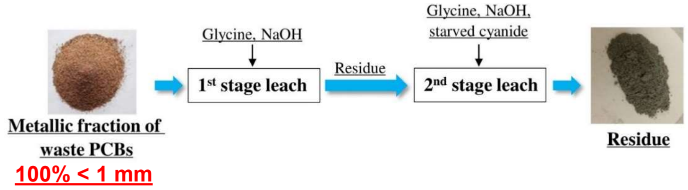
ABSTRACT

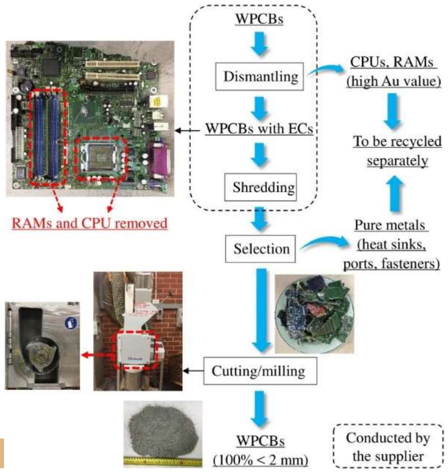
The past decade has seen significant interest in non-conventional lixiviants, particularly the growing interest in organic acids and amino acids. Of the amino acids, glycine has much coverage in the hydrometallurgical and mineral processing literature as a reagent that is non-toxic, non-volatile, and recyclable. The glycine leaching system (i.e., an aqueous system typically consisting of glycine or one of its alkali metal salts, a pH modifier, an oxidant and sometimes, other catalysts or modifiers) has been shown effective in selectively extracting base and precious metals from various resources. Using a glycine leaching system can, therefore, be considered a promising substitute for traditional methods in the hydrometallurgical industry. This paper reviews the application of amino acids and emphasizes the glycine leaching system to extract valuable metals from natural and secondary

---

Development of glycine-based process

Preliminary flowsheet:





Simple pre-treatment:

- ☐ PCBs source: PC motherboards
- ☐ Electronic components remained
  - Except high-values
- ☐ No complicated physical processing

	Metal content and economic value										
	Metal content in wt. %						Metal content in ppm				
	Cu	Fe	Al	Sn	Pb	Zn	Ni	Co	Au	Ag	Pd
Metal content	22.60	1.83	3.18	2.81	0.34	0.69	865.50	59.72	106.77	170.50	10.37
Metal value (%)	18.2	-	0.8	6.7	0.1	0.2	0.2	0.03	66.4	1.2	6.2

Source: Oraby, E.A., Li, H. and Eksteen, J.J., 2019. Waste Biomass Valori., 11(8): 3897-3909.
Li, H., Oraby, E., and Eksteen, J., 2020. Resour. Conserv. Recy., 154:104624.

Development of glycine-based process

Glycine-NaOH leaching of base metals:

- Restrained by solids%: <2-5% – waste PCBs are highly metal-rich!
- Slow leaching kinetics: ~3 days to complete copper leaching

Metals recovery from leachates:

Method	Impurity	Conditions required	Kinetics	Co-precipitation	Product
Hydrazine (N_2H_4) reduction	None (N_2 and H_2O)	pH > 12 Cu > 5g/L	>90% Cu recovery within 0.5-2 hours	Sn, Pb	Cuprite (Cu_2O) or metallic copper
Sulfide precipitation	Sulfur	Cu/HS^- molar ratio ≤1:1.2	Fast kinetics: >90% copper recovery within 5 min	Zn, Pb, Sn, Ni	Covellite (CuS) at 87.9% purity
Solvent extraction (Mextral 54–100 and Mextral 84H)	-	15% Mextral 84H & 200 g/L H_2SO_4; 30% Mextral 54–100 & 80 g/L H_2SO_4	85-95% Cu extraction; 99% Cu stripping	Ni (Pb and Sn not investigated)	Metallic copper or copper sulfate after downstream treatment

Source: Li, H., Oraby, E., and Eksteen, J., 2021. Hydrometallurgy, 199: 105540.
Li, H., Oraby, E., and Eksteen, J., 2020. Resour. Conserv. Recy., 154:104624.

Development of glycine-based process

Glycine-ammonia leaching of base metals:

- Solids% increased to 15%~: >40 g/L copper in PLS
- Copper leaching shortened to 24 hours
- Ammonia concentration reduced by ~80%, compared with conventional ammoniacal leaching
- Ammonia played the key roles of:
  - ✓ pH modifier
  - ✓ Synergist
  - ✓ Copper stabilizer

Cyanide-starved glycine leaching of precious metals:

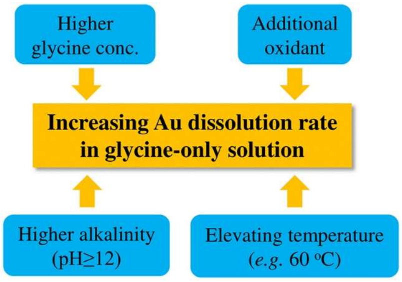
- Cyanide acted as “catalyst” for copper and silver dissolutions
- Similar or better PMs extraction, compared with stoichiometric or intensive cyanidation
- Cyanide use was reduced by 70-90% (250 ppm *vs* 3500 ppm, no free cyanide after 4 hours)

Source: Li, H., Oraby, E., and Eksteen, J., 2022. Resour. Conserv. Recy., 187: 106631.
Li, H., Oraby, E., and Eksteen, J., 2021. Waste Manage., 125: 87-89.

Development of glycine-based process

Non-cyanide glycine leaching of precious metals:

- Permanganate or ferricyanide with glycine solution effectively leached PMs, namely glycine-oxidant leaching system
- Similar or comparable PMs extraction, compared with cyanide-starved glycine leaching or intensive cyanidation
- Rapid decomposition of glycine was observed when permanganate added
- Oxidant consumptions were high, i.e., 630 kg/t K-permanganate or 610 kg/t K-ferricyanide



Higher glycine conc.

Additional oxidant

Increasing Au dissolution rate in glycine-only solution

Higher alkalinity ($pH \geq 12$)

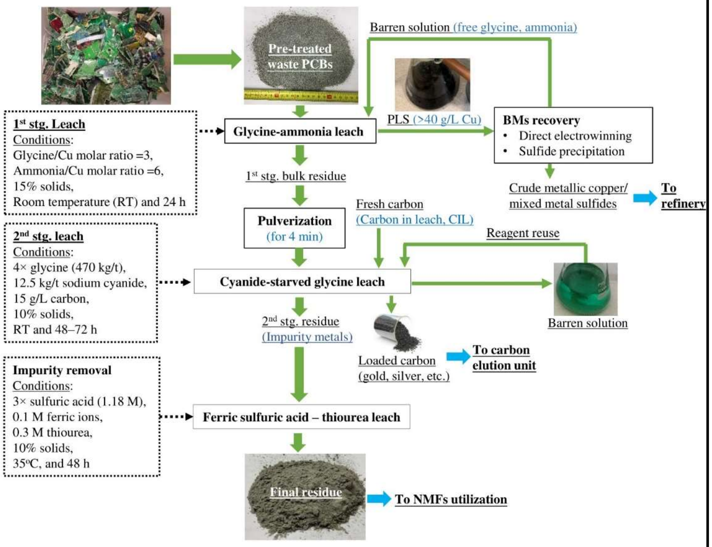
Elevating temperature (*e.g.* 60 °C)

Source: Li, H., Oraby, E., and Eksteen, J., 2022. Minerals Engineering, 181: 107501.

Development of glycine-based process

An improved process:

- 1st step glycine-ammonia leaching at 15% solids
- Pulverization used to liberate precious metals from plastics
- 2nd step cyanide-starved glycine leaching at 10% solids
- 3rd step sulfuric acid – Fe^{3+} – (thiourea) leaching at 10% solids

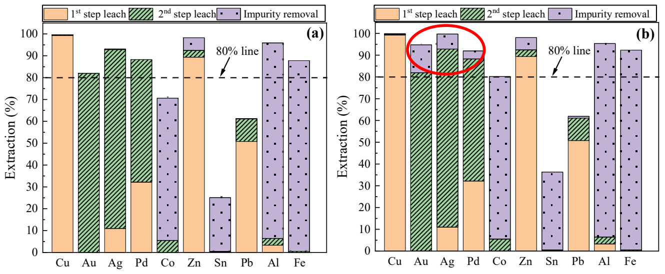


Source: Li, H., Deng, Z., Oraby, E., and Eksteen, J., 2023. Resour. Conserv. Recy., 187: 106631.

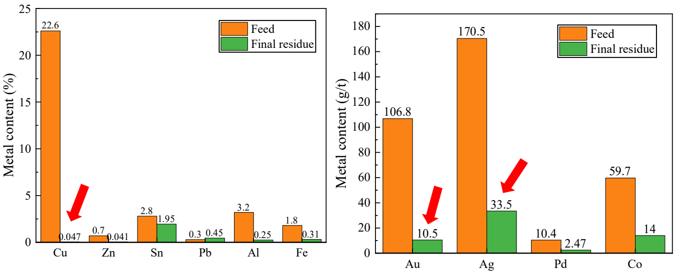
Development of glycine-based process

An improved process:

- Overall extraction: >99% copper, >90% gold, >95% silver and >85% palladium
- Metal content in residue: 0.047% copper, <1% base metals (except tin), <35 ppm precious metals



Figures above: Extraction of metals at each step of leaching, (a) without and (b) with thiourea in the 3rd step.



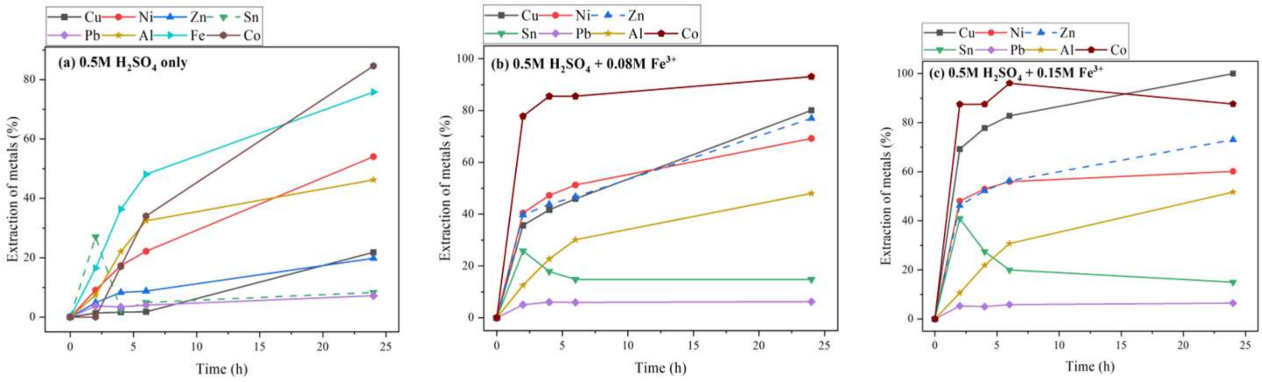
Figures above: Metal contents in the final residue after three leaching steps with thiourea in the 3rd step.

Source: Li, H., Deng, Z., Oraby, E., and Eksteen, J., 2023. Resour. Conserv. Recy., 187: 106631.

Development of sulfuric acid leaching process

Leaching of pre-treated waste PCBs (100%< 2mm):

- Oxidant, e.g., Fe^{3+}, was essential for copper leaching
- Leaching time can be shortened to ~24 hours
- Tin and lead (e.g., solder materials) remained low extractions (<20%)
- No gold and palladium were leached; silver leaching was <15%.

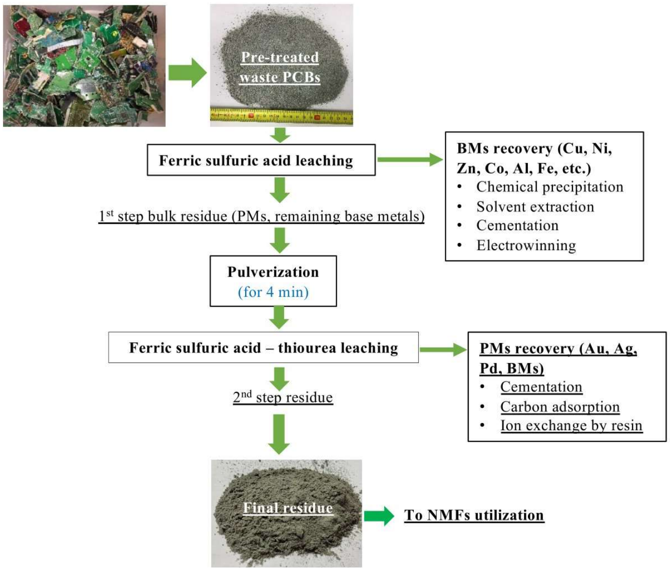


Figures above: Extraction of metals using different sulfuric acid leaching at 2% solids and room temperature.

Development of sulfuric acid leaching process

Sulfuric acid leaching process:

- Well known in chemistry
- Non-selective for base metals
- 1st step sulfuric acid – Fe^{3+} leaching of copper & base metals
- 2nd step sulfuric acid – Fe^{3+} – thiourea leaching of precious metals & remaining base metals
- Physical separation is suggested prior to hydrometallurgical leaching



Pre-treated waste PCBs

Ferric sulfuric acid leaching

BMs recovery (Cu, Ni, Zn, Co, Al, Fe, etc.)
- Chemical precipitation
- Solvent extraction
- Cementation
- Electrowinning

1st step bulk residue (PMs, remaining base metals)

Pulverization (for 4 min)

Ferric sulfuric acid – thiourea leaching

PMs recovery (Au, Ag, Pd, BMs)
- Cementation
- Carbon adsorption
- Ion exchange by resin

2nd step residue

Final residue

To NMFs utilization

Recommendations for future study

- More fundamental studies
- Mechanical-physical separation of bulk of metals
- Ferricyanide regeneration and reuse
- Glycine recycling and reuse
  - Glycine decomposition
  - Glycine quantification
- Residue utilization
- Pilot scale tests
- Techno-economic assessment

Acknowledgements

GOVERNMENT OF
WESTERN AUSTRALIA

NEW
INDUSTRIES
WA

total green
recycling

Curtin University

MINTEK’S IGOLI™PROCESS. HISTORY, PILOTING AND TECHNO ECONOMICS.

By

Elmar Muller and Sazini Makamu

Mintek
200 Malibongwe Drive
Randburg
South Africa

Presenter and Corresponding Author

Elmar Muller
elmarm@mintek.co.za

ABSTRACT

South Africa has an extensive abandoned mine problem, with the country having more than 6000 abandoned mines according to Statistics SA. Although the SA Mine legislation (MPRDA s.43(1)) requires mining operators to make financial provision for mine closure and rehabilitation, it is not effectively enforced. This has lead to extensive illegal mining and the rise of criminal enterprises.

The South African government has identified artisanal mining as an initiative to help to alleviate unemployment, but the criminality and safe use of cyanide and mercury have severely curtailed this initiative.

The iGoli™ project, which started at Mintek more than 20 years ago with its main purpose as an alternative technology to using mercury for gold recovery, has come a long way and its scope and applicability has increased.

The project has expanded and now covers:

- Alternative process to using mercury in gold recovery,
- Treatment of mine tailings,
- New life for abandoned mines,
- Assisting artisanal miners,
- Linking other Mintek technologies and
- Initiating down stream processes.

This paper will discuss the latest developments in the iGoli™ process as it applies to piloting for the treatment of mine tailings, techno economic evaluation as well as other Mintek’s offerings to assist in addressing the abandoned mine problem together with socio-economic upliftment of disadvantaged communities.

The benefits to the community includes job creation and the freeing of previously locked land for use while the SA government benefits through the reduction of the following adverse effects of abandoned mines and dumps:

- Environmental impact from dumps,
- Illegal mining,
- AMD,
- Locking of land and
- Loss of revenue
- Unemployment (Due to mine closure).

Keywords: iGoli™, illegal mining, artisanal mining, abandoned mines, socio-economic.

INTRODUCTION

South Africa has a rich mining history that started in 1852 when the first mine, a copper project, was constructed in what is today the town of Springbok, in the Northern Cape province[1].

This was followed by the boom in the diamond industry when 15-year-old Erasmus Stephanus Jacobs discovered South Africa's first diamond, the Eureka, in Hopetown in 1867 and kickstarted what historians call the Mineral Revolution[1].

Although small gold alluvial deposits were found near Pilgrim's Rest and Barberton in 1873, these were only precursors to the biggest discovery of all, the Main Reef on Gerhardus Oosthuizen's farm Langlaagte, Portion C, in 1886. This led to the Witwatersrand Gold Rush with mining starting in Ferreirasdorp in the same year which led to the establishment of Johannesburg.

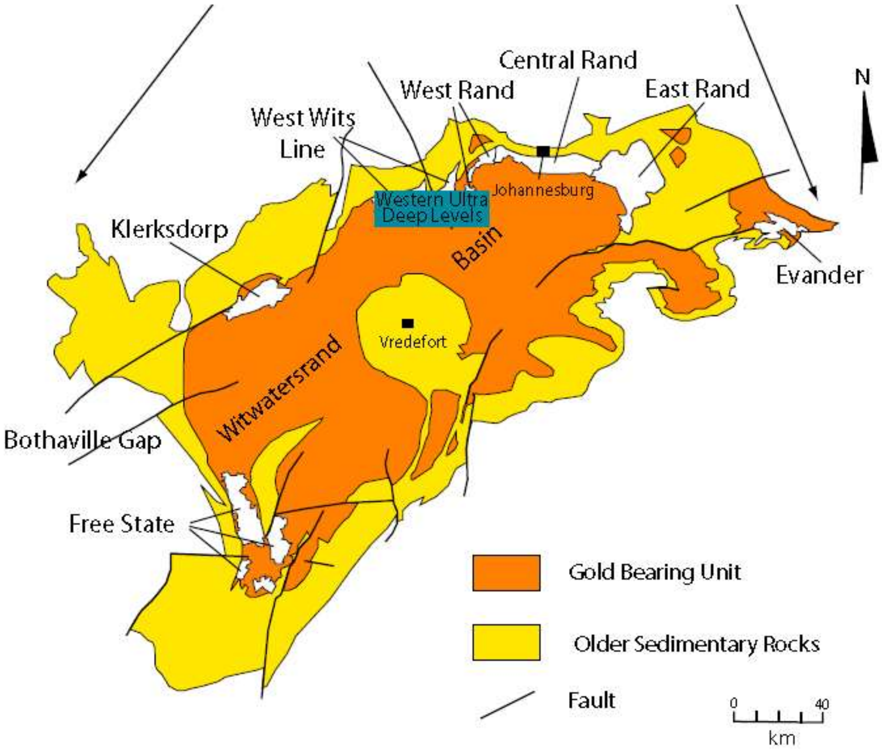


Figure 1: Map of the Witwatersrand basin[2].

Today, the mining industry remains one of the biggest contributors to South Africa's economy with an estimated yearly sales exceeding R850 Billion[3]. It is the world's fifth largest mining sector in terms of gross domestic product (GDP), contributing eight percent to South Africa's GDP.

According to the South African Council for GeoScience, there are more than six thousand (6000) abandoned mines in South Africa[4].

Planning for mine closure has been a requirement in South African legislation since the Minerals Act of 1991 came into effect, which required that an environmental management programme (EMP) be drafted, rehabilitation be undertaken, financial provision made, and an application submitted for a closure certificate.

Under the current regime, the granting of mining authorizations, regulation for operations and the issuing of closure certificates are all governed by the Department of Mineral Resources and Energy (DMRE), This is predominantly managed through 3 pieces of legislation, namely the Mineral and Petroleum Resources Development Act (MPRDA, Act 28 of 2002),the National Environmental Management Act (NEMA, Act 107 of 1998 and various updates), the National Environmental Management: Waste Act (NEMWA, Act 59 of 2008). Other government departments (most notably Environmental Affairs and Water and Sanitation) and various other pieces of legislation have lessor roles to play but also come into play.

The MPRDA, the main source of rehabilitation obligations, "……requires rights holders to 'as far as reasonably practicable' rehabilitate the land affected by the operation 'to its natural or predetermined state, or to a land use which conforms to the generally accepted principle of sustainable development' (Section 38 (1) (d))………".

Regulation 62 of the MDPRA regulation of 2004 require the submission of closure plan and objectives as part of the draft Environmental Management Programme (EMP) / Environmental Management Programme Report (EMPR) and which must contain '…the key objectives for mine closure to guide the project design, development and management of environmental impacts'; future land use objectives for the site, and the costs of closure...'(Reg 61 (1)).

Unfortunately, implementation and execution of these EMPs and submissions of the EMPRs are not effectively being enforced.

ISSUES

The South African Government inaction and large scale unemployment in the country have lead to a parallel mining industry in the country where illegal, artisanal and small scale miners (ASMs) are working these abandoned mines and to a lesser extent the associated mine waste - tailings dumps.

The South African Institute of Security Studies estimates that, only in the Gold industry, about 30 000 illegal miners produce R14-billion of gold a year. From the state's perspective this is "lost production".

The illegal miners live and mine under extremely hazardous conditions and stay underground for weeks fearing rock falls, government/police interventions and other highly armed gangs, while the masterminds and king-pins live in luxury (Figure 9). These miners still use cyanide and mercury-based technologies to recover the gold with extremely primitive equipment (Figures 6 – 8).

The Middle East and Central Europe have been identified as the primary export destinations.

Figure 2: Entrance to an abandoned mine.

Figure 3: Entering an abandoned mine[5].

Figure 4: Preparing meals underground.

Figure 5: Living conditions underground.

Figure 6: Gold bearing ore[5].

Figure 7: Gold processing in an informal settlement[6].

Figure 8: Gold processing in an informal settlement continued[6].

Figure 9: Suspected illegal mining kingpin[7].

Although Mining has and will continue to make a significant contribution to South Africa’s economy and livelihoods, it is also associated with various other negative impacts. One such impact is acid mine drainage (AMD) caused by effluents from waste rocks and tailings.

AMD occurs when sulphide rich minerals, in the presence of catalysing bacteria such as Thiobacillus ferroxidases are exposed to air and water and are oxidized to form sulfuric acid. The sulfuric acid, which is generated easily dissolves metals such as iron, copper, aluminium and lead. This problem is more common in mine dumps where proper mine closure and reclamation procedures did not take place, as discussed above.

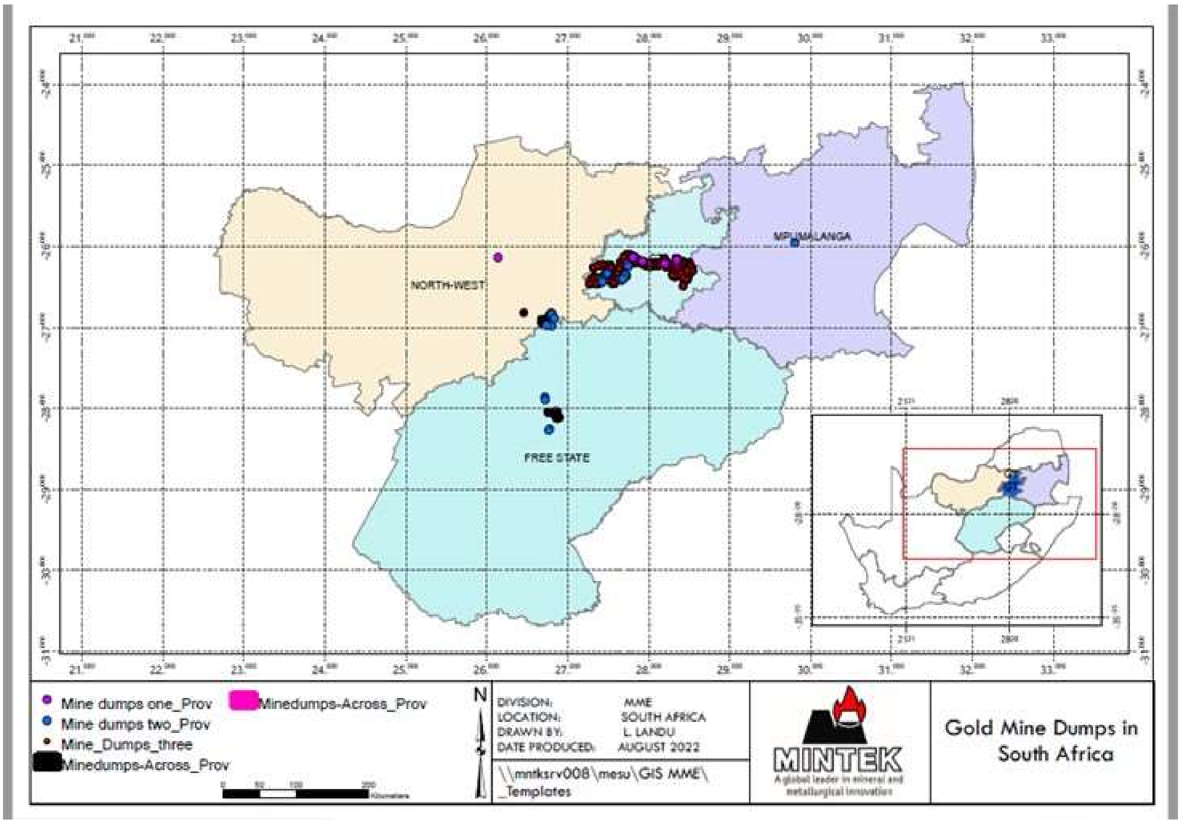


Figure 10: Location of the largest historic ownerless dumps in South Africa.

In a first step to understand the extent of the problem, the DMRE has embarked on a process of mapping the ownerless and derelict mines including mine dumps with Mintel providing the GIS services.

A further strategy that was put forward by the SA Minerals Council to the DMRE to reduce illegal mining in these areas (Figure 10), is to formally register co-operatives of miners (ASMs) and incorporate them into the existing mine work force. Working together with the Government, the idea is that some large scale mining (LSM) operations, can enter into agreements with the ASM operators for the latter to process tailings within the mine facilities. Using technologies like iGoli™, the ASM are afforded an opportunity to process not only historical tailings but also those continually produced through current LSM activity and sell back the concentrates to the LSM.

This approach is under investigation by the DMRE but again progress is slow due to the distrust in Government and many of these ASM are living illegally in South Africa.

TECHNOLOGY DESCRIPTIONS

The iGoli™ Technology package (the Package) now includes three technologies developed by Mintek to provide an integrated solution for rehabilitation of urban mined waste. The current piloting will also benchmark the Package against recovery data from local ASMs to illustrate the value of the Package for the treatment of abandoned mines and dumps.

Various pilot campaigns are planned, and the information obtained from the demonstration campaigns will aid in determining the most feasible method for processing the tailings and/or assisting the ASMs where abandoned mines are being reworked. Depending on the ore/tailings composition, it may be possible to create mobile plants using iGoli™ or all three technologies in the Package.

An added complication is that the abandoned mines and dumps in South Africa have varying gold, sulphide, and uranium concentrations. In addition, there is no assurance that these waste streams will react similarly to the treatment procedure. Consequently, process mineralogy is also important and Mintek is offering assistance to legal ASMs.

There are various reasons why the acceptance of the technology into the informal/small scale mining sector has been slow. As discussed above, Government impassiveness, overall distrust in Government and that many of the ASMs are illegal hinder the process. Mintek also has to carry part of the blame as Mintek previously failed to provide adequate training and technology support and did not efficiently engage the local communities. Finding a local partner with local knowledge and being trusted by the local community proved key in going forward.

Mintek also refocused their effort and over the last five years improved the process, made the process work with the simplest and cheapest materials of construction and decreased the operating costs. This enhancement of the iGoli process was performed in collaboration with BGRIMM and two South African universities as part of a larger research project that was initiated in 2018. South Africa and the People's Republic of China launched the South Africa-China Joint Research Centre for the Development and Utilisation of Mineral Resources (JRC-MR). This is an extensive program for the treatment of gold mine tailings to recover value and covers various aspects including communition, flotation, leaching and tailings stabilization and secondary use of the tailings as building materials.

iGoli™

As far back as 2001, Mintek started the development of hydrochloric and chlorine gas-based technology to recover gold. Although the original focused development on gold recovery from concentrate, this was expanded into the iGoli project to remove the use of cyanide and mercury in the recovery of gold as applied by ASMs.

The iGoli™ technique involves leaching gold concentration with diluted hydrochloric acid and bleach. The gold and base metals in the leach liquor are then reduced through precipitation to generate a metallic base metal and relatively pure gold powder. The reasonably pure metallic gold powder is created by reducing the gold in the leaching fluid with sodium meta-bisulphite.

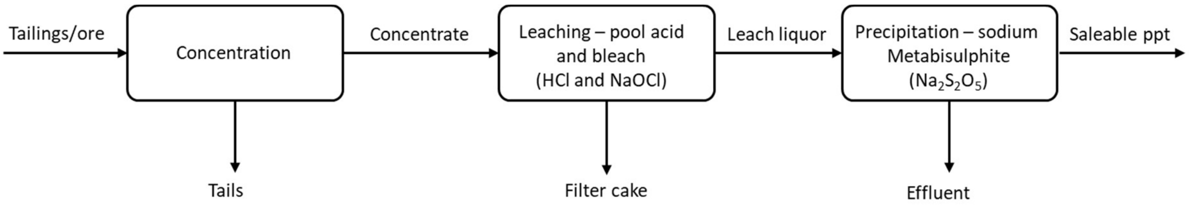


Figure 11: Simplified overview of the process flow.

Mintek sees iGoli™ is a sustainable process that could contribute to the global greening of the ASM gold sector.

The objective of the current pilot demonstration is to overcome the barriers of acceptance by demonstrating the technology (especially iGoli™) to potential users and industry partners. This demonstration was scheduled for 2021 but due to the Covid pandemic, it was delayed by more than 2 years.

ReFlAut™

This technology provide a feasible processing technique to process the mine tailings but also to address the environmental impacts of the tailings. The ReFlAut™ process developed by Mintek offers a feasible solution to address AMD in mine tailings while also promoting recovery of economic valuable minerals. The process comprises of re-processing of the gold mine dump material through a flotation circuit and producing uranium, sulphide and gold containing concentrates.

CloSURE™

Mintek's water programme is focused on developing both incremental and transformative technologies, products, processes and services to address the ecological challenges of water usage, water treatment and waste management in the minerals sector as well as to find their application in other industrial and resource-intensive sectors. The legacy of AMD in South Africa has caused widespread contamination of river catchments. The AMD is typically characterized by high sulphate concentrations, in excess of 3 g/L, with relatively low concentrations of metals.

Biological treatment of mining effluents using sulphate reducing bacteria offers a cost-effective and sustainable alternative to conventional technologies for the treatment of effluents produced by the gold and coal mining industries. Mintek developed CloSURE™, a technology which employs biological processes to treat mine impacted water. The aim was to produce water that is fit for re-use in irrigated agriculture. CloSURE™ is suitable for small point sources in remote locations that lack services and infrastructure, such as legacy mines and mines after closure. The process consists of two stages, namely a biological sulphate reduction step followed by an oxidation step for sulphide removal and bio-sulphur production.

INTEGRATED PACKAGE

The flowsheet described in **Error! Reference source not found.**2 is a depiction of the entire process flow for the iGoli™ Technology package for the rehabilitation of abandoned mines and dumps. Apart from the recoverable gold, silver and base metals, the abandoned mines and dumps also contain a substantial amounts of AMD water. A further consequence of these dumps is that they lock-up soil and large portions of land. Through the combination of the Mintek technologies listed above these waste streams can be rehabilitated, recycled and reused.

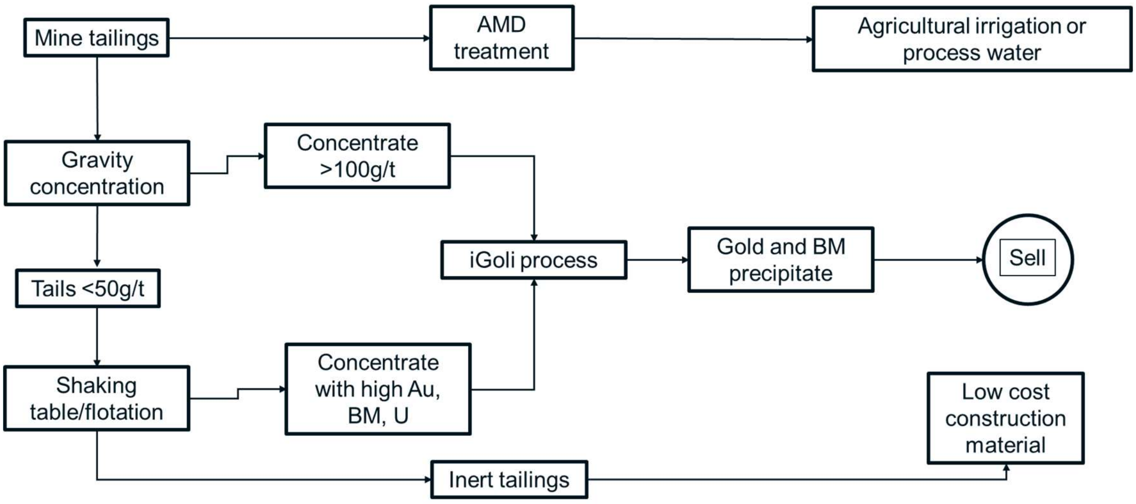


Figure 12: Flowsheet for the integrated package.

The current demonstration is based on the gold and base metal extraction process using iGoli™ for the concentrate and the ReFlAut™ flotation process of recovering gold, silver, uranium, sulphides and heavy metals from the tailings as shown in Figure 12. The final toxic free tailings will be collected and will be tested for stability and the manufacturing of a construction material via the JRC-MR collaboration. Laboratory testing has shown that this is a viable option for the treatment of the tailings waste.

The figure below is an updated plant design for the iGoli™ process and depicts the equipment that is being piloted.

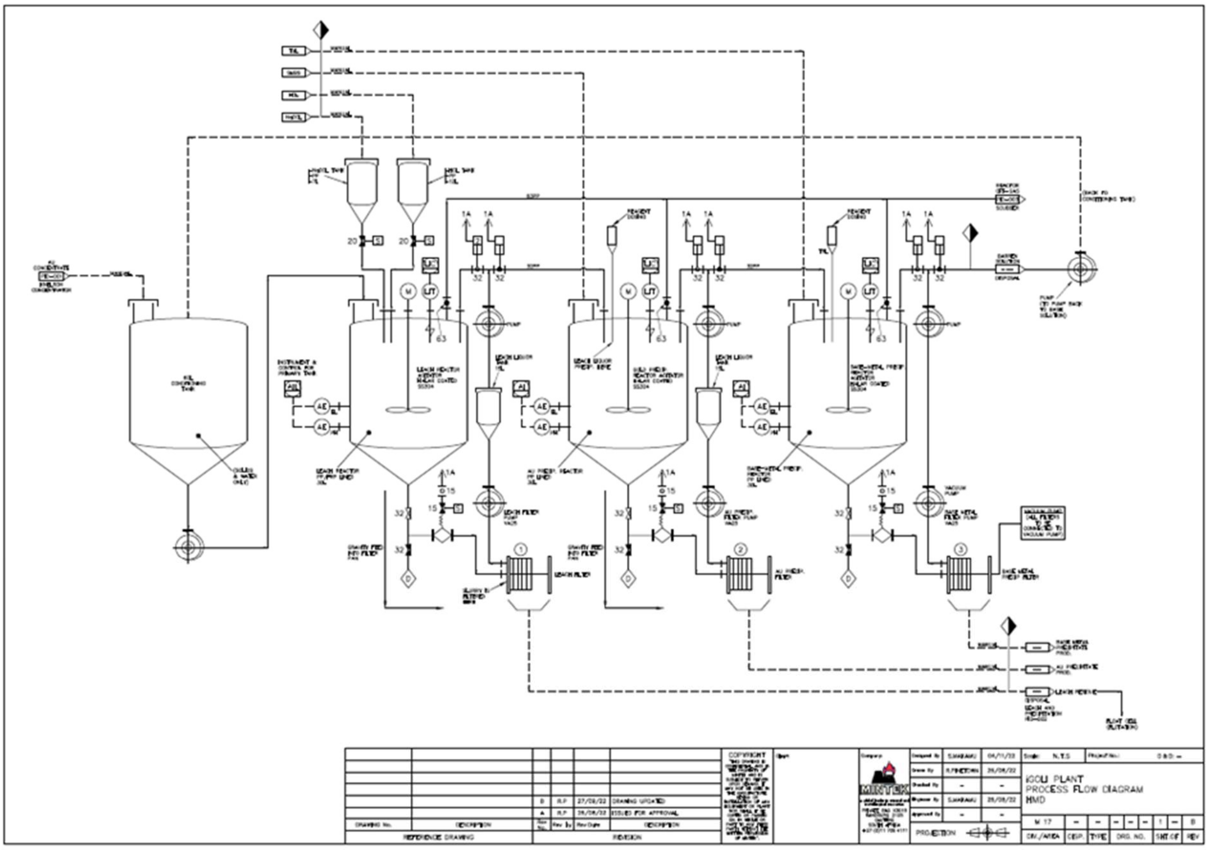


Figure 13: Improved iGoli™plant design.

PILOTING

Mintek is currently running a demonstration pilot campaign on-site in Barberton Mpumalanga, including commission and de-commissioning of the plant.

Although fairly remote, this site was chosen as the owners have a DMRE license to work the abandoned mine but also to rework the huge legacy mine dumps. In its heyday this area around Barberton was a key part of gold production in South Africa but has since fallen into hard times with many illegal mining operations.

The stakes are very high in the illegal mining business and the ASMs are ready to protect their income at all costs. Due the security concerns, the owners of the mine/dumps have approached the South African Police Service and the South Africa National Defense Force to assist in providing security.

Figure 14: Mintek project leader in discussion with the representatives from the SAPS and SANDF.

The objective of the demonstration plant is to produce gold salt and tailings as per the scheme in Figure 13. The intellectual property of iGoli™ will remain vested in Mintek, but the demonstration partner will have the right to use and implement the results from the report at the agreed technology licensing fee as will be determined in an official agreement between Mintek and the demonstration partner.

Below is a list of equipment that was installed for the pilot campaign:

- Leach reactor
- Leach residue filter and tank
- Gold precipitation reactor
- Gold precipitation filter and tank
- Gold precipitation reactor
- Gold precipitation filter and tank
- Peristalsis pumps for each reactor
- One vacuum pump with manifold for filtration attached to each filter pan
- Scrubber with manifold attached to each reactor

- pH/Eh meters for various stages
- Various motor and agitator for various tanks

Figures 15 to 18 below show the equipment being delivered and the plant being installed. As the various process steps are done at ambient conditions, plastic hardware can be used for most unit operations.

Figure 15: Equipment being delivered before installation.

Figure 16: “Tank farm” built by the demonstration partner.

Figure 17: Plant being installed.

Figure 18: Gold precipitation reactor in action.

Results and Techno Economics

The results reported below is for a pilot plant that is currently in operation at a site in Barberton Mpumalanga South Africa.

Although a small ReFlAut™ plant is on site to make the concentrate for the iGoli™ plant, the results reported here are for the iGoli plant only and the feed comes from the old gold tailings dumps.

The current cost of reagents and consumptions are presented in Table 1. The gold concentrate feed into the iGoli^{TM} plant is the product from the ReFlAut™ process where gold tailings were fed into the plant and upgraded to an average 120g/t concentrate.

The size of the plant is 20kg/hour but it is only being operated at 3kg/hour of concentrate for the first three months to establish the optimum operating parameters.

Table 1: Reagent Price and consumption per kilogram of a gold concentrate.

Plant flow)	Per hour	Per day	Cost/kg	Cost/day
	kg	kg	ZAR	
Solids	3	72		0.0
Water	17	408		0.0
HCl 15%	12.51	300.24	1.5	464.6
NaOCl 5%	36.00	864	1.5	1294.8
SMBS dry	0.11	2.592	13.0	33.7
NaOH dry	6.00	144	11.9	1717.6
Total cost of chemicals				**3510.8**

Table 2 below gives a summary of the first 4 campaigns specifically for the gold accountability. As expected, these gold recovery numbers show a very high gold recovery from the concentrate produced by ReFlAut™. The results indicate that the technology can be scaled from the laboratory to pilot plant.

As with all pilot campaigns the first three months had its fair share of breakdowns, equipment failures, clogging of sensors, etc. The issues were not unexpected and the remote area of also hampered bulk chemical delivery and logistics.

Table 2: Basic mass balance showing the Au "accountability".

#	Mass balance							
	IN Head	OUT Filtrate	OUT Precipitate	Au Ext by solution	Accountability	HCl	NaClO	Calculated Head
	Au	Au	Au	Au	Au	Based on feed		Au
	mg	mg	mg	%	%	L/kg	L/kg	g/t
1	46.80	45.0	0.3	96.2	95.6	0.067	0.120	5.2
2	46.80	45.7	0.4	97.7	96.9	0.227	0.133	5.2
3	46.80	45.7	0.4	97.7	96.9	0.067	0.120	5.2
4	46.80	55.4	1.8	118	114	0.267	0.200	5.2

Table 3 shows the estimated project income using the numbers for this pilot campaign The finances presented only show the income possible from a 3 kg/hr iGoli™ leaching plant using gold price and ZAR/dollar exchange averaged for April 2023. Although simplified, this income estimate does clearly show that the process flowsheet and size of equipment are economically feasible options as the breakeven point is reached after 10 months of continuous operation.

Table 3: Techno economics for the current pilot campaign.

Expected Gold grade concentrate		120	g/t
Gold recovery		95	%
Gold price		$ 60	per gram
Gold revenue		$ 492	per day
Exchange rate ZAR USD		$ 18	
Gold revenue		R 8,865	per day
CAPEX		R 749,694	
OPEX Labour		R 547,500	per year
OPEX chemicals		R 1,281,441	per year
Total upfront costs*		R 2,578,635	
Mintek costs*		R 571,512	
Gold revenue		R 3,235,594	12 months
Time required to breakeven (pay back period)		10	months

* The Mintek costs are related to support and is excluded for this calculation as this should be insignificant going forward.

CONCLUSIONS

One of the most important goals of this pilot plant demonstration of the previously developed iGoli^{TM} was to introduce the process to a new market in Barberton Mpumalanga. The demonstration has enabled Mintek to demonstrate the possibility of extending the life of gold mines as the pilot plant is running at an abandoned mine that was recently saved from illegal miners by local entrepreneurs.

Mintek is currently running this pilot plant to demonstrate the amenability of the plant but to also establish the design criteria for a larger medium-scale plant to provide a cost-effective metallurgical solution for the rehabilitation of abandoned mines and tailings dumps.

Although the construction and demonstration are continuing, the outcomes of the pilot plant demonstrations have already showed other local entrepreneurs that this initiative is worth their support.

The large scale unemployment and constant influx of illegal immigrants from all over Africa is putting a huge strain on the infrastructure in areas like Barberton and causing extensive sosio-economic unrest. Using this technology from Mintek and working with local entrepreneurs is leading to new enterprise development which will create jobs and business opportunities through urban mining and artisan processing of previously locked resources.

Mintek is also working with the DMRE to urgently speed up attempts to formalise artisanal mining as a livelihood strategy. The department is in discussions with the mining industry and civil society in this process to support entrepreneurs to license the mining of these abandoned mines and also to rework old tailing dumps to achieve the benefits as discussed in the Abstract.

Although progress is slow, a coordinated transnational effort to break the strangle-hold of the criminal syndicates must continue.

ACKNOWLEDGMENTS

The authors would like to thank Mintek and the local entrepreneurs of Barberton for the collaboration and provision of data to prepare this paper. The views of the authors do not necessarily present those of Mintek.

REFERENCES

1. https://www.chamberofmines.org.za

2. Taylor, C.D., Schulz, K.J., Doebrich, J.L., Orris, G.J., Denning, P.D., and Kirschbaum, M.J. - Geology and nonfuel mineral deposits of Africa and the Middle East, USGS Open-File Report 2005-1294-E.

3. Minerals Council of South Africa – Facts and Figures.

4. https://www.geoscience.org.za/index.php/projects-footer/159-derelict-ownerless-mines.

5. https://mg.co.za/news/ - various articles on illegal mining in South Africa.

6. https://www.groundup.org.za/article/lethal-toll-informal-gold-mining/ - various articles on illegal mining in South Africa.

7. https://www.iol.co.za/news/crime-and-courts/seven-suspected-illegal-mining-kingpins-arrested-in-the-west-rand.

THE DIVERSITY OF ReCYN TECHNOLOGY

Malcolm Roy Paterson

PT GreenGold Technology, Indonesia

Presenter and Corresponding Author

Malcolm Roy Paterson
malcolm@greengoldtechnology.com

ABSTRACT

GreenGold Technology (GGT) was formed in 2017 to commercialise ReCYN Technology, a process to recycle cyanide from gold plant tailings (ReCYN I) and recover metal from metal cyanide complexes for sale (ReCYN II). Both these objectives can result in a net revenue instead of the high cost of destructive detoxification methods. Clean-up of plant tailings with the ReCYN process also involved scavenging residual solubilised gold escaping the carbon circuit (ReCYN III).

The commercialisation has now reached the five-year milestone and is still not considered a proven technology, despite having three operational plants and two in the construction phase. One reason is the diversity of applications for which the technology is suitable, with each case requiring a customised solution. The "one size fits all" approach with carbon plants does not apply to ReCYN Technology.

GGT has completed over forty ReCYN plant studies, many with comprehensive testwork programmes and sometimes site testing. The result has been a different design for each case. This diverse applicability of ReCYN Technology has extended the commercialisation progress. Other impacts of design outcomes are determined by mine life, location, plant size, solution chemistry, reagent costs, existing detoxification costs, environmental requirements and economics.

The ReCYN process uses a macro reticular, anionic, strong base, polystyrene bead resin that is non-selective for the adsorption of most cyanide complexes. Activated carbon is more selective and therefore has a more straightforward process. The non-selective nature of the resin and specialist resin elution and recovery circuits gives the ReCYN process its unique diversity and advantage.

The range of applications includes the following:

1. Circuits that require high cyanide levels, such as silver-gold ores and concentrate leach circuits,
2. High cyanide soluble copper ores,
3. Favourable economics to replace detox,
4. Reducing the build-up of cyanide-consuming metals in recycled process water,
5. Removing contaminants deleterious to flotation efficiency,
6. Merrill-Crowe circuits to recover and recycle cyanide and zinc,
7. Removal of high iron and thiocyanate levels in tailings,
8. Gold scavenging from CIL circuit losses,
9. Increasing Reserves through the economic treatment of high copper-gold ores,
10. The recovery of PGE cyanide complexes,
11. Lowering the carbon footprint by reducing new cyanide purchases by 50%.

Each of the above examples has been the subject of studies conducted by GGT for specific projects, either operating or study phase. This Paper describes each application to demonstrate the diversity of the ReCYN process.

Three previous ALTA papers (2017, 2019, 2020) have described ReCYN I, ReCYN II and ReCYN III, including chemistry and flowsheet information.

Keywords: ReCYN, cyanide, gold, copper, resin, carbon, detoxify, recovery, economic and environmental.

History

- The ReCYN technology concept has been known for seventy years, coming out of the uranium industry in South Africa in the 1950's.
- As with most technology developments, the commercialisation stage is the hardest step.
- Signet were tasked with commercialising the Vitrokele process, a forerunner of ReCYN, in the period 1988 to 1996.
- A combination of many factors, including mixed project success and the loss of key personnel saw the technology shelved.
- A gold/silver project in Indonesia , including a cyanide recovery plant, had been put on hold due to the 1998 Asian Financial Crisis. The plant was commissioned in 2014 and re-established interest in the technology.

ReCYN Technology

- Following the success of a second cyanide recovery project in Indonesia in 2017, GreenGold was formed to commercialise the technology.
- The modified version of the technology was called the ReCYN process. There are four variants.
- **ReCYN I**, Cyanide Recovery.
- **ReCYN II**, Cyanide and Copper Recovery.
- **ReCYN III**, Cyanide and copper recovery and gold scavenging.
- **ReCYN IV**, Primary gold recovery.

ReCYN I, Cyanide Recovery.

ReCYN II, Cyanide and Copper Recovery.

ReCYN III, Cyanide and copper recovery and gold scavenging.

Naivety

- The four versions of ReCYN were meant to cover most applications. However, this was far from reality.
- Unlike carbon, the non-selective nature of the particular resin used, whilst seen as an advantage, has meant customization in most instances..
- The result has been a wide diversity of applications.

Diversity

- **Examples of the range of applications are:**

1. Circuits that require high cyanide levels, such as silver-gold ores.

Gold ores with a high soluble silver content generally require a high free cyanide concentration. The Mirah Project in Central Kalimantan had average silver levels of 100g/t. The project was made economic by using ReCYN for cyanide recovery.

2. High cyanide soluble copper gold ores.

Excessive cyanide consumption due to copper is a common problem in gold projects. Besides increasing operating costs there can be downstream problems with copper recirculating in process water. A ReCYN II circuit can turn a copper problem into an opportunity.

Diversity

3. Favourable economics to replace detox.

Detoxification of gold plant tailings to meet compliance levels can be expensive. The ReCYN process can replace destructive detox methods and give a positive economic return with cyanide recycling and copper production.

4. Reducing the build-up of cyanide-consuming metals in recycled process water.

If dissolved metals are not removed from process water they can build up and cause increased cyanide consumption, sometimes impacting gold recovery and negatively impact flotation circuits.

Diversity

5. Used in Merrill-Crowe circuits to recover and recycle cyanide and zinc.

The addition of zinc metal in an M-C circuit adds to increased cyanide and detox costs. The inclusion of a ReCYN circuit can reduce capital and operating costs. Recycling of both cyanide and zinc metal is a significant benefit and reduces detox costs.

CIP circuits are sometimes added to an M-C flowsheet to scavenge gold. A ReCYN plant can replace the carbon plant and simultaneously recover cyanide and zinc and replace the detox circuit.

Diversity (Continuing)

7. Removal of high iron and thiocyanate levels in tailings,
8. Gold scavenging from CIL circuit losses,
9. Increasing Reserves through the economic treatment of high copper gold ores,
10. The recovery of PGE cyanide complexes,
11. Lowering the carbon footprint by reducing new cyanide purchases by 50%.

Commercialisation

- GGT are now five years into the commercialisation of ReCYN Technology.
- With three operations having several years of track record and two plants under construction, ReCYN is still not considered proven technology.
- The reason is partly due to the conservative nature of the mining industry and partly due to the diversity of the technology.
- We probably have another five years to go before ReCYN is seen as a proven technology, but we are well into the process, albeit increasing the diversity!

ALTERNATIVE GOLD REFINIG PROCEDURE USING "ORGANIC AQUA REGIA"

By

[1]Akihiro Yoshimura, [2] Pariya Torkaman, [1]Yasunari Matsuno, [2]Marcello M Veiga

[1]Chiba University, Japan
[2]University of British Colombia, Canada

Presenter and Corresponding Author

Akihiro Yoshimura
a.yoshimura@chiba-u.jp

ABSTRACT

Artisanal small-scale gold mining (ASGM) is one of the most important incomes for miners in developing countries. Mercury (Hg) is mainly used for the extraction of gold (Au) from ore in the ASGM process. The use and emission of Hg cause environmental impact and health damage, and the Minamata Convention on Mercury propose the reduce and elimination of the use and emission of Hg from ASGM. However, as mentioned above, ASGM is very important income for the miners including their families and the developing countries. An alternative procedure is required to balance the reduction or elimination of Hg use and income keeping.

The authors suggest the application of organic aqua regia (OAR) for the refining of Au. This OAR consists of an organic solvent containing copper halide (CuX_2, X= Br or Cl). In the organic solvent like dimethyl sulfoxide (DMSO) or propylene carbonate (PC), the cuprous ion (Cu^+) is stable while the cupric ion (Cu^{2+}) is stable in water. The OAR can dissolve Au because Cu^{2+} liberated from CuX_2 performs as a strong oxidant. And dissolved Au can be recovered by the addition of water to OAR. The authors used this solvent for the recycling of Au from secondary sources like a waste of electric and electronic equipment (WEEE) and successfully recovered Au.

In this research, the authors applied this OAR for the gold refining from ore, considering the utilization at ASGM sites. We used $CuBr_2$ or $CuCl_2$ as an oxidant and potassium bromide (KBr) or sodium chloride (NaCl) as a supplier of halogen elements. The simulated ore which is a mixture of sand and Au wire and ore containing 48 ppm of Au was treated by Br or Cl type OAR and the authors evaluated the extraction ratio from the ore under an ambient temperature or 40 °C. After the extraction, the authors evaluated the recovery of Au by the addition of acidic water to the solvent. Sulfuric acid (H_2SO_4) or lemon juice were used for this recovery.

As a result, the authors successfully extracted most of the Au from ore in every case. The ore used in this research was hard to treat by the conventional ASGM procedure using mercury. Thus, the OAR can be a candidate for the effective extractor of Au. In addition, extracted Au was recovered by adding acidic water like sulfuric acid or lemon juice. Considering the availability at ASGM sites, lemon juice is suitable for the precipitant media.

However, the cost of this procedure was much higher than the conventional ASGM process. Optimization is required for the spreading of the OAR process.

Keywords: Organic aqua regia, artisanal small-scale gold mining, Minamata Convention on Mercury

INTRODUCTION

Gold (Au) mining is an important industry for many countries especially developing countries of its high price. In such countries, artisanal small-scale gold mining (ASGM) is mainly operated for the extraction of Au from ore. Mercury (Hg) is mainly used for extraction in this procedure and this use causes environmental impact and health damage. In addition, cyanidation is also used together to extract Au and this causes mercury cyanide ($Hg(CN)_2$) which has strong toxicity for human health and the environment (1). Minamata Convention on Mercury was proposed for the reduction and elimination of Hg use in ASGM (2). However, as mentioned above, ASGM is a very important income for the miners including their families and the developing countries. An alternative procedure is required to balance the reduction or elimination of Hg use and income keeping.

Considering this situation, the authors suggest the application of organic aqua regia (OAR) for the refining of Au This OAR consist of organic solvent, e.g. dimethyl sulfoxide (DMSO) or propylene carbonate (PC) as a solvent and copper halide (CuX_2, X = Br, Cl) as an oxidant. The liberated cupric ion (Cu^{2+}) from CuX_2 can be a strong oxidant and dissolve Au. In addition, dissolved Au can be recovered by the addition of water to OAR (3). The authors used this solvent for the recycling of Au from secondary sources like a waste of electric and electronic equipment (WEEE) and successfully recovered Au (4).

In this research, the authors applied this OAR for the gold refining from ore, considering the utilization at ASGM sites. We used $CuBr_2$ or $CuCl_2$ as an oxidant and potassium bromide (KBr) or sodium chloride (NaCl) as a supplier of halogen elements. First, the authors used the simulated ore which is a mixture or sand and Au wire by Cl type OAR to establish a fundamental procedure. After that, ore containing 48 ppm of Au was treated by Br or Cl type OAR and the authors evaluated the extraction ratio from the ore under ambient temperature or 40 °C. After the extraction, the authors evaluated the recovery of Au by the addition of acidic water to the solvent. Sulfuric acid (H_2SO_4) or lemon juice was used for this recovery.

MATERIALS AND METHODS

Materials

In this research, the authors used simulated and actual ores. Figure 1 shows the samples used in this research.

The simulated ore consists of sand and Au wire (ϕ = 0.20 mm) for the confirmation of a fundamental procedure of Au extraction and recovery. 20 – 200 g of sand was added for approx. 100 mg of Au to simulate 500-5,000 ppm of ore.

The authors also used an actual ore from La Maria, Colombia. This ore consists of quartz (SiO_2), pyrite (FeS_2), chalcopyrite ($CuFeS_2$) and periclase (MgO) as rock contents. In addition, 48.9 $\pm$ 6.30 ppm of Au was contained. In the previous tests, 83.7% of Au was extracted by the cyanidation process (5).

The OAR of this research consists of DMSO as a solvent and CuX_2 as an oxidizing agent. In addition, potassium bromide (KBr) was added for $CuBr_2$ solution ($CuBr_2$-KBr OAR) and sodium chloride (NaCl) was added for $CuCl_2$ solution ($CuCl_2$-NaCl OAR), respectively.

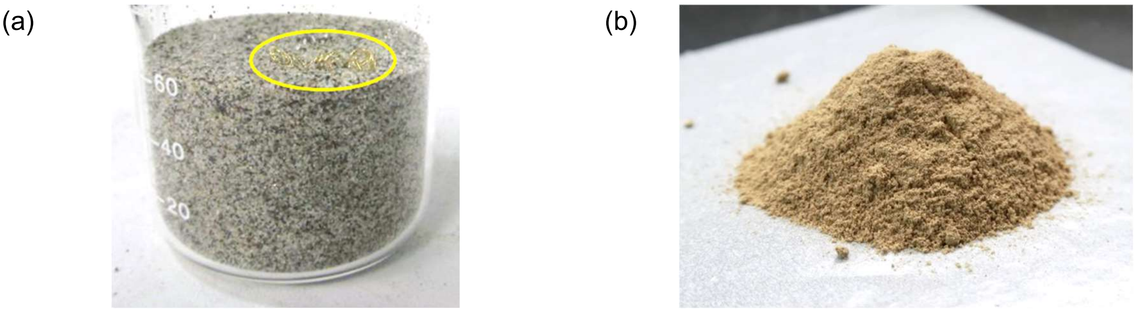


Figure 1: Used samples in this research, (a) the simulated ore and (b) the ore from La Maria

For the precipitation procedure, H_2SO_4, vinegar (CH_3COOH) and lemon juice (pure and half diluted) were selected precipitants.

Mechanism

In DMSO, the cupric ion (Cu^{2+}) and the cuprous ion (Cu^{+}) are stable. Dissolved CuX_2 in DMSO forms the complex of $[CuX_3]^-$ with halogen ion (X^-), while Cu^+ forms the complex of $[CuX_2]^-$. On the other hand, Au forms the complex of $[AuX_4]^-$ or $[AuX_2]^-$ in DMSO. Thus, Cu^{2+} ion can be a strong oxidant and dissolve Au by following reactions (6). In addition, as shown in Eqs. (1) and (2), an excessive amount of X- is required for the effective dissolution of Au. Thus, halogen compounds e.g. KBr or NaCl were added to the OAR.

$$Au + 3[CuX_3]^- + 4X^- \leftrightarrow [AuX_4]^- + 3[CuX_2]^- \quad (1)$$
$$Au + [CuX_3]^- + 2X^- \leftrightarrow [AuX_2]^- + [CuX_2]^- \quad (2)$$

On the other hand, reverse reactions will occur in water. Thus, after the Au dissolution, dissolved Au will be recovered by the addition of water to OAR. To prevent the precipitation of other metal compounds, especially Cu and Fe, acidic water like hydrochloric acid (HCl) or H_2SO_4 is preferable for the precipitation.

Methods

The proposed flow is shown in Figure 2. Au in ore was extracted by OAR treatment for several hours. In some cases, the authors tried to recover the dissolved Au. The detailed conditions of each method are described below.

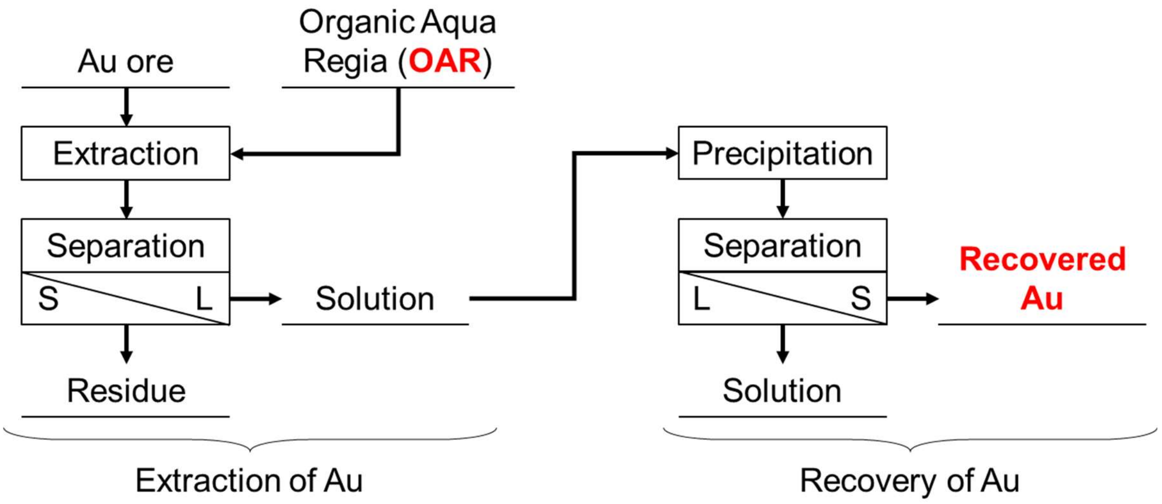


Figure 2: Experimental flow of this research

Treatment of the simulated ore

(1) Extraction of Au from ore
As described above, the authors used mixtures of 100 mg of Au wire and sand as samples. OAR volume was varied from 10 mL (for pure Au wire), 20 mL (for 5,000 ppm sample), 40 mL (for 2,500 and 1,000 ppm samples) to 80 mL (for 500 ppm sample), respectively. In every case, OAR contained 403 mg (3 mmol) of $CuCl_2$ and 117 mg (2 mmol) of NaCl.
As common conditions, the treatment temperature and time were set at 80 °C for 6.0 hours. The change of Au wire weight was checked on 0.5, 1.0, 1.5, 2.0, 3.0, 4.0 and 6.0 hours to confirm the dissolution amount by OAR treatment. The dissolution rate was compared with pure Au wire to evaluate the effect of the coexistence of sand.

(2) Recovery of extracted Au
After extraction, the OAR solution containing dissolved Au was separated from the Au sample residue. H_2SO_4 was added to this solution and obtained the precipitation. The recovery ratio of Au was calculated from Au content in the precipitation by SEM-EDS analysis and its weight.

Treatment of the ore from La Maria

(1) Extraction of Au from ore
In this research, the authors used $CuBr_2$-KBr or $CuCl_2$-NaCl OAR as the solvent. In the fundamental treatment, the authors used a high temperature (80 °C) to achieve the high-speed dissolution of Au, however, it causes an extra cost to keep the temperature at ASGM sites. Considering this, the treatment temperature was set as lower temperature.
In the extraction experiments, the solid-liquid (S:L) ratio, the concentration of solvents and treatment time were evaluated. The detailed conditions are following:

- $CuBr_2$-KBr OAR
  20 g of ore was treated with 50 mL or 100 mL of OAR. This OAR contained 0.82-3.27 g of $CuBr_2$ and 0.29-1.16 g of KBr. The treatment temperature was 40 °C and the treatment time was 4-24 h, respectively.

- $CuCl_2$-NaCl OAR
  10 g of ore was treated with 3- or 10-times volume of OAR. This OAR contained 1.34-2.7 g of $CuCl_2$ and 0.6-1.16 g of NaCl. The treatment temperature was ambient and the treatment time was 8 or 24 h, respectively.

(2) Recovery of extracted Au
In the case of $CuBr_2$-KBr OAR treatment, dissolved Au was recovered by the addition of acidic solvents referring to the existing research (6). As a fundamental procedure, enough amount of H_2SO_4 solution was added. However, it is hard to purchase and use such chemicals at ASGM sites. Considering this, the authors tried to use CH_3COOH and lemon juice which are easy to purchase and use for the precipitation of extracted Au.

EXPERIMENTAL RESULTS

Extraction of Au from Ore

$CuCl_2$-NaCl OAR for the simulated ore

Figure 3 shows the dissolution rate of Au from ore by $CuCl_2$-NaCl OAR for the simulated ore. As shown, the dissolution rate of Au from simulated ore decreased compared with pure Au wire. On the other hand, except 500 ppm sample, all or most of added Au were dissolved within 6 h. In addition, considering the result, all of Au can be dissolved by the continuous treatment from the 500 ppm sample.
From these results, the coexisting sand may not affect the Au dissolution in the case of pure Au exposed on the ore surface. And these results suggest the importance of pre-treatment of ore, e.g., grinding or crushing before the extraction by OAR.

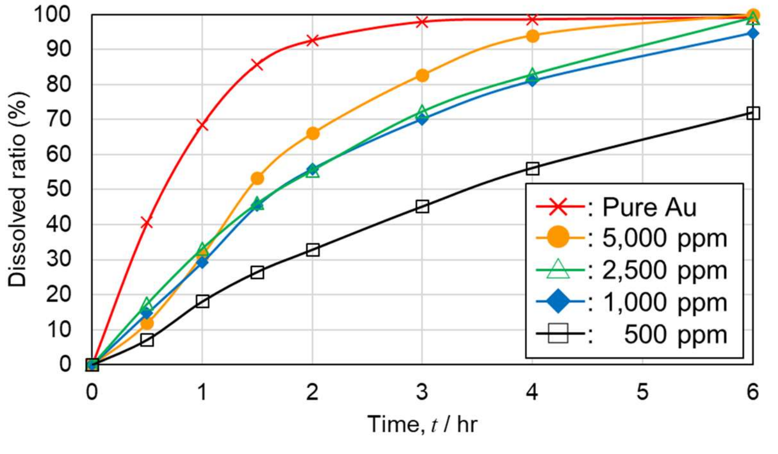


Figure 3: Dissolution rate of Au from the simulated ore by $CuBr_2$-KBr OAR

$CuBr_2$-KBr OAR for the Ore from La Maria

Figure 4 shows the extraction ratio of Au by $CuBr_2$ (3.27 g) - KBr (1.16 g) OAR treatment for the ore from La Maria. As shown, approx. 100% of Au was extracted by over 12 h treatment at 40 °C.
Au dissolution speed decreased compared with the simulated ore, however, all of the Au content was extracted even the concentration of Au was significantly smaller than the simulated ore.
In addition, all of Au was extracted from the ore in the case of 100 mL of OAR, even if the concentration of $CuBr_2$ and KBr was 1/2 or 1/4, while 50 mL of OAR could extract only 40%.

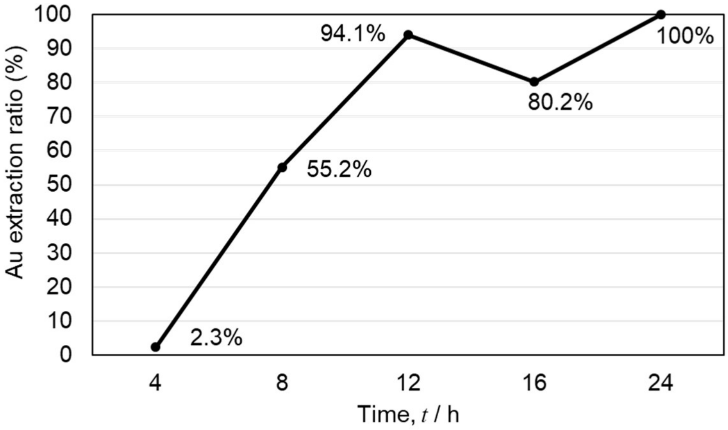


Figure 4: Extraction ratio of Au from the La Maria ore by $CuBr_2$-KBr OAR

$CuCl_2$-NaCl OAR for the Ore from La Maria

Figure 5 shows the extraction ratio of Au by $CuCl_2$-NaCl OAR treatment for the ore from La Maria. (a) shows the effect of the solid-liquid ratio and (b) shows the effect of treatment time. As common conditions, the treatment time was 24 h in the case of (a) and the S:L ratio was 10% in the case of (b), respectively. These experiments were carried out at ambient temperature.
Considering these results, the S:L ratio and treatment time are the important factors for the Au extraction from ore, while the concentration of $CuCl_2$ and NaCl didn't affect significantly.
In addition, the treatment at ambient temperature could extract all of the Au content. This can be a strong feature for the OAR refining procedure because of the easy operation conditions and the low costs.

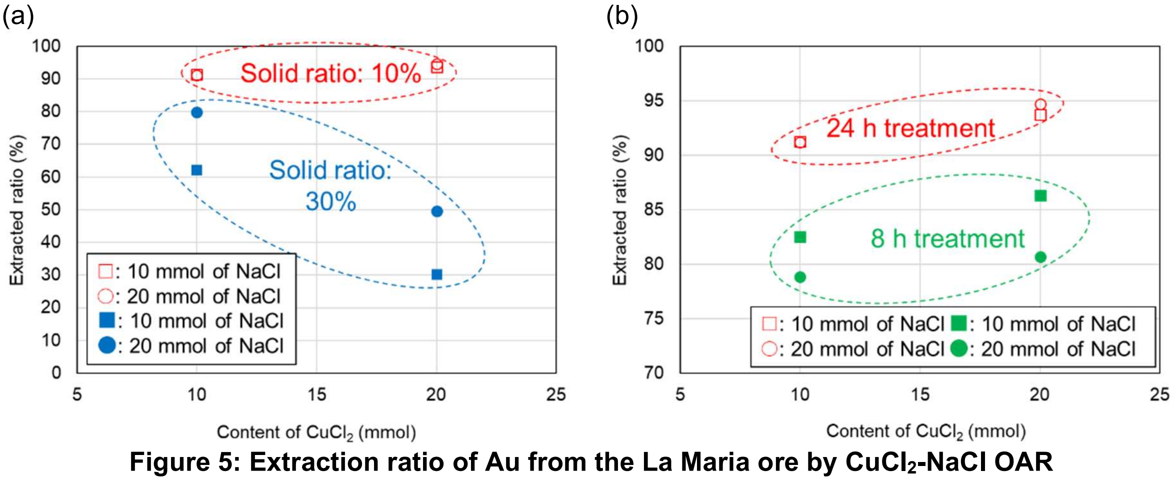


Figure 5: Extraction ratio of Au from the La Maria ore by $CuCl_2$-NaCl OAR

As described above, $CuBr_2$-KBr and $CuCl_2$-NaCl OAR can extract all of Au from ore in the optimized conditions, while cyanidation could extract only 84%. In addition, the contact of OAR with ore is a more important factor for the Au extraction than the concentration of the extractant. Considering this, the pulverizing to obtain a fine ground ore as shown in Figure 1 is an effective pre-treatment for the extraction.
Through these experiments, the authors confirmed the availability of OAR for the Au extraction from ore.

Recovery of Extracted Au

From the simulated ore

Figure 6 shows the recovered material from the simulated ore by $CuBr_2$-KBr OAR treatment and its SEM image. As shown, the recovered material was brown colored and uniform size and shape particles were recovered. The recovered material consists of almost pure Au without other elements. This was because of H_2SO_4 which can evade the precipitation of other elements, especially Cu. In addition, the content in sand e.g. SiO_2 or other oxides were not dissolved by the OAR treatment. As mentioned in the authors' previous research, pretreatments like oxidation or carbonization were effective in the prevention of the dissolution of other elements (4).

Table 1 shows the recovery ratio of Au from each condition. As shown, the existence of sand did not affect the precipitation of Au particles in most cases.

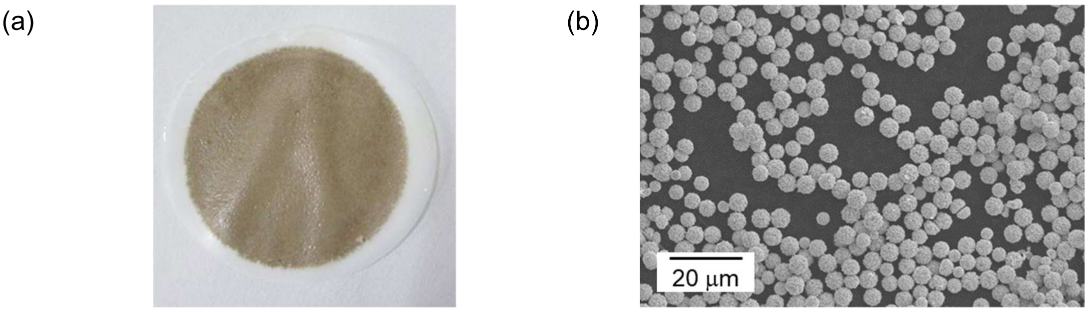


Figure 6: (a) Recovered material from the simulated ore and (b) its SEM image

Table 1: Recovery ratio of Au from the simulated ore

Au content (ppm)	Dissolved ratio (%)	Recovery ratio (%)
Pure Au	100	94.0
5,000	100	79.1
2,500	94.7	87.4
1,000	87.8	52.9
500	72.0	87.8

Figure 7 shows the shape difference of recovered materials between the dissolution conditions. As shown, the shapes of recovered particles differed from spherical or confeito-like by the recovery condition. These shapes are the same as in the previous research in which authors confirmed the particle growth mechanism using pure Au wire as the material (7).

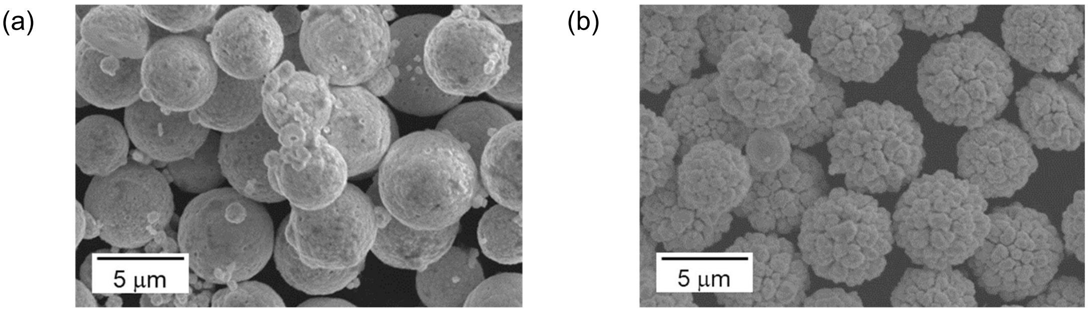


Figure 7: SEM images of recovered materials from (a) 1,000 ppm sample and (b) from 500 ppm sample

The same tendency between these results and the previous research suggests the possibility of Au recovery by the addition of acidic water to Au containing OAR.

From the La Maria ore

Figure 8 shows the recovered material and its SEM image from the La Maria ore by $CuBr_2$-KBr OAR treatment. In this precipitation, the authors added H_2SO_4 solution to prevent the precipitation of some elements e.g. Cu, Fe and S. As a result, this material contained approx. 1.5% of Au and consists of mainly Cu compound. As mentioned above, the ore contains $CuFeS_2$ and this OAR can dissolve $CuFeS_2$ (8). Thus, dissolved $CuFeS_2$ was precipitated by the addition of an acidic solution.

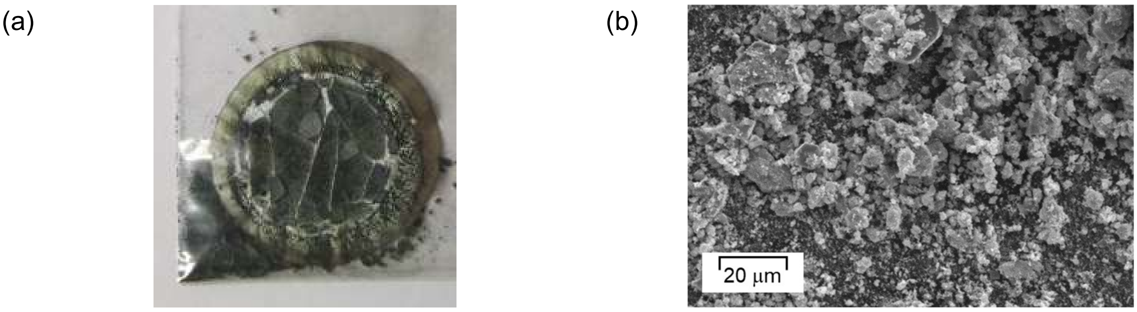


Figure 8: Recovered material from the La Maria ore by H_2SO_4 addition, (a) the precipitated material, and (b) its SEM image

Figure 9 shows the recovered material from the La Maria ore by $CuBr_2$-KBr OAR treatment. Small and dispersed particles were recovered from H_2SO_4 addition, while large and agglomerated particles were from lemon juice addition. In both cases, the Au content in particles was approx. 0.6-1.0% and all of extracted Au was recovered.

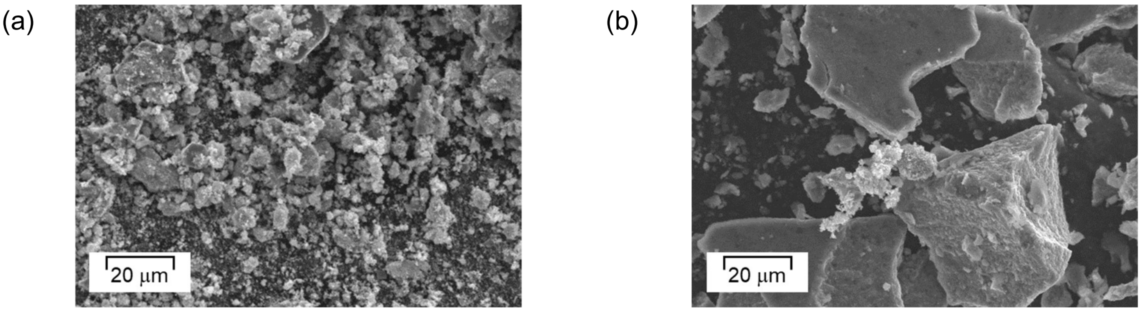


Figure 9: SEM images of recovered material from the La Maria ore, precipitated by (a) H_2SO_4 addition and (b) lemon juice addition

Table 2 summarizes the recovery ratio by the type of added acidic solutions. In the cases of H_2SO_4 or lemon juice, most or all of Au was recovered while vinegar could recover only 25%. As mentioned above, the accessibility of chemicals is an essential factor for spreading ASGM, and lemon juice is appropriate as a precipitant of extracted Au.

Table 2: Recovery ratio change by acidic solutions from the La Maria ore

$CuBr_2$ - KBr (g)	Added acidic solution	Precipitation amount (mg)	Au content (%)	Recovery ratio (%)
1.64 - 0.58	Sulfuric acid (H_2SO_4)	69.9	1.60	100
0.82 - 0.29	Sulfuric acid (H_2SO_4)	61.2	1.41	89.6
3.27 - 1.16	Vinegar (CH_3COOH)	10.9	2.20	25.0
1.64 - 0.58	Lemon juice	261	0.58	100
1.64 - 0.58	Diluted lemon juice	163	0.61	100

For the spreading of the Au refining procedure using OAR

As described above, the authors confirmed the availability of OAR for Au refining. However, the process cost of OAR treatment is much more expensive than the conventional amalgamation process. In this research, the authors used 100 mL of DMSO for approx. 1 mg of Au extraction. This means 100 L of DMSO is required to extract 1 g of Au, while only 1.07 g of mercury was required (9). In addition, the use of acidic water caused the denaturation of OAR and used OAR should be refreshed by evaporation or other treatment. Considering these factors, optimizing the OAR process, including reducing or reusing chemicals, is essential for spreading this process.

CONCLUSIONS

In this research, the authors applied the “organic aqua regia” (OAR) for Au refining from the ore, especially which is refractory for the conventional extraction procedure.
For the application to the simulated ore which consists of Au wire and sand, all or most of added Au wire was successfully dissolved and most of dissolved Au was successfully recovered by the addition of H_2SO_4 solution as pure Au particles.

For the application to the actual ore from La Maria, contained Au was successfully extracted by ambient or low-temperature treatment. In addition, the extracted Au was recovered by the addition of acidic solutions, e.g., H_2SO_4 or lemon juice. Simple operation procedures, easy treatment conditions and easy accessibility of materials enable the operation of the OAR process at ASGM sites.

Through this research, the authors confirmed the availability of OAR for the Au refining. However, the cost of this OAR process is much higher than the conventional ASGM procedure because this procedure requires a large amount of chemicals, especially DMSO. Process optimization, which can reduce the cost, is essential for spreading OAR in developing countries.

ACKNOWLEDGMENTS

Our Group would like to express deep appreciation for the following:
-Gaylord Chemical Company, L.L.C., for providing DMSO for our experiments and sharing their valuable experience with us.
- Newlox Gold Ventures Corp. and their President & C.E.O. Mr. Ryan Jackson for their support and partial funding of our research.
- The Environment Research and Technology Development Fund (JPMEERF20195R52) of the Environmental Restoration and Conservation Agency of Japan for their partial funding of our research.

REFERENCES

1. Hinton, J. J., Veiga, M. M., Veiga, A. C. T., 2003. Clean artisanal mining: a utopian approach? J. Clean Prod., 11, pp. 99-115

2. United Nations Environment Programme, 2020. Minamata convention on mercury.

3. Yoshimura, A., Takai, M., Matsuno, Y., 2014. Novel process for recycling gold from secondary sources: Leaching of gold by dimethyl sulfoxide solutions containing copper bromide and precipitation with water. Hydrometallurgy, 149, pp.177-182.

4. Yoshimura, A., Takatori, K., Matsuno Y., 2020.Environmentally Sound Recovery of Gold from Waste Electrical and Electronic Equipment Using Organic Aqua Regia. Int. J. Automation Technol., 14 (6), pp. 999-1004.

5. Veiga, M. M., Goncalves, A., Aguilar, Y., Li, H., Liu, S., Zhang, D., LA MARIA PROJECT METALLURGICAL TESTWORK

6. Yoshimura, A., Matsuno, Y., 2016. A Novel Process for the Production of Gold Micrometer-Sized Particles from Secondary Sources. Materials Transactions, 57 (3), pp.357-361.

7. Matsuno, Y., Okonogi, E., Yoshimura, A., Sato. M., C. Sato., In-Situ Observation for Formations of Gold Micrometer-Sized Particles in Liquid Phase Using Atmospheric Scanning Electron Microscopy (ASEM). Materials Transactions, 59 (1), pp. 146-149.

8. Takatori, K., Kato, H., Yoshimura, A., Matsuno, Y., 2021. Chalcopyrite Leaching in a Dimethyl Sulfoxide Solution Containing Copper Chloride. Mining, Metallurgy & Exploration, 38, pp.1477-1485.

9. Gonçalves, O. A., 2016. Analysis of gold extraction processes of artisanal and small-scale gold mining In Portovelo-Zaruma, Ecuador, Thesis of University of British Columbia.

INTRODUCING THE THIRD FORCE IN METALLURGY

By

Melinda Moore and Dr Hassan Elsentriecy

Vinca Technologies Limited, Canada

Presenter and Corresponding Author

Melinda Moore
mm@clnresources.com

ABSTRACT

To provide a much-needed hedge against the rising social costs of air, soil and water pollution, the HOLY GRAIL for critical precious and base metal recoveries is to:

a) remove toxic pollutants;
b) lower energy consumption;
c) reduce water use;
d) deliver high metal recoveries;
e) deliver fast kinetics; and
f) return any waste back into the Circular Economy.

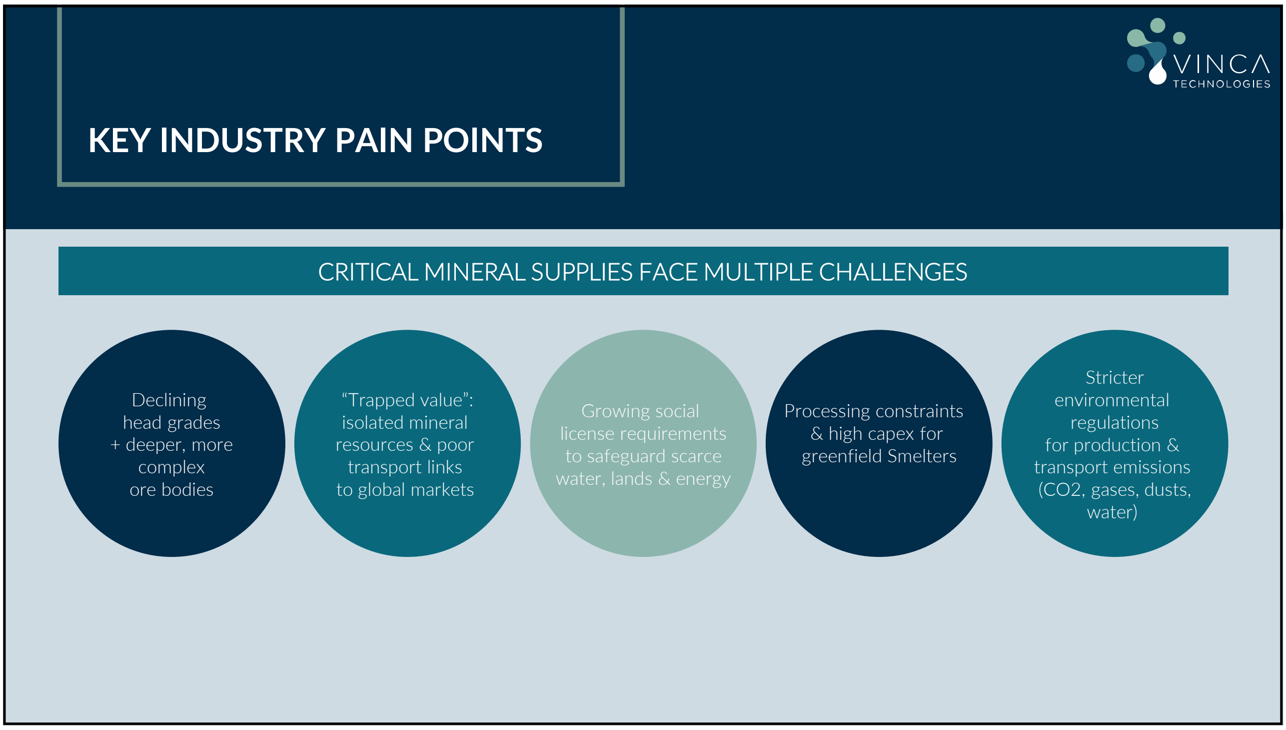
The industry needs to adopt transformational technologies which meet these lofty industry ambitions and address the significant energy and toxicity shortcomings plaguing both the hydrometallurgical and pyrometallurgical metal extraction methods.

The ground-breaking work at Vinca Technologies offers such a solution, by applying chlorides in their molten state to cleverly act as both a lixiviant and as an electrolyte. In so doing, they enable homogeneous processing features capable of capturing multiple critical precious and base metals simultaneously within a single non-toxic, non-aqueous leach and electrodeposition circuit, without the need for ultra-fine grinding; toxic lixiviants or ultra-high temperatures.

Best of all, the Vinca process consistently achieves outstanding recovery results from both oxide and sulphide (*single and double refractory*) hosted polymetallic orebodies, in the range of 90-99+%, and is suitable for recovering Au, Ag, Pt, Pd, Rh, Ru, Co, Cu, Ni, Pb, Zn. Reaction kinetics are rapid, while the spent salts are regenerated, and the gangue re-useable as construction aggregate.

Our paper will present the key technical and economic advantages of the Vinca process, as well as its flowsheet steps in detail, while highlighting why this innovative process is the game-changing chloride solution the industry has long been seeking.

Keywords: Vinca, 3rd Force in Metallurgy, Critical metals, gold, silver, PGMs, cobalt, copper, nickel, Molten chloride salt, Polymetallic orebodies



KEY INDUSTRY PAIN POINTS

CRITICAL MINERAL SUPPLIES FACE MULTIPLE CHALLENGES

- Declining head grades + deeper, more complex ore bodies
- "Trapped value": isolated mineral resources & poor transport links to global markets
- Growing social license requirements to safeguard scarce water, lands & energy
- Processing constraints & high capex for greenfield Smelters
- Stricter environmental regulations for production & transport emissions (CO2, gases, dusts, water)

KEY MACRO GROWTH THEMES

RESONSIBILTY SOURCED CRITICAL MINERAL SUPPORT COUNTLESS RESOURCING OPPORTUNITIES

- Skills Reshoring & Poverty Mitigation
- THE PLANETARY HEALTH Mandate
- Global Industrialisation / Urbanisation Developments

VINCA'S MISSION

Cutting-edge, patented NEXTGEN "green" metal processing technology

DELIVERING CLEANER METALS

- HIGHER RECOVERIES
- LOWER EMISSIONS
- CLEANER OUTCOMES
- SUPERIOR NET ECONOMICS
- IMPROVED LIVELIHOODS

Changing the way metals are sourced for the benefit of all communities and stakeholders

---

ALIGNMENT WITH SOCIETAL GOALS

Sustainable Materials
Responsible Sourcing
Circular Economy
Advanced Materials Innovations

GOAL 1

Clean Water and Sanitation

Ensure availability and sustainable management of water and sanitation for all

GOAL 2

Industry, Innovation and Infrastructure

Build resilient infrastructure, promote inclusive and sustainable industrialisation and foster innovation

GOAL 3

Sustainable Consumption & Production

Achieve sustainable management & efficient use of natural resources

GOAL 4

Climate Action

Take urgent action to avert climate change and negative impacts to local environments

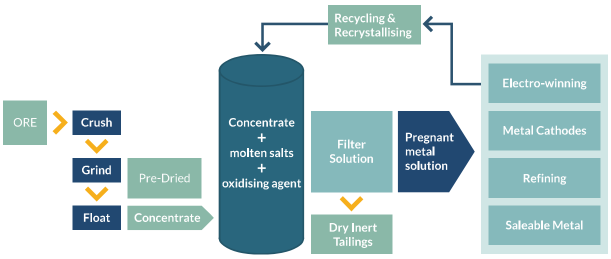
VINCA
Process Advantages

Breakthrough Technology
granted patent recognition

VINCA Advantages

- Non-toxic metal extraction process
- Novel molten salt chemistry
- Simple flow sheet design
- High recoveries & fast kinetics
- Dry, inert, solid resaleable circular economy tailings
- Safe, stable, modular, standardised parameters

SIMPLE FLOW SHEET CONFIGURATION



Multi-metal recoveries:
Au, Ag, Pd, Pt, Rh, Ru, Co, Cu, Ni, Zn, Pb, Sn

NEAR TERM STRATEGY

- Optimise Recoveries
- Expand Global Database
- De-risk Lab scale Flowsheet
- Evolve Comparative Economics

DISCLAIMER

This confidential presentation is provided to the recipient for information only. It is not an offer of securities and is not intended to be relied upon as the basis for an investment decision. The presentation does not and is not to be construed as providing any legal, business, investment or tax advice, and any person wishing to invest in Vinca Technologies Pte. Ltd should obtain appropriate advice from their own advisers.

Any opinion, projection and other forward-looking statement regarding future events or performance of, including but not limited to, countries, markets or companies, is not necessarily indicative of, and may differ from, actual events or results.

Recipients of this presentation agree that Vinca Technologies Pte. Ltd, and its shareholders, employees, officers, agents and representatives shall have no liability for the contents of this presentation, or for any loss howsoever arising from any use of or reliance on its contents. Each recipient agrees that information contained in this presentation is proprietary and confidential in nature and shall: (i) not copy, reproduce, or distribute this presentation, in whole or in part, to any person or party without the prior written consent of Vinca Technologies Pte. Ltd other than to the recipient's professional advisors; and (ii) keep confidential all information in this presentation, except to the extent it is or becomes public.

UPDATE ON CYANIDE FREE LEACHING OF GOLD ORES USING GLYCINE LEACHING TECHNOLOGY

By

Mining and Process Solutions Pty. Ltd.

Presenters and Corresponding Authors

Associate Professor Elsayed Oraby
elsayed.oraby@curtin.edu.au

Dr Glen O'Malley, R & D Manager
gomalley@mpsinnovation.com.au

ABSTRACT

Mining and Process Solutions (MPS) is developing the Glycine Leaching Technology (GLT) for the recovery of base and precious metals. While the focus has been developing GlyCat^{TM} through testwork and piloting programs they are also progressing the less developed GlyLeach^{TM} process. Testwork has been previously presented and published including validation by Outotec.

GlyLeach^{TM} is an alkaline leaching process that can complex gold, much like cyanidation, whose real advantage is its ESG benefit of being a non dangerous good and harmless to humans and the environment. Downstream recovery process is unchanged and can use existing known processes such as conventional activated carbon or zinc cementation (Merrill-Crowe process) to recover the gold and silver from solution. Cyanide-free based glycine leaching includes heated glycine at high pH, glycine in the presence of oxidants (e.g. permanganate, peroxide or ferricyanide) and glycine-thiourea in an acid medium. High gold extraction >90% can be achieved from leaching different gold recourses using different cyanide-free glycine based leaching systems.

This paper updates the progress of the GlyLeach^{TM} process for the recovery of gold where it's our hope to pilot within 2023 to demonstrate a viable process to treat gold containing ores using cyanide-free processes where conventional cyanidation has been banned.

Key words: Glycine Leaching Technology, GlyLeach^{TM}, Gold, Precious metals, Base metals, cyanide-free

Content

- Introduction
- Options of Gold Leaching by Glycine
- Leaching Testwork
- Continuous Piloting and Demonstration
- Q&A

Introduction

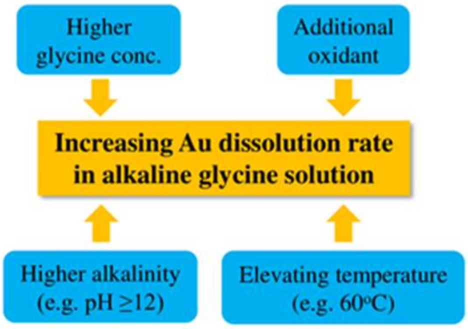
- Cyanidation still remains the best process for the gold extraction because of its efficiency and simplicity.
- However, there are a number of drawbacks associated with the cyanide use
  - Environmental concerns are due to cyanide toxicity
  - High cyanide consumption and low gold recovery in the treatment of complex or refractory gold resources.
- In order to overcome these drawbacks, intensive research has been made to find non-cyanide alternatives to cyanidation.
- These alternatives include thiourea, glycine, thiosulfate, thiocyanate and halides.
- Of all these alternatives, recently glycine has been shown as a promising alternative for gold extraction.

Introduction

- Glycine (NH_2-CH_2-COOH) is a stable amino acid that has various aqueous ionic forms.
- Glycine has a high solubility in water of around 250 g/litre at 25°C, a density of 1.607 kg/ton, molar mass of 75.07 g/mole and a melting point of 233°C.
- Initial research indicated that using only alkaline glycine and air, in the absence of catalyst, leaching times would be too long for conventional agitated tank leaching of ores.
- The leach rate of gold with glycine is significantly slower than with cyanide

Introduction

- Some improvement has been reported with a combination of high temperature and high pH, but the rate is still less than cyanide.
- More development of the glycine leaching technology has been shown comparable extraction to cyanide.



Higher glycine conc.

Additional oxidant

Increasing Au dissolution rate in alkaline glycine solution

Higher alkalinity (e.g. pH ≥12)

Elevating temperature (e.g. 60°C)

Options of Glycine Leaching Technology

GlyCat™
Glycine + NaCN

Leaching of Gold, Silver and PGMs

GlyLeach™
Glycine + Additive

Leaching of **Copper, Nickel, Zinc, Gold, Silver, PGMs and Cobalt**

GlyAmm™
Glycine + Ammonia

Leaching of **Copper, Nickel, Zinc, and Cobalt** Sulphide

GT™
Glycine + Thiourea

Leaching of **Gold, Silver, Copper** and **Rare Earths**

Introduction

Options of cyanide-free glycine based leaching systems

- **Glycine leaching-High Temperature**
- **Glycine leaching-High pH**
- **Glycine-Strong Oxidant**
  - **Glycine-Permanganate**
  - **Glycine-Ferricyanide**
- **Glycine-Thiourea for acidic leaching**

GlyLeach^{TM} Chemistry

❑ **Gold**

$$Au^+ + 2\,NH_3CH_2COO^- \rightarrow Au(NH_2CH_2COO)_2 \quad logK \sim 18.0$$

$$Au + 2\,NH_2CH_2COO^- + 0.25\,O_2 \;+\; 0.5\,H_2O \rightarrow Au(NH_2CH_2COO)_2^- \;+\; OH^-$$

❑ **Copper**

$$Cu + 2\,NH_2CH_2COO^- + 0.5\,O_2 \;+\; H_2O \rightarrow Cu(NH_2CH_2COO)_2 +\; 2\,OH^-$$

GT^{TM} Chemistry

❑ **Gold**

$$Au^+ + 2\,SC(NH)_2 \rightarrow Au(SC(NH)_2)_2^+ \quad logK = 22.1$$

$$Au + 2\,SC(NH)_2 + Fe^{3+} \rightarrow Au(SC(NH)_2)_2^+ + Fe^{2+}$$

$$Au^+ + NH_2CH_2COOH + SC(NH)_2 \rightarrow Au(NH_2CH_2COOH)(SC(NH)_2)_2^+ \;\; logK > 22$$

$$Au + NH_2CH_2COOH + SC(NH)_2 + Fe^{3+} \rightarrow Au(NH_2CH_2COOH)(SC(NH)_2)_2^+ + Fe^{2+}$$

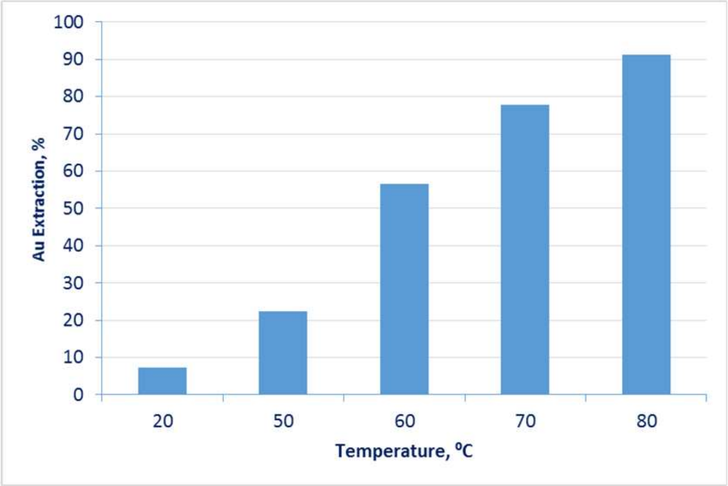
❑ **Copper**

$$2\,Cu + 2\,SC(NH)_2 + 2\,Fe^{3+} \rightarrow Cu_2(SC(NH)_2)_2^{2+} + 2\,Fe^{2+}$$

$$Cu + NH_2CH_2COOH + SC(NH)_2 + Fe^{3+} \rightarrow Cu(NH_2CH_2COOH)(SC(NH)_2)_2^+ + Fe^{2+}$$

GlyLeach^{TM} and Temperature

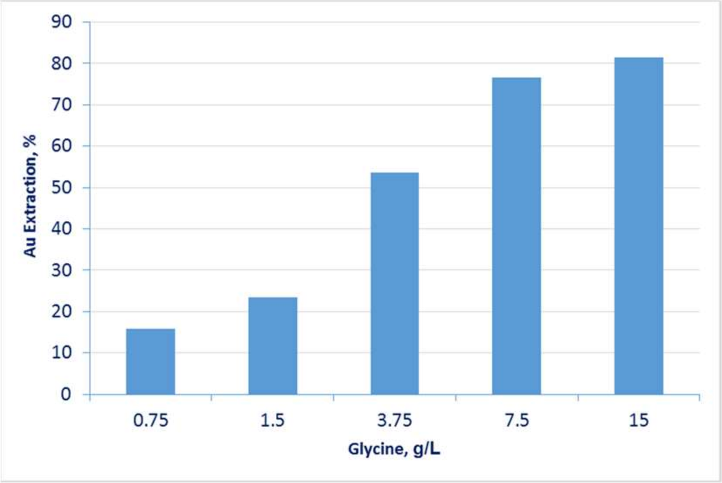
Gold extraction from oxides ore (1.8 g/t)- **Effect of temperature**



0.2 M Glycine, 20 ppm Cu^{2+}, O_2, pH 10.5-11, 48 hrs

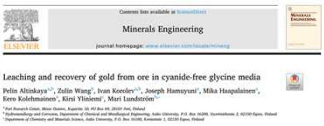
GlyLeach^{TM} and Glycine Concentration

Gold extraction from oxides ore (1.8 g/t)- **Effect of Glycine**



Glycine, 20 ppm Cu^{2+}, O_2, pH 10.5-11, 70 C, 48 hrs

GlyLeach^{TM} Heated Glycine



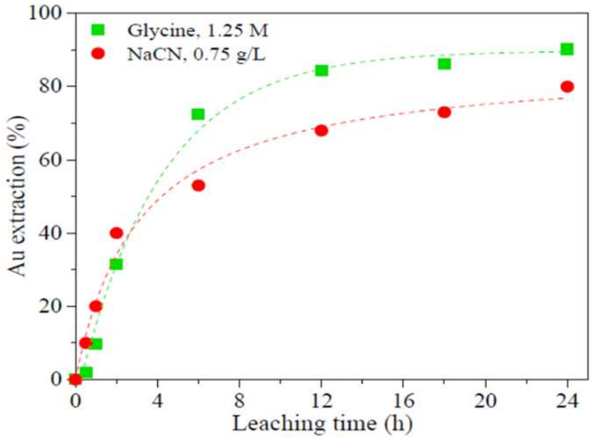
Contents lists available at ScienceDirect

Minerals Engineering

journal homepage: www.elsevier.com/locate/mineng

Leaching and recovery of gold from ore in cyanide-free glycine media

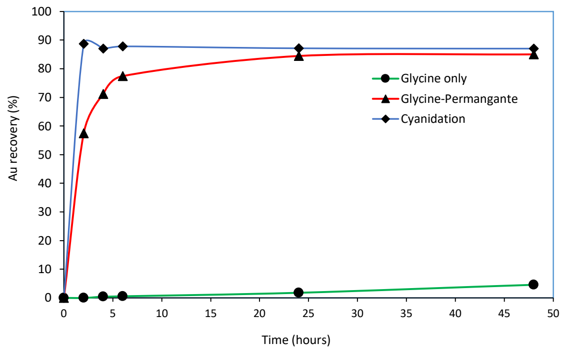
Pelin Altinkaya[a,b], Zulin Wang[b], Ivan Korolev[a,b], Joseph Hamuyuni[b], Mika Haapalainen[a], Eero Kolehmainen[a], Kirsi Yliniemi[c], Mari Lundström[b,c]



Gold extraction from gold oxide ore by glycine at 1.25M glycine, pH 12 and 60C

Glycine-Permanganate

Gold leaching in Glycine-permanganate system

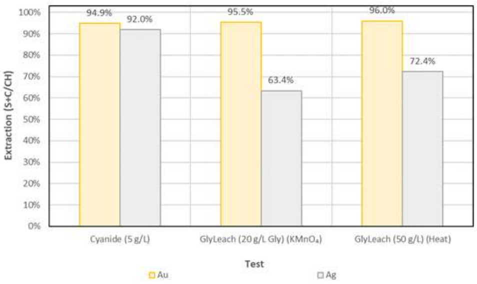


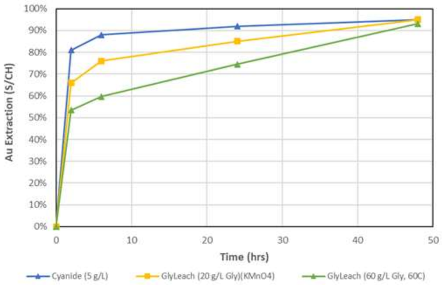
Conditions	Cyanide	Glycine+Reagen
Glycine, g/l	0.0	2.0
Cyanide, ppm	1000	0.0
pH	10.5	10.5
KMnO4, g/l	0.0	2.0

Bottle Roll, Room Temperature, 33% Solids, pH 10.5, 2 g/L Glycine, 2 g/L KMnO4

Ore Leaching Examples

Example 1 - 17.6 g/t Au, 1,245 g/t Ag, 203 g/t Cu

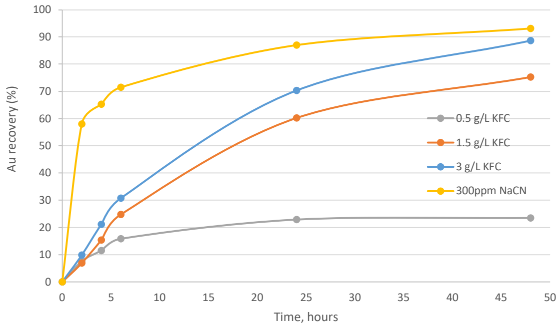




BRTs, 40% Solids, pH 10.5 and room temperature (except last test at 60^{o}C)

Glycine-Ferricyanide

Gold leaching in Glycine-ferricyanide system

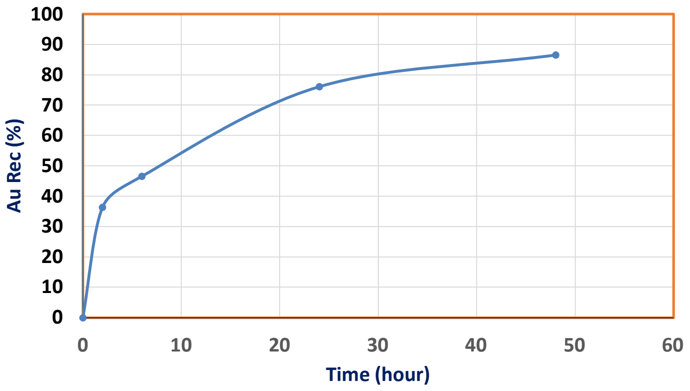


Conditions	Cyanide	Glycine+Reagent
Glycine, g/l	0.0	7.5
Cyanide, ppm	300	0.0
pH	10.5	10.5
KFC, g/l	0.0	0.5, 1.5, 3

Bottle Roll, Room Temperature, 33% Solids

Bottle Roll, Room Temperature, 33% Solids, pH 10.5, RT

GT^{TM} Leaching of BOX concentrate



Bottle Roll,, pH 1.0, 7.8% solids, 20 kg/t GT, Room Temperature.

GT^{TM} Leaching

Advantages

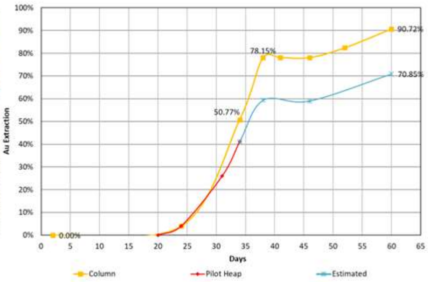
- Low thiourea consumption
- Low thiourea concentration during the leaching
- Leaching at a wide range of pH and Eh
- Glycine complexes ferric and base metals during the leaching
- Activated carbon can be used to recover gold from solutions
- Leaching in an acidic medium for POX, BIOX and Albion process

Disadvantages:

- Glycine should be recycled
- Glycine can be adsorbed on the clay minerals or consumed by bacteria in acidic medium

GT^{TM} - Heap Demonstration (Chile)

- ➢ Draslovka is committed to conducting 10 demonstration with the Glycine Leaching Technology in 2023
- ➢ We have trialed the GT^{TM} in heap demonstration, but it was prematurely stopped at short time than it was required



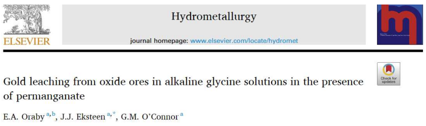
GlyLeach^{TM} and GT^{TM} (Optimum conditions)

- ❑ **Heated glycine (>45ºC) and high pH (>12) for glycine only leaching system**
- ❑ **Alternatively adding an oxidant at ambient temperature such as permanganate, peroxide or ferricyanide**
- ❑ **GT^{TM} required pH<3 and pH>1 and ferric present**
- ❑ **Gold can be recovered onto activated carbon, ion exchange resins or Merrill Crowe Precipitation**
- ❑ **Any copper or any other base metals in solution through precipitation, IX or SX**

Learnings

- ☐ An oxidant or temperature is required to improve leaching kinetics
- ☐ Glycine needs a minimum, but excess only increases copper extraction
- ☐ Metal stability is a problem if not sufficient reagent in solution
- ☐ Once the metals have been recovered, glycine can be easily recycled.
- ☐ Glycine main losses (<5%) occur through adsorption and water loss in the residue
- ☐ High [Temperature, glycine concentration and pH] significantly improves gold leaching.

Publications in cyanide-free glycine based leach



ELSEVIER

Hydrometallurgy

journal homepage: www.elsevier.com/locate/hydromet

Gold leaching from oxide ores in alkaline glycine solutions in the presence of permanganate

E.A. Oraby[a,b], J.J. Eksteen[a,*], G.M. O'Connor[a]

Contents lists available at ScienceDirect

ELSEVIER

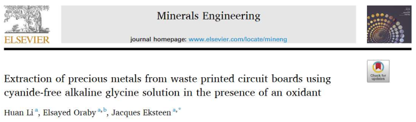
Minerals Engineering

journal homepage: www.elsevier.com/locate/mineng

Leaching and recovery of gold from ore in cyanide-free glycine media

Pelin Altinkaya[a,b], Zulin Wang[b], Ivan Korolev[a,b], Joseph Hamuyuni[a], Mika Haapalainen[a], Eero Kolehmainen[a], Kirsi Yliniemi[c], Mari Lundström[b,*]

[a] Pori Research Center, Metso Outotec, Kuparitie 10, PO Box 69, 28101 Pori, Finland
[b] Hydrometallurgy and Corrosion, Department of Chemical and Metallurgical Engineering, Aalto University, P.O. Box 16200, Vuorimiehentie 2, 02150 Espoo, Finland
[c] Department of Chemistry and Materials Science, Aalto University, P.O. Box 16100, Kemistintie 1, 02150 Espoo, Finland



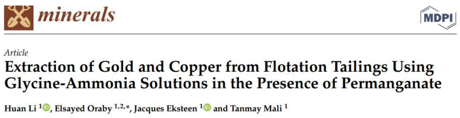
ELSEVIER

Minerals Engineering

journal homepage: www.elsevier.com/locate/mineng

Extraction of precious metals from waste printed circuit boards using cyanide-free alkaline glycine solution in the presence of an oxidant

Huan Li[a], Elsayed Oraby[a,b], Jacques Eksteen[a,*]



minerals

MDPI

Article

Extraction of Gold and Copper from Flotation Tailings Using Glycine-Ammonia Solutions in the Presence of Permanganate

Huan Li 1, Elsayed Oraby 1,2,*, Jacques Eksteen 1 and Tanmay Mali 1

Minerals Engineering 138 (2019) 112–118

Contents lists available at ScienceDirect

ELSEVIER

Minerals Engineering

journal homepage: www.elsevier.com/locate/mineng

Gold extraction from paleochannel ores using an aerated alkaline glycine lixiviant for consideration in heap and in-situ leaching applications

E.A. Oraby[a,c], J.J. Eksteen[a,*], A. Karrech[b], M. Attar[b]

[a] Faculty of Science and Engineering, Western Australian School of Mines: Minerals, Energy and Chemical Engineering, Curtin University, GPO Box U1987, Perth, WA 6845, Australia
[b] Department of Civil, Environmental and Mining Engineering, The University of Western Australia, Australia
[c] Mining and Metallurgical Engineering, Faculty of Engineering, Assiut University, Egypt

Conclusions

❑ **GlyLeach^{TM} can extract gold without cyanide**

❑ **GlyLeach^{TM} process for gold will be more expensive than cyanide due to needing more reagent, oxidant and temperature preliminary economics suggest it will be still viable for gold deposits >3 g/t**

❑ **GT^{TM} has shown to be able to leach gold and some residual copper but further testing to illustrate the economics is needed**

❑ **Both methods show the base and precious metals can be recovered from solution using existing process.**
❑ **Glycine in the presence of permanganate or KFC showed very promising competitive gold recovery compared with cyanide.**
❑ **Gold glycinate has high affinity to the carbon adsorption**
❑ **Glycine-permanganate or KFC is suitable for free milling gold ores at lower reagents consumptions. For sulfidic resources, high oxidant concentrations are required**

UPDATE ON GLYCAT™ LEACHING OF GOLD ORES AND CONCENTRATES

By

Mining and Process Solutions Pty. Ltd.

Presenter and Corresponding Author

Dr Glen O'Malley, R & D Manager
gomalley@mpsinnovation.com.au

ABSTRACT

Over the past 2 years Mining and Process Solutions (MPS) have completed extensive testwork and piloting programs proving their GlyCat^{TM} Process on the flotation concentrate, ores, and tailings. GlyCat^{TM} is the only process that can be easily adapted to existing cyanidation plants in the market today whose real advantages over cyanidation are when treating high cyanide consuming complex material.

With such complex material, GlyCat^{TM} requires 70-90% less cyanide addition to achieve the same precious metal extraction. However, GlyCat^{TM} is not simply cyanidation with added glycine. It is a completely different process where the solution chemistry and conditions needed to effectively leach the precious metals are quite different.

Downstream however, the process is largely unchanged as the recovery of the leached gold and silver uses existing known processes such as conventional activated carbon or zinc cementation (Merrill-Crowe process). Another advantage is lower SCN, OCN, WAD and no free cyanide in the effluent which can substantially reduce cyanide detoxification requirements and decrease the OHS and ESG risk of the operation.

This paper outlines some of these differences and presents testwork, piloting and process examples to highlight the advantages of this new process over conventional cyanidation.

Keywords: Glycine, GlyCat^{TM} leaching, Cyanidation, flotation concentrate, ores, tailings, gold, silver

Outline

- Background
- Chemistry
- Leaching Testwork
- Continuous Piloting and Demonstration
- Questions

What is Glycine?

- Glycine is an amino acid which is non-toxic chemical, classified as non-dangerous good that is fully bio-degradable
- It is used as a food additive for both humans and animals and has sugar-like appearance
- It generally is cheaper than cyanide and readily available from at least 20 different manufacturers
- Is unique in its ability to selectively leach both base and precious metals leaving iron, mercury, or arsenic in residue
- It is not chemically consumed so as a primary reagent it is recoverable and recyclable – providing a major advantage in costs
- Works synergistically with a range of lixiviants

Cyanidation Chemistry

Gold

$Au^+ + 2\,(CN)^- \rightarrow Au(CN)_2^- \quad logK = 39.3$

$Au + 2\,CN^- + 0.25\,O_2 + 0.5\,H_2O \rightarrow Au(CN)_2^- + OH^-$

Copper

$Cu + 2\,CN^- + 0.5\,O_2 + H_2O \rightarrow Cu(CN)_2 + 2\,OH^-$

$Cu(CN)_2 + CN^- + OH^- + 0.25\,O_2 \rightarrow Cu(CN)_2^- + OCN^- + 0.5\,H_2O$

$Cu(CN)_2^- + CN^- \rightarrow Cu(CN)_3^{2-}$

Overall for Copper

$Cu + 4\,CN^- + 0.75\,O_2 + 0.5\,H_2O \rightarrow Cu(CN)_3^{2-} + OCN^- + OH^-$

Overall for Gold and Copper

$Au + Cu + 6\,CN^- + O_2 + H_2O \rightarrow Au(CN)_2^- + Cu(CN)_3^{2-} + OCN^- + 2\,OH^-$

Cyanidation Chemistry

- How operator deal with high cyanide consuming ore that contain copper they adopt technologies like ReCYN or SART i.e., accept what cyanide is needed (high) and recover back what they can to recycle
- SART works by making the copper (I) cyanide (WAD) unstable and precipitate as a copper sulphide by acidifying down to at least pH of 4

$2\,Cu(CN)_3^{2-} + S^{2-} + 3\,H_2SO_4 \rightarrow Cu_2S + 6\,HCN + 3\,SO_4^{2-}$

- How much cyanide gets recovered depends on the cyanide speciation where recoveries between 25-70% are typical
- Unfortunately, any iron cyanide is not recovered with SART as you need to drop the pH to below 1 to make that happen
- Also any cyanate or thiocyanate is not recovered or is reusable cyanide

GlyCat^{TM} Chemistry

Gold

$Au^+ + NH_3CH_2COO^- + CN^- \rightarrow Au(NH_2CH_2COO)(\mathrm{CN})^- \quad logK{\sim}36.0$

$Au + NH_2CH_2COO^- + CN^- + 0.25\,O_2 + 0.5\,H_2O \rightarrow Au(CN)\big(NH_2CH_2COO)^- + OH^-$

Copper

$Cu + 2\,CN^- + NH_2CH_2COO^- + 0.5\,O_2 + H_2O \rightarrow Cu(CN)_2(NH_2CH_2COO)^- + 2\,OH^-$

$Cu(CN)_2(NH_2CH_2COO)^- + NH_2CH_2COO^- \rightarrow Cu(NH_2CH_2COO)_2 + 2\,CN^-$

$Cu(CN)_2(NH_2CH_2COO)^- + OH^- \rightarrow Cu(CN)(NH_2CH_2COO)^- + CN^-$

$Cu(CN)(NH_2CH_2COO)^- + NH_2CH_2COO^- + 0.25\,O_2 + 0.5\,H_2O \rightarrow Cu(NH_2CH_2COO)_2 + CN^- + OH^-$

Overall for Copper

$Cu + 2\,NH_2CH_2COO^- + 0.5\,O_2 + H_2O \rightarrow Cu(NH_2CH_2COO)_2 + 2\,OH^-$

Overall for Gold and Copper

$Au + Cu + 3\,NH_2CH_2COO^- + CN^- + 0.75\,O_2 + 1.5\,H_2O \rightarrow Au(CN)(NH_2CH_2COO)^- + Cu\big(NH_2CH_2COO)_2 + 3\,OH^-$

GlyCat^{TM} Chemistry

- Works by starving the available cyanide which forces the utilisation of WAD and even SAD metal complexes to give up their cyanide for precious metal leaching
- Can achieve 50-95% reduction in cyanide needed to extract the same or greater amount of precious metals compared to cyanidation
- Kinetics tend to be equivalent or faster than cyanidation
- Copper is converted to copper (II) in solution
- Final PLS is very low in cyanate and thiocyanate compared to cyanide
- Much easier to recover the copper or any other base metals in solution through precipitation, IX or SX
- Can use existing processes like activated carbon or Merrill Crowe to recover the precious metals

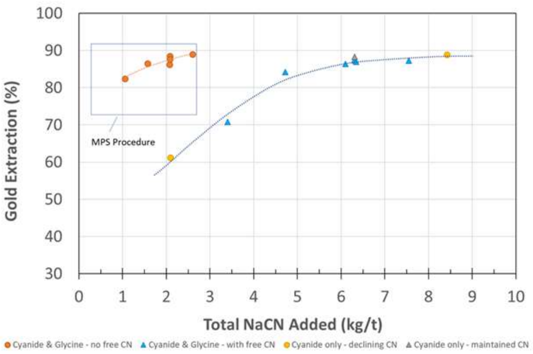
Discovered for Process Control

- Need to add optimal cyanide (too little or too much give poor results)
- Need to add optimal oxygen (too little or too much gives poor results)
- Glycine needs a minimum, but excess only increases copper extraction
- If you recover your solution, you will recycle glycine
- Glycine main losses (<5%) occur through adsorption and water loss in the residue
- Glycine oxidation (<1%) can occur, but we have determined to be minimal under operating conditions

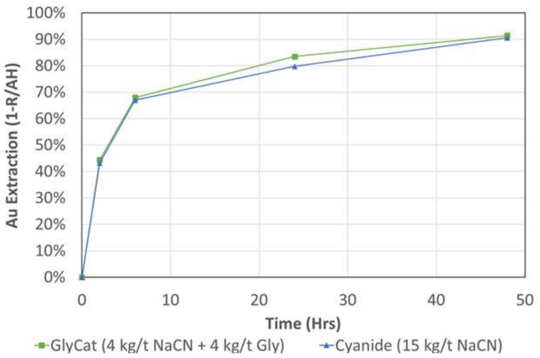
---

Concentrate Leaching Examples

Example 1 19.5 g/t Au, 0.28% Cu



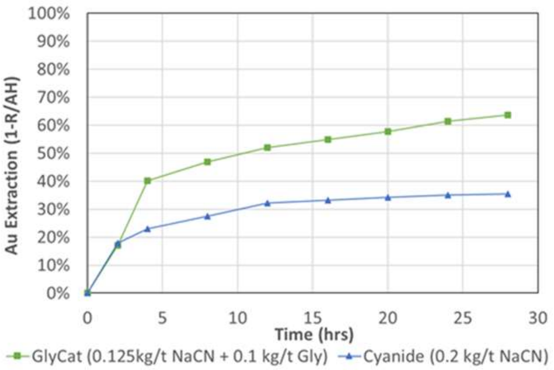
Example 2 35.9 g/t Au, 0.36% Cu



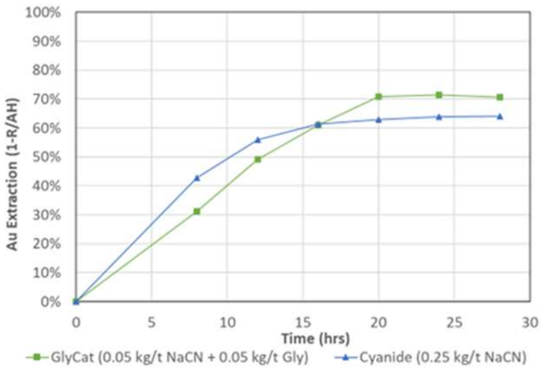
BRTs, 40% Solids, pH 10.5 and room temperature

Flotation Tails Leaching Examples

Example 1 0.24 g/t Au, 0.08% Cu



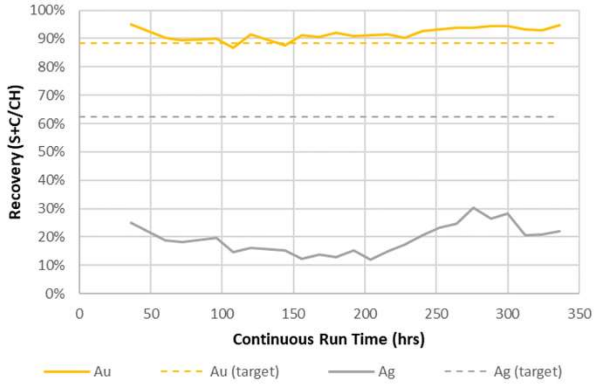
Example 2 0.2 g/t Au, 0.1% Cu



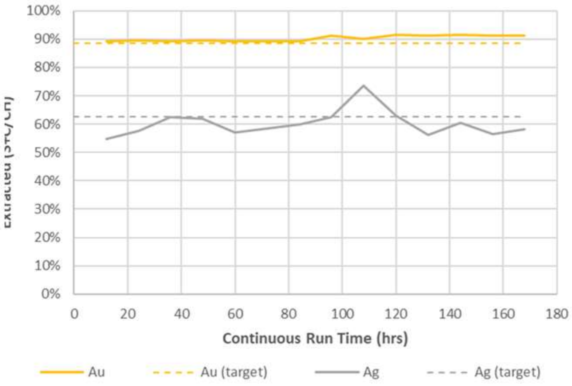
BRTs, 50% Solids, pH 10 and room temperature

Pilot Plant

Pilot Results for Concentrate



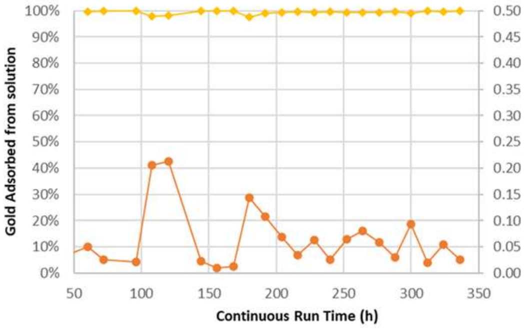
Carbon in Pulp (CIP)



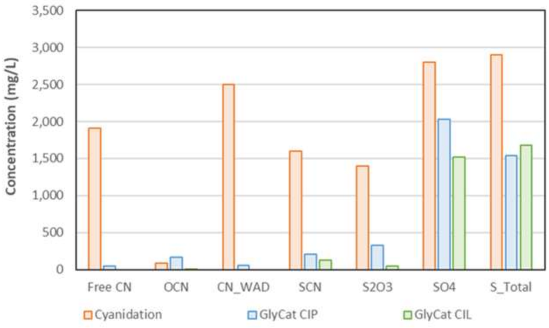
Carbon in Leach (CIL)

40% Solids, pH 10.5, room temperature, 43 hrs leach

Pilot Results for Concentrate



Carbon in Pulp (CIP)



Cyanide and Sulphur Speciation

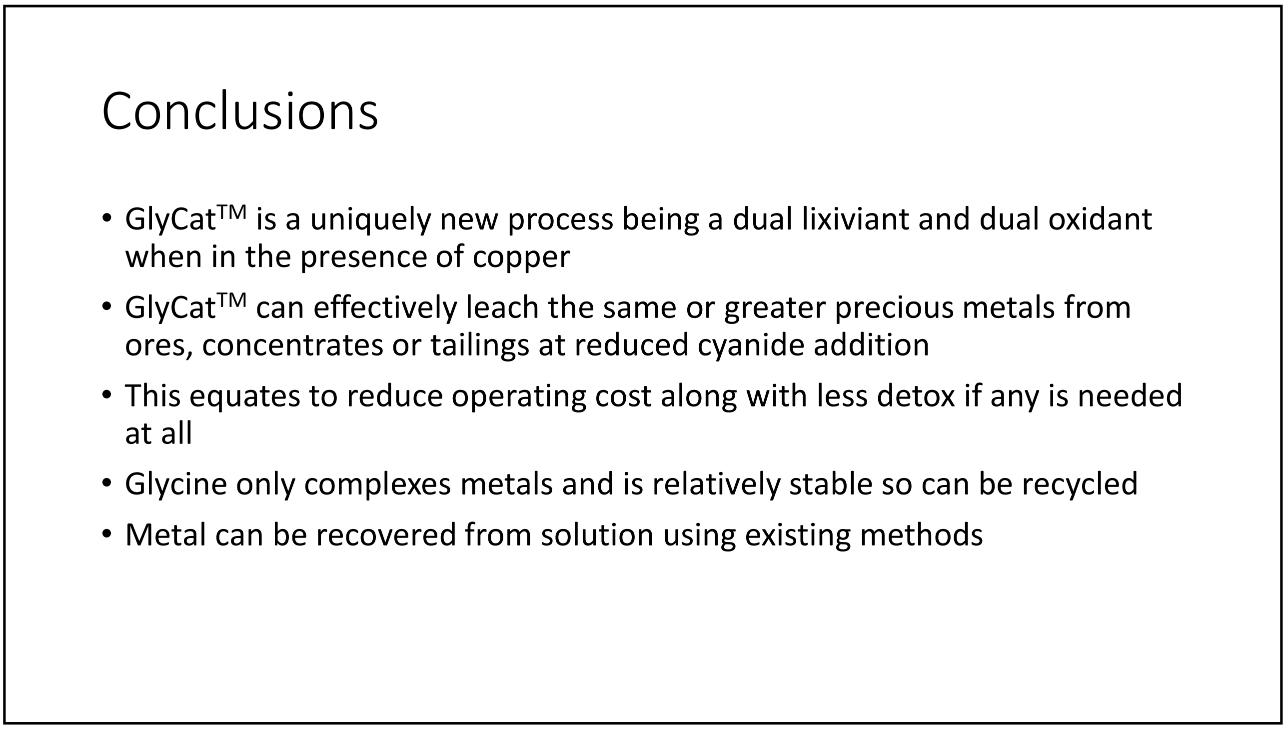
12.5 g/L carbon, 28 hrs adsorption, 1,000 g/t Au (CIP), 2000 g/t Au (CIL)

Demonstration Projects

- Draslovka is committed to conducting 10 demonstration with the Glycine Leaching Technology in 2023
- Currently have commitment for 7 demonstrations

Overall Comparison

Item	Cyanidation	GlyCat^{TM}
% Solids	40-60%	40-60%
pH	8.9-12	8-12.5
Residence time (hrs)	24-48	24-48
Free cyanide maintained	>0.01% or 100 ppm	<10 ppm
DO (mg/L)	>8	4-5
HCN generation	5-20%	<1%
Gangue Dissolution	Cu, Zn, Ni, Co, Fe, Hg, As	Cu, Zn, Ni, Co
WAD Cyanide (mg/L)	20-100	<10
Residual Free Cyanide (mg/L)	20-200	<1



Conclusions

- GlyCat^{TM} is a uniquely new process being a dual lixiviant and dual oxidant when in the presence of copper
- GlyCat^{TM} can effectively leach the same or greater precious metals from ores, concentrates or tailings at reduced cyanide addition
- This equates to reduce operating cost along with less detox if any is needed at all
- Glycine only complexes metals and is relatively stable so can be recycled
- Metal can be recovered from solution using existing methods