

NICO Project Alberta Refinery Process Pilot Plant

Public Final Report
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Prepared for

Alberta Innovates

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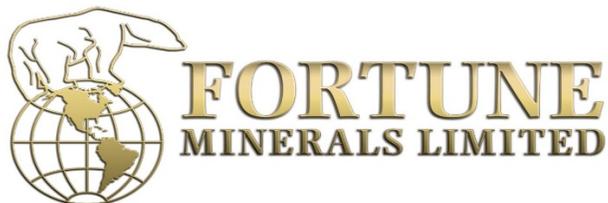


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CLEAN RESOURCES FINAL PUBLIC REPORT



1. PROJECT INFORMATION:

Project Title:	NICO Project Alberta Refinery Process Pilot Plant
Alberta Innovates Project Number:	232402717
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Total Project Cost:	\$1,190,831
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AI Project Advisor:	Amanda Mitchell

2. APPLICANT INFORMATION:

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3. PROJECT PARTNERS

Please provide an acknowledgement statement for project partners, if appropriate.

RESPOND BELOW

SGS Canada Inc. (SGS) in Lakefield, Ontario served as the primary laboratory partner for the NICO Project's testing program. Under the program, SGS completed a wide range of metallurgical test work spanning ore preparation, flotation, hydrometallurgy, impurity removal, and product recovery.

Natural Resources Canada (NRCan) provided funding for this project through their Critical Minerals Research, Development and Demonstration (CMRDD) program.

EXECUTIVE SUMMARY

Provide a high-level description of the project, including the objective, key results, learnings, outcomes and benefits.

Fortune Minerals Limited ("Fortune" or the "Company") is advancing the vertically integrated NICO Cobalt-Gold-Bismuth-Copper Critical Minerals Project, which includes a planned mine and concentrator in the Northwest Territories (NWT) and a dedicated hydrometallurgical refinery in Lamont County, Alberta (Alberta Refinery). Once operational, the project will produce cobalt sulphate, gold doré, bismuth ingots, and copper cement—products that are essential to clean energy technologies, advanced manufacturing, and national security.

This report summarizes the results of a comprehensive seven-phase pilot program undertaken to validate recent flow sheet optimizations, reduce processing risks, and strengthen the future economic and environmental performance of the Alberta Refinery. The primary objective of the program was to generate the hydrometallurgical data required to support an updated Feasibility Study, help attract investment to develop the NICO mine and Alberta Refinery, and advance detailed engineering for the NICO Project's construction and development.

The program involved establishing and operating a concentrate production pilot at SGS Canada Inc. (SGS) in Lakefield, Ontario, initially using approximately 15 tonnes of ore to generate a bulk concentrate emulating the product from the planned concentrator at the NICO site. This concentrate was then reground, subjected to secondary flotation to cobalt and bismuth concentrates, feed characterization, and comprehensive hydrometallurgical testing, and residue stability studies. Targeted improvements included optimized sizing of the bismuth recovery circuit, potential compatibility with third-party feedstock from Rio Tinto's Kennecott smelter, and significant improvements in the hydrometallurgical flowsheet autoclave discharge circuit. Cost-saving measures were demonstrated through the adoption of high-pressure grinding rolls (HPGR), vertical stirred mills, and Jameson cells, which together improved gold and bismuth recoveries, enhanced process efficiency, and mitigated the escalation of both capital and operating costs.

Key results of the pilot program include successful validation of the crushing, grinding and flotation circuits, which achieved high metal recoveries confirming the low mass pull of approximately 4%, and reducing the amount of ore required to be shipped for downstream processing from 4,650 tonnes to 180 metric tonnes. The program also produced high-quality bismuth and cobalt concentrates, confirmed efficient single-stage bismuth leaching and cementation circuits, and demonstrated ferric chloride regeneration at 95% efficiency. Pyrometallurgical testing confirmed the production of 99.99% (4N) bismuth ingots. Pressure oxidation (POX) testing achieved strong cobalt and copper recoveries, while pre-neutralization effectively removed impurities to enable production of market-ready copper cement and a

purified cobalt sulfate solution. This solution was further processed to produce battery graded cobalt sulphate heptahydrate. Gold recovery reached 98.5% through classic carbon-in-leach (CIL) cyanidation, and all process residues were validated to meet environmental standards for safe disposal.

These outcomes confirm the technical robustness of the optimized flow sheet, demonstrate the project's ability to produce high-purity marketable products, and provide the critical data required to advance the project toward detailed engineering and future commercial operation. The program also reinforced important learnings around feed flexibility, residue management, and cost optimization, which will enhance the NICO Project's resilience and long-term performance.

With a previous investment estimate of C\$769 million, including C\$340 million for the Alberta Refinery (updated estimates are in preparation), the NICO Project is expected to create approximately 350 direct and 700 indirect jobs. It will strengthen Alberta's role as a domestic refining hub, contribute to Canada's battery and clean energy supply chains, and foster economic partnerships with northern governments and Indigenous communities.

Aligned with Canadian and U.S. critical minerals strategies, the NICO Project will reduce reliance on supply chains from foreign entities of concern, improve North American supply resilience, and uphold strong environmental, social, and governance (ESG) standards. Fortune is actively pursuing additional public funding through Canadian and U.S. grant programs to accelerate project development and deliver these strategic national and regional benefits.

A. INTRODUCTION

Please provide a narrative introducing the project using the following sub-headings.

Sector introduction: Include a high-level discussion of the sector or area that the project contributes to and provide any relevant background information or context for the project.

Knowledge Gaps: Explain the knowledge gap that is being addressed along with the context and scope of the problem.

RESPOND BELOW

Sector Introduction

This project contributes directly to Canada's critical minerals sector, which supports the production of clean energy technologies, electric vehicles (EVs), national defense systems, and advanced manufacturing. Critical minerals such as cobalt, bismuth, and copper are essential inputs for lithium-ion batteries, renewable power infrastructure, environmentally safe materials, and high-tech applications. However,

current global supply chains for critical minerals are concentrated in few jurisdictions, leaving Canada vulnerable to supply disruptions, price volatility, and geopolitical risks.

To address this challenge, Canada has prioritized the development of secure, domestic critical mineral supply chains as part of its national strategy. The NICO Project directly supports this objective by establishing a vertically integrated operation that will produce cobalt, bismuth, and copper, along with gold and gypsum coproducts. Once operational, the project will help reduce Canada's reliance on foreign sources of critical minerals, enhance the resilience of Canadian and allied supply chains, and position Canada as a global leader in sustainable, responsible mineral development.

Knowledge or Technology Gaps

The technology gaps addressed as part of this project include improved concentrate metal concentrations and recoveries, and improvements to the metallurgical refining processes for the proposed Alberta refinery to mitigate capital and operating cost escalation.

Since acquiring the NICO Deposit, Fortune has conducted extensive bench- and pilot-scale testing to develop a process flowsheet tailored to its unique mineralogy. The resulting integrated flowsheet combines flotation concentration with hydrometallurgical refining to recover cobalt, bismuth, copper, and gold while minimizing hazardous waste. Recent test work at SGS focused on validating further improvements to refining processes, reducing waste volumes, and enhancing by-product recovery to support updated feasibility studies and detailed engineering.

Addressing these technology gaps is critical in the context of North America's limited capacity to process complex critical mineral ores. Conventional smelters and refineries in North America are not equipped to process complex ore types like those of the NICO Deposit, which has historically led to these materials being exported overseas to China for processing, with associated losses of economic value-add jobs, and control over strategic resources. By developing a Canadian-based processing solution, this project advances domestic capacity for critical mineral refining. The proposed Alberta Refinery is positioned to serve not only the NICO Project but also as a strategic hub for processing other domestic ores, smelter residues, and recycled materials, thereby strengthening North America's critical mineral supply chains in support of clean energy, advanced manufacturing, and national defense priorities.

B. PROJECT DESCRIPTION

Please provide a narrative describing the project using the following sub-headings.

- **Knowledge or Technology Description:** Include a discussion of the project objectives.
- **Updates to Project Objectives:** Describe any changes that have occurred compared to the original objectives of the project.

Technology Description

To address the identified technology gaps, this project was structured around three project Milestones aimed at validating and gathering key data for the proposed concentration and extraction processes on materials from the NICO mine. The data and test results generated from these Milestones were intended to directly inform engineering design and feasibility studies for the proposed concentrator in the NWT and hydrometallurgical facility in Alberta.

Milestone 1 proposed setting up and operating a concentrate production pilot at SGS. The project includes establishing pilot plant benchmarks and testing of at least 10 tonnes of ore mined and concentrate from the NICO NWT site.

To address the technology gaps presented, two project Milestones (2 and 3) were established to validate and quantify the process design criteria for the proposed concentrator and Alberta Refinery for ores produced from the NICO mine. Evaluation of materials and processes included characterization, testing and lab analysis of feed concentrates, intermediary concentrates and end products against end product quality parameters. Testing proposed also included separation of ferric arsenate from the gypsum residue and testing the stability of ferric arsenate for disposal. As part of the key indicators of success from these objectives, bismuth ingot purity of 99.995% and cobalt sulphate recovery of 20.5% were identified as desired hydrometallurgical recovery targets to meet the specific needs of the battery industry.

The data and test results generated were intended to influence engineering and feasibility studies for the vertically integrated project and supported the objectives of advancing technology readiness level (TRL) of Fortune's processes from TRL 5 to 6.

Final deliverables for the project included engineering, SGS project management and hydrometallurgical reports. Successful completion of the pilot and testing will be shared with the mineral sector and attract investment to develop the NICO mine and Alberta refinery, which were also key measures of success identified for the project.

In addition to the project objectives, Fortune committed to engage an Alberta post-secondary institution to identify minerals and/or other feedstocks that could benefit from this integrated processing approach. Fortune also committed to provide a commercial pathway document for the NICO mine and refinery advancement, with a specific focus on the acquisition of raw materials from another jurisdiction and reduce liability of wastes from the refinery process, notably gypsum, where possible.

Updates to Project Objectives

The project objectives established under the original scope remained consistent, with testing, validation, and engineering activities executed as planned. Timelines for the project activities were extended to allow for additional testing of ore materials. Some additional testing and inclusion of Rio Tinto feedstocks, to meet project approval conditions were also included in the scope of work for the project.

C. ACTIVITIES

Please provide a narrative describing the activities that were used to execute and complete the project. Use subheadings as appropriate.

The following scope of work was undertaken from January 2024 to the end of November 2025 for the Project and aligns with the revised Milestone and Scope of work with the project.

Milestone 1 included a review of previous test work to optimize methods to concentrate ores prior to shipping, assess the risks associated with the new processes, and identify potential improvements based on test work results. The concentrate production pilot provided a platform to verify trialed improvements to Fortune's crushing, grinding and flotation processes. These include lower costs and more efficient crushing and grinding equipment, optimizing grind size and reagent mix, and validation of the secondary flotation process to generate distinct cobalt and bismuth concentrates. Specific activities associated with Milestone 1 work are detailed below in Phases.

Phase 1: Review of Previous Test Work & Test Work Design

- Review of previous test work
- Complete risk identification and mitigation
- Provide technical memo for test work design draft and final documents (consultants)

The program began with a detailed review of previous bench- and pilot-scale test work, led by Fortune's Chief Metallurgist, Alex Mezei, in collaboration with engineering partner Worley. Key activities in this phase included compiling and analyzing historical test data, identifying areas requiring further validation

to guide subsequent test planning and conditions, and conducting joint engineering sessions to review and refine process design criteria and equipment selections. The review also included an assessment of process risks and an evaluation of opportunities to enhance the existing flowsheet.

Phase 2: Feed Source Retrieval & Transportation

- Feed Source preparation and retrieval at NICO site
- Transport and delivery to Yellowknife of 15 tonnes of ore and trucking to SGS Lakefield in Ontario

Approximately 15 tonnes of ore were initially collected from existing stockpiles at the NICO site using Fortune's onsite equipment. The material was placed into super sacs, flown by helicopter to a nearby road access point, and then transported by truck to SGS. An additional 1.5 metric tonnes were subsequently collected to support other test work gaps as part of Milestones 2 and 3.

Phase 3: Concentrate Production – 2024 lab and mini-pilot

- Receipt and crushing of ore
- Feed characterization
- Verification of expected metallurgy
- Set benchmark for concentrate pilot plant
- Operate pilot plant to separate cobalt and bismuth concentrates
- Project management and flotation component interim reports and any interim memos, as well as final reports of relevance

Upon arrival at SGS, the NICO ore was inspected, crushed, and sampled to verify feed quality and suitability for the planned flotation program. Feed characterization confirmed that gold, bismuth, and copper grades met the Company's target specifications, while cobalt grades were slightly below the deposit average.

The crushed ore was ground to a particle size of approximately 54 microns and processed through bulk flotation to produce a primary concentrate, which was subsequently reground to 14 microns for secondary flotation to separate distinct cobalt and bismuth concentrates. This work successfully demonstrated the ability to achieve clean separation between the two metals, establishing the metallurgical benchmark for subsequent hydrometallurgical test work.

Additional samples were subsequently collected to emulate the average production grade criteria and it was proven that the metallurgical response was insensitive to the cobalt grade of the concentrate samples. This was done by producing an additional amount of cobalt concentrate from an additional ore sample. Both the "On-spec" and "Off spec" feed samples were blended separately with Bismuth leach

residue at the required ratio and subjected to pressure oxidation and subsequent downstream unit operations . The results were materially similar.

Milestone 2 of the project included hydrometallurgical testing for the bismuth circuit, including ferric chloride leaching and cementation to finalize the process design criteria and verify product quality, followed by pyrometallurgical processing to make high quality (4N) bismuth ingots. POX tests were carried out on the cobalt concentrate blended with bismuth leach residue to confirm gold recovery rates. It also attempted to produce a gypsum by- product from the process residue and prove the process residue can still be disposed in a government approved Class 2 landfill. Specific activities associated with Milestone 2 work are detailed below in Phases.

Phase 4: Value enhancement studies & procurement of additional samples for HPGR + Jameson cell optimization

- Procured additional ore samples from mine site in NWT (2024).
- Conducted HPGR Labwall comminution testing and Jameson float cell tests to recover additional fine gold and bismuth.
- Used the reject fraction from the Jameson cell tests to make additional cobalt and bismuth concentrates for subsequent hydrometallurgical test work.
- Subjected the HPGR samples to Metso for Jar and Bond Work Index (BWI) tests.
- Procured more additional samples (2025) to produce bulk concentrate for physical testing to generate data required by Worley: liquid-solid separation / dewatering (Metso) and flow properties (Jenike and Johanson).

Phase 4 focused on value enhancement studies and optimization of the NICO Project concentrator flowsheet to identify opportunities for improving metallurgical performance while reducing capital and operating costs. Worley completed detailed assessments of the existing mill and concentrator designs and identified several potential optimizations, including replacement of the cone crusher and ball mill with High Pressure Grinding Rolls (HPGR) and vertical stirred mills to improve energy efficiency and reduce equipment footprint. The studies also recommended incorporating Jameson cells at the back end of the bulk cleaner flotation circuit to enhance recovery of fine gold and bismuth. In support of these evaluations, Chief Metallurgist Alex Mezei prepared a comprehensive process opinion summarizing all metallurgical test work completed to date to guide Worley's engineering review.

To support the Jameson cell trade-off testing, additional ore samples were collected from existing stockpiles at the NICO mine site during coordinated site visits to the NWT and Alberta with Worley engineers. The objective of these tests was to assess the potential to recover additional fine bismuth and gold from the bulk flotation cleaner circuit. Results demonstrated a 3–5% improvement in bismuth and gold recovery relative to previous flotation tests, though accompanied by a higher sulphide mass pull that

would require further evaluation in a cost-benefit analysis. The reject fraction from these tests was subsequently used to produce cobalt concentrate meeting the target grade criteria for downstream hydrometallurgical testing. This material was used to produce the “On-spec” cobalt concentrate mentioned in the previous sub-section.

In addition, core samples were collected to support HPGR, Jar, and BWI variability testing required for equipment sizing and validation of the updated comminution design. Initial LABWAL HPGR tests performed by SGS, along with complementary JAR and BWI tests conducted by Metso, confirmed the expected ore hardness and absence of significant variability, further validating the proposed comminution flowsheet and concentrator design.

In tandem with the collection of the BWI variability testing, an additional batch of samples were collected and used to produce a bulk concentrate needed for above mentioned engineering data. These tests are in progress at the time of writing this report. They include liquid -solid separation (thickening and pressure filtration tests) by Metso, followed by flow-property testing by Jenike and Johanson.

Phase 5A: Hydrometallurgy

- Performed ferric chloride leaching tests on bismuth concentrate, followed by cementation tests onto iron powder and determine the Process Design Criteria (PDC) for any optimization.
- Liquid-solid separation engineering data were generated through each unit operation above.
- Conducted test work on Rio Tinto bismuth oxychloride samples using the Fortune PDC and perform the same tests on material blended with Fortune process streams.
- Validated gold recovery tests on various concentrate products.
- Validated POX tests on the cobalt concentrate / bismuth leach residue blend.
- Prepared bismuth hydrometallurgical circuit PDC and submitted these to Worley.

The bismuth concentrate produced during Phase 3 was subjected to single-stage ferric chloride leaching tests at SGS, followed by cementation onto iron powder to produce a high-quality bismuth cement. These tests were completed to refine process parameters, operating conditions, and reagent optimization for the bismuth circuit. Additional tests were carried out to evaluate lixiviant regeneration using hydrogen peroxide to assess recycling potential, followed by supplementary tests using chlorine as a lower-cost alternative. The later option was included in the PDC. Bismuth cement purification tests using sulfuric acid were also conducted to remove residual iron but were determined to be unnecessary, as the acid treatment produced a lower-quality feed for smelting. Detailed geochemical and mineralogical investigations were completed at XPS to verify these findings.

Rio Tinto provided a bismuth oxychloride intermediate from its Utah Kennecott smelter for compatibility testing with Fortune's hydrometallurgical process. Initial bulk leaching tests conducted on unblended Rio Tinto material resulted in high acid consumption. A subsequent blended test, using a 1:10 ratio of Rio Tinto material to Fortune's bismuth concentrate, demonstrated successful leaching and cementation, confirming potential process compatibility.

Additional tests were conducted at SGS to recover gold from the bismuth leach residue prior to blending it with cobalt concentrate for subsequent POX testing. These tests demonstrated an alternative process option for gold recovery from the bismuth circuit, as a redundancy in the event of a temporary autoclave shut down situation. Preliminary POX sighting tests were then carried out at SGS on the blended cobalt concentrate and bismuth leach residue, confirming suitable autoclave operating conditions and metal recoveries. The autoclave discharge material was subsequently used to support iron-arsenic separation and to conduct PLS neutralization tests.

Phase 6A: Initiated Bismuth Smelting & Refining Tests

- Delivered bismuth cement samples to XPS Industry Relevant Solutions (XPS) for mineralogical and chemical characterization.
- Executed detailed chemical and mineralogical analyses, prepared a detailed test work program.
- Commenced initial smelting tests to make crude bismuth and refining tests to make a 99.99% pure bismuth ingots.

The bismuth cement produced during Phase 5A was shipped to XPS in Sudbury, Ontario for pyrometallurgical testing, following unsuccessful initial smelting trials at SGS due to equipment limitations and a lack of specialized expertise in bismuth smelting. At XPS, the bismuth cement underwent detailed mineralogical and chemical characterization to establish a baseline for subsequent test work. Smelting tests were then conducted to produce a crude bismuth ingot suitable for further refining. Based on the results, process design criteria were developed for refining the crude product to achieve high purity 99.99%+ (4N) bismuth ingots, which were validated in the next phase of testing.

Milestone 3 optimized the downstream process to advance the NICO Project's refining and hydrometallurgical process validation for the proposed Alberta Refinery. This milestone encompassed the successful smelting and refining of bismuth into high-purity ingots, as well as comprehensive pressure oxidation (POX) test work on blended cobalt concentrate and bismuth leach residue. These programs confirmed efficient iron and arsenic removal and enabled optimization of autoclave pregnant leach solution (PLS) neutralization, followed by manganese removal, solvent extraction (S-X) purification, and evaporation and crystallization of high-purity cobalt sulphate heptahydrate suitable for battery applications. A sensitivity pre-assessment on the production of cobalt hydroxide was also undertaken to evaluate potential flexibility in product outputs under varying market conditions and upfront capital and

operating costs and commissioning risks. It also focused on validation of the copper cementation process design criteria. Toxicity Characteristic Leaching Procedure (TCLP) tests were also conducted to verify the environmental stability of the final process residues, demonstrating safe disposal options following gold recovery. Specific activities associated with Milestone 3 work are detailed below in Phases.

Phase 5B: Cobalt Hydrometallurgy

- Conducted POX tests of combined cobalt concentrate and bismuth leach residue to dissolve cobalt and copper.
- Conducted optimization tests on copper removal from the PLS and cementation.
- Conducted iron/arsenic removal optimization tests.
- Precipitated a gypsum process by-product – while technically valid, the results were not deemed successful in terms of potential commercial outcome.
- Successfully tested a significantly simplified flowsheet based on Pre-Neutralization of the PLS resulting in a significant implied reduction of the capex and opex needed to eliminate most of the iron and arsenic.
- Conducted further purification tests to remove the traces of arsenic and iron from the PLS, which became feed for the Manganese removal.
- Liquid-solid separation engineering data were generated for selected unit operations.
- The purified cobalt-bearing solution was subsequently treated subjected to manganese removal by precipitation using a sulphur dioxide/oxygen gas mixture.
- The manganese-free solution was subjected to solvent extraction (SX) to produce a cobalt sulphate solution while rejecting trace metals including magnesium, nickel, zinc, and copper.
- The resulting purified strip solution was then subjected to evaporation and crystallization to produce a high-purity cobalt sulphate heptahydrate product suitable for battery-grade applications. Prepared PDC and submitted it to Worley for engineering.

Phase 5B focused on completing the remaining hydrometallurgical test work to validate and optimize the recovery, purification, and refining steps within the cobalt circuit. POX tests were conducted on blended cobalt concentrate and bismuth leach residue to confirm the autoclave operating conditions, metal recoveries, and impurity behavior. Test materials included cobalt concentrate from the 2024 flotation pilot and additional concentrate generated from Jameson cell test rejects. The resulting POX liquor was used to evaluate its suitability for gypsum production. While technically valid, the results were not deemed successful in terms of potential commercial outcome due to arsenic concentrations that exceeded the product specification.

Instead of producing gypsum, Fortune successfully tested a significantly simplified flowsheet based on non-oxidative pre-neutralization without using oxygen and heat, and for 30 minutes of residence time

instead of 4 hours previously. The results allowed for significant reductions for the forecast capital and operating costs needed to eliminate most of the iron and arsenic from the PLS.

The pre-neutralization tests were conducted comparatively using limestone and lime as neutralization reagents to provide process design and cost flexibility. The residues produced were collected for gold recovery testing, while the corresponding solutions were used in copper cementation trials.

Copper cementation was completed on the pre-neutralized PLS using both coarse and fine iron powders to ensure residual copper concentrations could be reduced below analytical detection limits (<0.3 mg/L), confirming suitability for subsequent cobalt processing. Secondary oxidative neutralization tests were then conducted to maximize the removal of remaining iron and arsenic using air and oxygen as oxidants.

The purified cobalt-bearing solution was subsequently treated through manganese precipitation using a sulphur dioxide/oxygen gas mixture, followed by solvent extraction (SX) purification to remove trace metals including nickel, zinc, and copper. The purified solution was then subjected to evaporation and crystallization to produce a high-purity cobalt sulphate heptahydrate product suitable for battery-grade applications.

All testing data, reagent optimization results, and process parameters generated during Phase 5B were compiled to prepare the final PDC for engineering use in the Alberta Refinery design.

Phase 6B: Completed Bismuth Smelting & HPGR tests

- Completed Bismuth Smelting and Refining test work at XPS Industry Relevant Solutions and validate production of 99.99% bismuth ingots.
- Completed PDC for Bismuth pyrometallurgical circuits.
- Completed remaining tests for HPGR process design criteria at Metso including Jar and Bond Work Index tests.

Following the preliminary smelting work completed in Phase 6A, additional pyrometallurgical testing was conducted at XPS to refine and finalize the bismuth smelting and refining process. The objective of this phase was to validate the production of high-purity bismuth ingots and complete the Process Design Criteria for the bismuth pyrometallurgical circuits. The initial crude ingot produced during Phase 6A, assaying approximately 99.8% bismuth (“2N”), was subjected to successive refining stages including liquation, sulphuration, and chlorination to further increase product purity. These refining steps successfully achieved 99.99% (“4N”) bismuth, meeting the Company’s specification for high-purity bismuth metal suitable for specialty alloy and chemical applications. The final report from XPS provided the Process Design Criteria required for detailed engineering of the Alberta Refinery’s bismuth circuit.

In parallel, remaining HPGR comminution tests were completed to finalize the PDC for the concentrator. Labwall HPGR testing was conducted at SGS, and representative samples were sent to Metso for Jar and BWI testing to characterize ore hardness and variability. These tests were performed on four ore types, including material from NICO test-mining samples and core collected during the November site visit.

Phase 7: Validated Gold Recovery & Process Residue Stability

- Complete gold recovery tests for the combined autoclave leach residue and the cleaner flotation tails were conducted
- Cyanide destruction using the sulphur dioxide/oxygen process (a.k.a. Inco)
- Process residue stability for disposal in a Class 2 Government Approved landfill.

Phase 7 focused on validated gold recovery from the combined autoclave leach residues, as well as from the bulk scavenger and cleaner flotation tails. It also confirmed the cyanide destruction conditions, and re-assessed process residue stability for disposal in a Class 2 government-approved landfill.

Carbon-in-leach (CIL) bottle roll tests were conducted on the autoclave leach residues generated from POX of cobalt concentrate blended with bismuth leach residue. Test work included residues neutralized using both lime and limestone under revised process conditions to optimize gold extraction within the updated hydrometallurgical flowsheet.

In addition to the main gold recovery circuit testing, cyanidation test work was carried out on the bismuth ferric leach residue to assess alternative gold recovery options that could be implemented during an autoclave shutdown or upset condition. Complementary CIL testing was also performed on bulk scavenger and cleaner flotation tails to evaluate the potential to recover gold from these process streams and to understand its impact on overall circuit performance. Cyanide destruction tests were undertaken on selected leach solutions and tailings to develop suitable detoxification conditions ahead of discharge or disposal.

To evaluate process residue stability, Toxicity Characteristic Leach Procedure (TCLP) tests were completed on gold plant discharge residues produced under both lime and limestone neutralization conditions, as well as on the POX residues, to assess the leachability of metals such as iron and arsenic. Acid-Base Accounting (ABA) tests were also performed on the autoclave and cleaner scavenger tailing residues, both individually and as blended composites, to characterize their long-term geochemical behaviour and support confirmation of their suitability for disposal in a government-approved Class 2 landfill.

Phase 8: Reporting

Phase 8 focused on completion of all final documentation and reporting deliverables associated with the pilot plant test work program. Metallurgical consultants prepared comprehensive final reports summarizing the results from each phase of testing and presented the findings to Fortune.

Activities Associated with Project Approval Conditions

Fortune has engaged with post-secondary institutions and experts to initiate potential studies. The company has also discussed data sharing and potential thesis sponsorship with researchers Pilar Lecumberri-Sanchez and her Ph.D student, Juan Rodriguez will be supported with data from the NICO Project and will have access to the NWT in the 2026 field season. The Company also engaged with Wei Victor Liu, Qui Liu and Alex Umnikov at the University of Alberta, and Sathish Ponnurangam at the University of Calgary.

Fortune included materials from Rio Tinto in its test work programs to explore the feasibility of recovering additional bismuth and cobalt contained in waste residues from the Kennecott Smelter facility in Utah. Fortune also undertook discussions with an Alberta company with access to metaliferous shales from The Birch Mountain area of Alberta, and has had discussions with companies with projects in the NWT with lithium-bearing spodumene pegmatites, rare earth element-bearing intrusions and tungsten concentrates regarding providing feedstocks for processing at the proposed Alberta Refinery. Fortune also continues to explore opportunities for recovering metal from recycling scrap, chemical residues, batteries and electronic wastes as a potential future feedstock.

Fortune has advanced the NICO Project to a near shovel-ready stage, featuring proven and probable mineral reserves of 33.1 million tonnes and validated metallurgical processes to recover the contained metals. An updated feasibility study is currently underway regarding mine options in the NWT with proposed hydrometallurgical refining in Lamont County, Alberta.

D. PROJECT RESULTS

Please provide a narrative describing the key results using the

- Describe the importance of the key results.

PHASE 1 RESULTS

The technical review of previous test work delivered an optimized, risk-mitigated flowsheet and a clear basis of design for the pilot-scale test program. This review informed improvements to existing, well-established metallurgical processes that have been successfully employed over the past two decades for similar products. Key outcomes included the provision of more detailed process information for the bismuth circuit to enable accurate sizing and engineering of process equipment, and confirmation of the

ability to process third-party materials, such as Rio Tinto's bismuth oxychloride intermediate from the Kennecott Smelter, to augment overall cobalt and bismuth production. The review also identified the potential to generate a saleable Agri-gypsum by-product through neutralization of process waste acid streams and defined test parameters to evaluate its quality and environmental suitability. In addition, feed source variability and process condition refinements were identified as critical factors influencing downstream product quality and residue stability, establishing the basis for subsequent test work.

This comprehensive review enabled finalization of the flowsheets, operating conditions, and design parameters required to test and validate the proposed process improvements at pilot scale during the following phases of metallurgical and hydrometallurgical operations.

PHASE 2 RESULTS

Phase 2 resulted in the successful retrieval and transportation of approximately fifteen tonnes of representative feed material from the NICO site to SGS Lakefield for downstream test work. The material was collected from existing stockpiles at the NICO site, placed into super sacs, transported by helicopter to a nearby road access point, and then shipped by truck to SGS.

PHASE 3 RESULTS

The Phase 3 concentrate production pilot successfully validated key improvements to the NICO flotation circuit and generated critical data for downstream hydrometallurgical processing. The flotation test work confirmed optimal crushing and grinding criteria, as well as reagent conditions required to maximize recovery of the four target metals into a single bulk flotation concentrate.

Fortune previously conducted a pilot plant in 2007 that determined that a grind size of approximately 54 microns, combined with an optimized reagent mix, yielded the best metallurgical performance. Under these conditions, a low bulk mass pull of roughly 4% was achieved, reducing the ore mass requiring downstream processing from approximately 4,650 tonnes per day to 180 tonnes per day. This substantial reduction in material volume represents a major cost benefit, significantly lowering transportation and handling requirements between the mine and the Alberta Refinery. Following bulk flotation, the concentrate was reground to 14 microns and subjected to secondary flotation to separate the bulk concentrate into distinct auriferous cobalt and bismuth concentrates. This secondary flotation step demonstrated effective separation and provided key inputs for the PDC and cost modeling of the hydrometallurgical circuits planned to validate subsequent project milestones. The 2024 mini-pilot exercise was conducted using the fresh samples collected in Phase 2 and produced a bismuth concentrate with an average grade of approximately 45% bismuth at 71% recovery. The cobalt concentrate averaged 1.5% cobalt at 82% recovery. Although the cobalt feed grade was below the deposit average, overall recoveries met expectations and validated the effectiveness of the optimized circuit design. The flowsheet is depicted in Figure 30.

In total, the 2024 mini-pilot program produced approximately 44 kilograms of dry bismuth concentrate and 277 kilograms of dry cobalt concentrate. Both concentrates were retained for hydrometallurgical testing in later project phases to further refine process parameters and confirm downstream recoveries.

Figure 1 Mini-pilot concentrator flowsheet schematic

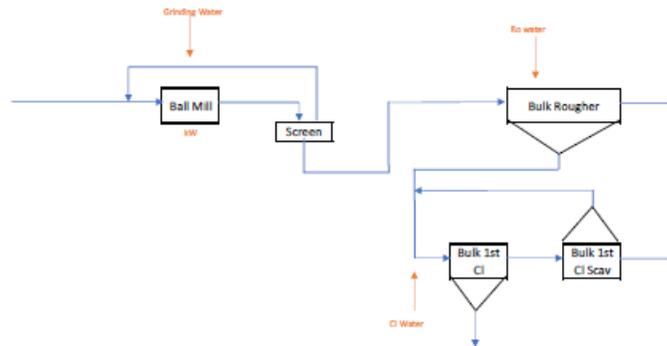


Figure I: Phase 1 Pilot Plant Flowsheet

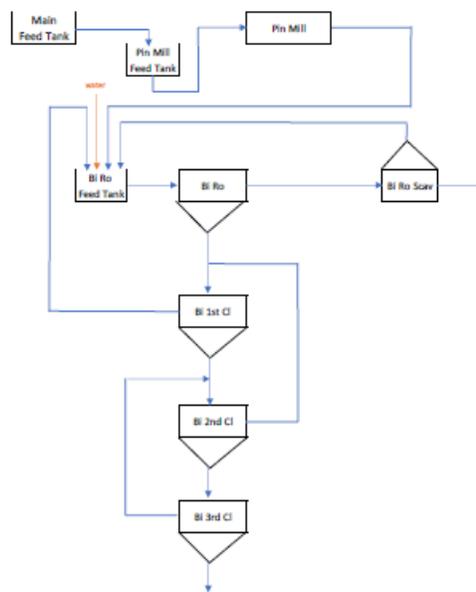


Figure II: Phase 2 Pilot Plant Flowsheet

PHASE 4 RESULTS

Worley's value enhancement studies identified several opportunities to improve the crushing, grinding, and flotation circuits of the NICO concentrator, resulting in an anticipated C\$5 million reduction in capital costs and a C\$1.5 million reduction in annual operating costs, primarily through lower energy

consumption. The studies recommended replacing conventional cone crushing and ball milling with High Pressure Grinding Rolls (HPGR) crushing and vertical stirred mill grinding to improve efficiency and reduce the circuit size.

Initial LABWAL HPGR tests conducted at SGS confirmed the suitability of the HPGR circuit for NICO ores. Additional Jar and Bond Work Index (BWI) tests were subsequently completed at Metso to characterize ore hardness and variability, supporting the design parameters for the updated comminution circuit.

The remaining HPGR comminution tests were completed to establish the Process design Criteria (PDC) for the concentrator. Labwall HPGR test work at SGS, supplemented by Jar and BWI testing at Metso on four representative ore types provided a quantitative measure of ore hardness and grindability across the deposit. The results confirmed the suitability of HPGR as a replacement for conventional crushing and milling equipment and generated the data required to finalize the PDC for the concentrator flowsheet, including equipment sizing and power requirements for subsequent engineering.

The Jameson cell trade-off study and associated flotation test work demonstrated a 3–5% improvement in fine bismuth and gold recoveries from the bulk rougher tailings, validating the inclusion of Jameson cells in the flotation circuit. Reject material from these tests were used to produce a cobalt concentrate grading approximately 4% cobalt, meeting the design grade criteria for downstream hydrometallurgical testing in the next project milestone.

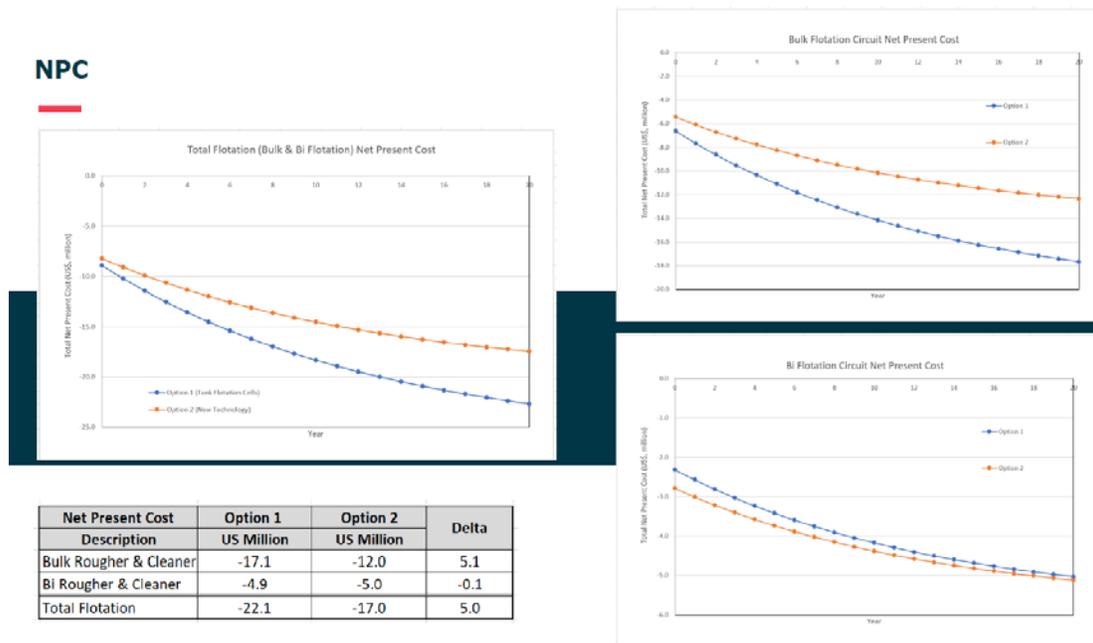


Figure 2 Flotation enhancement study summary

PHASE 5A RESULTS

Phase 5A hydrometallurgical test work successfully advanced development of the bismuth processing circuits for the NICO Project. Bulk ferric chloride leaching of the Milestone 1 bismuth concentrate, followed by cementation onto iron powder, achieved high recoveries of approximately 97% and produced a bismuth cement averaging 95% purity. Large-scale testing demonstrated that the leach residence time could be reduced from three hours to one hour using a single-stage configuration without loss of recovery. This optimization supports a 67% reduction in circuit size and lower associated capital and operating costs for the bismuth circuit at the Alberta Refinery.

Lixiviant regeneration tests were completed using hydrogen peroxide and a lower-cost chlorine alternative, both of which proved effective in enabling reagent recycling and improving water management efficiency. The results from these tests were incorporated into the PDC, supporting a more compact, cost-effective, and environmentally sustainable configuration for the bismuth hydrometallurgical circuit.

Compatibility testing with Rio Tinto's bismuth oxychloride intermediate from the Kennecott Smelter confirmed that blending at a 1:10 ratio with Fortune's bismuth concentrate did not negatively impact bismuth recoveries. This validation demonstrated the potential to augment Fortune's annual bismuth production by 10–20% and confirmed the process flexibility to accommodate third-party feedstocks in future refinery operations.

Gold deportment studies conducted during ferric chloride leaching confirmed that gold remained within the leach residue, enabling downstream recovery. Subsequent CIL cyanidation testing on this residue achieved high gold extraction, validating its potential as an alternative recovery stream. However, the preferred approach remains blending the bismuth leach residue with cobalt concentrate, followed by pressure oxidation and cyanidation of the combined residue, which will be further developed in subsequent project phases.

Detailed results summary – Bismuth recovery from the selective concentrate

Bismuth Leaching

Parametric

The metallurgical response of the bismuth concentrate feed-sample to ferric chloride leaching was investigated at bench scale through batch tests. Classic agitated leaching was performed under controlled temperature, and with intermittent pH and oxidation potential (ORP) measurements. The test products were subjected to detailed characterization and subjected to chemical analyses. The tests outcome consisted of detailed metallurgical balances. The test conditions are summarized in Table 1 and the results in Table 2 through Table 5.

In summary:

The parametric ferric chloride leaching test work determined the optimized conditions as follows: single stage, 8.4% wt. solids, one hour batch-reaction time at 85 °C (Test BiL8). Initial acidity of the lixiviant was reduced from 20 g/L HCl equivalent to 7 g/L HCl equivalent. These conditions produced 99.5% bismuth leaching efficiency from the NICO feed (Test BiL8) and 99.3% from the NICO/Rio Tinto feed blend (Test RTL2). The design criteria emerging from test BiL8 (supported by separate bulk-validation tests) are summarized in the dedicated sub-section.

Test RTL2 investigated the metallurgical response to ferric chloride leaching of a 1:10 blend ratio of NICO and Rio Tinto feed, on an equivalent bismuth content basis. Test BiL9 confirmed the possibility of using hydrogen-peroxide regenerated lixiviant (Test FeOx2) producing 98.5% extraction with 1.2% Bi grade in the residue. One additional bulk leaching test BiL10 was run under the previously determined optimum conditions to generate more cementation feed solution. The results confirmed the validated conditions

Furthermore, three additional leach tests were conducted.

For BiL11, using chlorine gas regenerated FeOx4 liquor diluted with 125 g/L NaCl to reduce the Fe level down to the pre-defined target of ~42 g/L Fe³⁺ and still maintain target total Cl in solution. The test produced 99.5% bismuth extraction, leaving 0.38% Bi in the leach residue, hence validating the effectiveness of the chlorine-gas regenerated lixiviant.

BiL12 and BiL13 investigating the effect of reducing the NaCl addition to 75 g/L NaCl and 25 g/L NaCl, respectively (while holding the Fe³⁺ at 41-42 g/L). They produced 99.1% and 97.7% leaching extraction, suggesting the possibility of allowing the sodium chloride concentration to drop to 75 g/L NaCl, but not below that value

Additional comments are provided in the leaching design criteria section.

Table 1 Ferric chloride leaching conditions summary

Test ID	BiL1	BiL2	BiL3	BiL4	BiL5	BiL6	BiL7
Feed Concentrate	Bi Conc Blend	Bi Conc Blend	Bi Conc Blend	Bi Conc Blend	BiL4 Residue	Bi Conc Blend	Bi Conc Blend
Objective	Base line	FA optimization	FA optimization	Temp. optimization	Temp. optimization	Bulk validation	Bulk validation+LSR
Secondary Feed	-	-	-	-	-	-	-
Lixiviant	41.7 g/L Fe3+, 125 g/L NaCl	41.7 g/L Fe3+, 125 g/L NaCl	41.7 g/L Fe3+, 125 g/L NaCl	41.7 g/L Fe3+, 125 g/L NaCl	41.7 g/L Fe3+, 125 g/L NaCl	41.7 g/L Fe3+, 125 g/L NaCl	41.7 g/L Fe3+, 125 g/L NaCl
Simulated Leach Stage	1	1	1	1	2	1	1
Target Acidity (g/L HCl)	20	10	7	7	7	7	7
Actual Acid Add'n (kg/t)	189	101	72	67	71	74	92
Initial % Solids	8.5%	8.5%	8.5%	8.5%	8.5%	8.5%	8.5%
Temperature (°C)	85	85	85	ambient	85	85	85
Retention Time (h)	3	3	3	6	2	3	3
Final PLS Acidity (g/L HCl)	12.6	4.2	2.1	4.7	7.7	3.1	2.7
Acid Consumption (kg/t)	73	64	54	24	1	48	92
Final Filtrate Ferric (g/L Fe(III))	2.5	-0.2	2.0	11.5	16.1	0.3	0.0
Ferric Consumption (kg/t)	328	351	333	247	201	358	361
Initial pH	0.05	0.31	0.56	0.27	-0.59	-0.07	0.19
Final Pulp pH	0.27	0.79	1.09	0.24	-0.87	0.67	0.72
Initial ORP (mV)	763	888	896	782	789	803	950
Final Pulp ORP (mV)	438	371	345	467	498	348	389
Weight Loss (%)	53%	48%	47%	39%	23%	50%	50%
Test ID	BiL8	BiL9	BiL10	BiL11	BiL12	BiL13	RTL2
Feed Concentrate	Bi Conc Blend	Bi Conc Blend	Bi Conc Blend Pail 2	Bi Conc Blend Pail 2	Bi Conc Blend Pail 2	Bi Conc Blend Pail 2	Bi Conc Blend
Objective	Confirm Kinetics	Confirm use of regenerated HP lixiviant	Bulk validation+LSR	Confirm use of regenerated Cl2 lixiviant	Evaluate lower Cl content in leach	Evaluate lower Cl content in leach	RT feed base line
Secondary Feed	-	-	-	-	-	-	RT Sample (Bag 1)
Lixiviant	41.7 g/L Fe3+, 125 g/L NaCl	Diluted FeOX2 Filtrate	41.7 g/L Fe3+, 125 g/L NaCl	Diluted FeOX4 Filtrate	41.7 g/L Fe3+, 75 g/L NaCl	41.7 g/L Fe3+, 25 g/L NaCl	41.7 g/L Fe3+, 125 g/L NaCl
Simulated Leach Stage	1	1	1	1	1	1	1
Target Acidity (g/L HCl)	7	7	7	7	7	7	7
Actual Acid Add'n (kg/t)	83	77	83	75	105	97	91
Initial % Solids	8.5%	8.4%	8.5%	8.5%	8.7%	8.9%	8.5%
Temperature (°C)	85	85	85	85	85	85	85
Retention Time (h)	1	1	1	1	1	1	3
Final PLS Acidity (g/L HCl)	-	2.8	2.8	1.9	5.7	6.7	7.7
Acid Consumption (kg/t)	83	53	83	59	50	31	12
Final Filtrate Ferric (g/L Fe(III))	0.9	43.5	2.6	2.1	0.8	1.1	1.3
Ferric Consumption (kg/t)	353	-29	344	349	358	354	348
Initial pH	0.11	0.51	0.03	0.20	0.16	0.48	0.30
Final Pulp pH	0.60	1.27	0.72	0.67	0.70	0.80	1.14
Initial ORP (mV)	882	718	861	912	781	774	848
Final Pulp ORP (mV)	446	458	468	490	495	505	281
Weight Loss (%)	46%	43%	50%	46%	56%	55%	50%
Blue font indicates Revision 1 test-data							
* all reagent additions/consumptions relative to dry feed equivalent							
** all acid data expressed as HCl equivalent							

Table 2 Ferric chloride leaching solution departments

Solution Department (%)	BiL1	BiL2	BiL3	BiL4	BiL5	BiL6	BiL7
Bi	99.7	99.5	98.7	73.1	99.6	99.7	99.6
Co	0	2	2	5	9	3	3
Cu	23	24	16	7	38	28	25
Fe	81	78	76	76	71	79	78
Mg	20	15	14	10	8	14	14
Al	39	32	29	16	16	30	27
Cr	42	42	40	31	15	41	40
Mn	30	26	24	15	12	26	23
Ca	20	15	13	13	3	15	14
Na	100	100	100	100	100	100	100
As	4	3	3	2	10	6	4
Ag	70	50	54	17	51	51	48
Pb	98	97	95	72	93	98	97
Sb	88	79	71	45	41	76	78
Sn	65	62	62	61	55	65	-
Solution Department (%)	BiL8	BiL9	BiL10	BiL11	BiL12	BiL13	RTL2
Bi	99.5	98	99.7	99.5	99.1	97.7	99.3
Co	2	7	3	5	0	1	2
Cu	13	17	11	14	15	12	12
Fe	76	76	79	76	79	79	77
Mg	12	16	13	16	11	10	11
Al	23	29	24	29	21	18	24
Cr	39	52	41	58	39	40	38
Mn	21	42	21	40	20	18	21
Ca	12	18	14	18	13	12	13
Na	100	100	100	100	100	99	100
As	3	8	4	4	9	11	8
Ag	60	59	52	58	60	71	75
Pb	96	96	97	98	97	96	98
Sb	76	46	83	69	68	56	63
Sn	63	62	65	63	67	67	61

Blue font indicates Revision 1 test-data

Table 3 Ferric chloride leaching feed vs. residue difference extractions

Feed/Residue Diff. Extract. (%)	BiL1	BiL2	BiL3	BiL4	BiL5	BiL6	BiL7
Bi	99.7	99.5	99	73	99.6	99.7	99.6
Co	9	1	-2	11	9	6	6
Cu	25	20	10	8	38	27	23
Fe	31	21	12	14	14	21	17
Mg	24	15	13	17	5	14	16
Al	41	31	27	21	13	27	26
Cr	53	48	47	39	23	50	50
Mn	35	28	26	21	13	28	27
Ca	25	16	12	15	4	18	15
Na	16	3	9	2	7	-3	5
As	61	39	37	35	34	60	40
Au	-	-	-	-	-	21	0
Ag	49	46	45	19	47	50	50
Pb	98	97	95	74	93	97	97
Sb	89	80	72	49	40	77	79
Sn	53	48	47	39	23	50	50
Feed/Residue Diff. Extract. (%)	BiL8	BiL9	BiL10	BiL11	BiL12	BiL13	RTL2
Bi	99.5	98	99.7	99.5	99.2	97.8	99.2
Co	0	4	7	4	20	23	2
Cu	3	9	8	12	24	21	8
Fe	9	4	13	9	26	23	15
Mg	12	6	22	13	30	27	11
Al	21	9	29	16	32	26	20
Cr	46	43	50	58	56	55	50
Mn	22	11	30	19	34	31	22
Ca	9	3	22	11	26	24	13
Na	-1	-7	7	-2	18	18	43
As	14	46	50	33	64	64	57
Au	-	-	-	-	-	-	-
Ag	43	40	53	53	60	59	76
Pb	96	92	96	96	97	96	98
Sb	77	48	83	70	72	62	65
Sn	46	43	50	46	56	55	66

Blue font indicates Revision 1 test-data

Table 4 Ferric chloride leaching final solutions

Final Solution (mg/L)	BiL1	BiL2	BiL3	BiL4	BiL5	BiL6	BiL7
Bi	52100	49600	52200	34600	20600	48200	46800
Co	1	11	13	7	62	13	13
Cu	306	355	230	38	745	357	326
Fe	44300	43800	43800	39100	42200	42400	41100
Mg	195	163	147	63	103	138	130
Al	284	250	227	90	156	226	199
Cr	8	9	8	5	3	8	7
Mn	18	17	15	7	10	16	14
Ca	361	294	265	180	83	257	244
Na	49400	46200	49200	46700	47700	49000	47500
As	72	94	99	68	348	109	108
Ag	3	1	2	0	2	1	1
Pb	143	150	146	91	58	141	140
Sb	275	247	231	123	106	233	225
Sn	2	2	2	2	2	2	< 2
Fe(II)	41800	44000	41800	27600	26100	42100	41100
Final Solution (mg/L)	BiL8	BiL9	BiL10	BiL11	BiL12	BiL13	RTL2
Bi	49800	48000	53500	47100	46200	46000	56700
Co	12	35	12	28	1	5	11
Cu	187	243	151	188	178	140	162
Fe	42200	43500	46300	41800	40300	40700	42800
Mg	114	173	112	169	90	84	120
Al	173	268	167	249	129	118	186
Cr	8	14	8	13	6	7	7
Mn	13	40	12	34	10	10	13
Ca	229	396	237	374	202	197	244
Na	47400	45500	52800	47500	28200	9730	48600
As	104	198	104	127	151	190	168
Ag	2	2	1	2	2	3	4
Pb	144	235	152	220	135	137	339
Sb	228	141	257	213	186	151	353
Sn	2	2	2	2	2	2	19
Fe(II)	41300		43700	39700	39500	39600	41500

Blue font indicates Revision 1 test-data

Table 5 Ferric chloride leaching final residues

Final Residue (g/t)	BiL1	BiL2	BiL3	BiL4	BiL5	BiL6	BiL7
Bi	2810	4060	11400	195000	952	2830	3700
Co	8660	8560	8540	6500	7710	8380	8450
Cu	18800	18200	20000	17800	14400	17200	18200
Fe	191000	200000	216000	185000	207000	208000	217000
Mg	14600	15000	14900	12400	15300	15600	15300
Al	8250	8810	9060	8590	9690	9670	9740
Cr	200	200	200	200	200	200	200
Mn	791	810	800	751	849	833	840
Ca	26500	27300	27700	23300	29200	27500	28500
Na	2280	2420	2210	2060	2500	2660	2450
As (%)	3	5	5	4	4	3	5
S	-	-	-	-	-	28	24
Au	-	-	-	-	-	622	781
Ag	26	25	25	32	22	24	24
Pb	65	74	113	546	49	65	75
Sb	674	1100	1490	2400	1860	1340	1190
Sn	20	20	20	20	20	20	20
Final Residue (g/t)	BiL8	BiL9	BiL10	BiL11	BiL12	BiL13	RTL2
Bi	4050	12000	2930	3800	8520	21500	7100
Co	8260	7510	8370	8020	8090	7660	8160
Cu	21000	18700	22000	19300	20300	20500	20300
Fe	220000	219000	229000	222000	220000	222000	207000
Mg	14800	14900	14300	14700	14400	14700	15200
Al	9670	10500	9500	10300	10200	10900	9920
Cr	200	200	200	158	200	200	200
Mn	832	895	821	875	861	878	845
Ca	28000	28300	26300	27700	28200	28000	27000
Na	2390	2420	2410	2460	2390	2350	2470
As (%)	6	4	4	5	3	3	3
Au	0	0	0	0	0	0	0
Ag	25	25	23	21	22	22	23
Pb	94	176	94	86	89	122	107
Sb	1210	2620	996	1600	1810	2390	3390
Sn	20	20	20	20	20	20	203

Blue font indicates Revision 1 test-data

Liquid solid separation

Static and dynamic settling testing were conducted on a ferric chloride bulk leaching discharge slurry sample. The results allowed for the most accurate estimation of the resulting high-rate thickener unit areas and triggered a change in the design criteria, as detailed in the dedicated sub-section. The resulting underflow was subjected to rheology determinations as well as vacuum filtration testing. Additional vacuum filtration tests were conducted. Results summaries are included in Table 6 and Figure 3.

The results confirmed the previous design criteria providing optimized cake thickness and resulting belt-vacuum filter output, as summarized in the dedicated sub-section.

Table 6 Leach discharge vacuum filtration test conditions and results summary

Company:	Fortune	Table No:	1 of 3
Sample:	BiL10 (Underflow)	Date:	05-Feb-25
Project No.	11758-11	By:	MW

Filter cloth: Testori P 4408TC, Material: polypropylene, Weave: plain, Permeability: 1.03 cfm, Warp/Weft: multi/multi, Finish: heat-set/ calendered

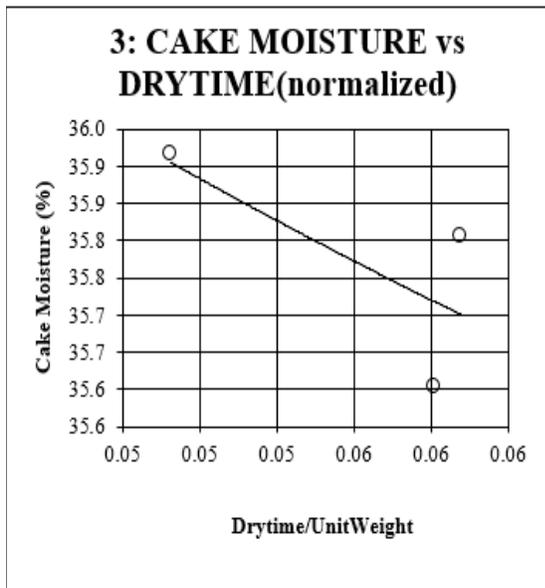
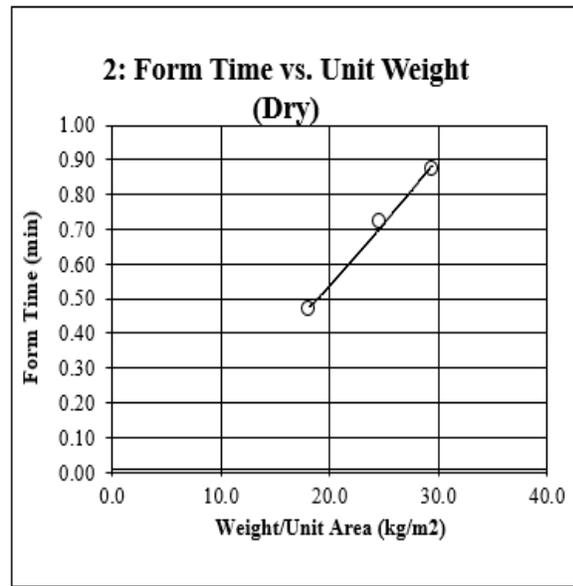
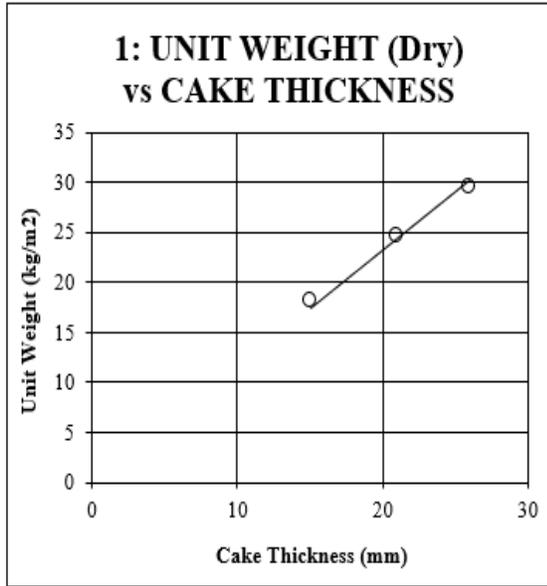
Filteraid:	None		
pH:	< 1	Particle Size, d80:	31 μ m
Feed Temp:	80 *C	Feed solids content (density):	45 % wt. solids
S.G. Solids:	3.18	Filter dia:	113 mm (for Pannevis Filter)
S.G Liquid:	1.20		

Test ID	1	2	3
Actual cake thickness, m	0.026	0.021	0.015
Filter area m ²	0.0100		
Actual cake volume, L	0.261	0.211	0.150
Cake wet weight, kg	0.460	0.383	0.284
Cake dry weight, kg	0.295	0.247	0.182
Wet cake solids, %	64.2	64.4	64.1
Wet cake density, kg/L	1.763	1.820	1.888
Dry cake density, kg/L	1.132	1.172	1.210
Filtrate volume, L	0.152	0.132	0.094
Filter loading, L/m ² , L= (Volume/Area)	41.2	34.2	24.4
Form time, h	0.015	0.012	0.008
Dry time, h	0.029	0.024	0.016
Form/Dry Time Ratio	0.50	0.50	0.51
Total filtration time (form+dry), h	0.043	0.036	0.023
Filtrate TSS, mg/L	184	160	112

*Calculated filter throughput versus test conditions			
Vacuum level, form, inches mercury	20		
Vacuum level, dry, inches mercury	17		
Feed solids, %wt	45.0		
Final wet cake moisture, %wt	35.8	35.6	35.9
Average - above, %	35.8		
Percent deviation - above, %	0.1	-0.48	0.39
Dry solids capacity, kg/m ² h	679	687	778
Filter capacity, L/m ² h	950	953	1045

* From raw test data / not for sizing any particular type of filter.
 Conversion: 0.098 bar = 2.896" Hg = 1 mH₂O = 3.28 ft H₂O = 1.42psi

Figure 3 Leach discharge vacuum filtration relevant dependencies



Relationships:

Unit Weight =	1.158	*	Cake Thickness
Form Time =	0.011360	*	Unit Weight ^ 1.287
Moisture =	31.723	*	(Drytime/Unit Weight) ^ -0.042

Bismuth Cementation

Parametric

The metallurgical response to cementation on iron powder was investigated at bench scale through batch tests. Classic agitated mixing cementation was performed under controlled temperature, and with intermittent pH and oxidation potential (ORP) measurements. The test products were subjected to detailed characterization and subjected to chemical analyses. The tests outcome consisted of detailed metallurgical balances. The test conditions are summarized in Table 7 and the results in Table 8 through Table 11.

In summary:

The parametric cementation from the ferric leach pregnant leach solution (“PLS”) determined the optimized conditions as follows: single stage, 15 minutes batch-reaction time at 60 °C with a ~100% stoichiometric dose of iron relative to bismuth (Tests BiC11 and BiC12) under the conditions as defined (free acidity, pH, etc.). The difference between tests BiC11 and BiC12 was the residence time of 10 minutes and 20 minutes, respectively, leading to designating the design criteria to 15 minutes.

Both tests BiC11 and BiC 12 produced 99.99% cementation efficiency vs. solids from the NICO feed, leaving from 3 to 6 mg/L Bi in the barren solution. Both tests BiC11 and BiC 12 produced cements consistently assaying above 99% Bi by direct assays and above 98% by difference vs. impurities. Most notably, both cement products contained less than 0.6% Fe. This value was estimated by difference vs. impurities based on analytical results from laboratories. The design criteria emerging from bulk-validation tests BiC11 and BiC12 are summarized in the dedicated sub-section.

Unoptimized test RTBiC1 also produced 99.99% cementation efficiency from the NICO/Rio Tinto feed blend, producing a cement assaying 91.5% Bi with about 3.2% Fe content. The additional cementation test work was triggered by the adverse response to purification. The results negated the need for it, simplifying the process whilst generating a cement of increased purity, and still without the need for the counter-current sequence.

Table 7 Cementation conditions summary

Test ID	BiC1	BiC2	BiC3	BiC4	BiC5	BiC6	RTBiC1
Feed Solution	BiL2 Filtrate	BiL3 Filtrate	BiL3 Filtrate	BiL6 Filtrate	BiL6 Filtrate	BiL7 Filtrate	RTL2 Filtrate
Objective	Base line	Counter-current PLS	Single stage	Bulk validation	Bulk validation	Bulk validation	Single stage
Feed Solids (if applicable)	Bi Conc Blend	BiC1 Part B Washed Cake	-	-	-	-	-
Simulated Stage	1	2	1	1	1	1	1
Pre-Neut pH Limit	1.0	-	-	-	-	-	-
Acidity to Cementation (g/L HCl)	2.4	2.1	2.1	3.1	3.1	2.7	7.7
Iron Powder Source	Atomet 110	N/A	Atomet 110	Atomet 110	Atomet 110	Atomet 110	Atomet 110
Iron Powder Dosage (% Bi)	109%	0%	103%	104%	104%	104%	110%
Cementation pH Limit	1.5	1.5	1.5	1.5	1.5	1.5	1.5
Final Pulp pH	1.50	1.42	0.70	0.70	0.79	0.97	1.38
Cementation Temperature (°C)	60	60	60	60	60	60	60
Fe Addition (kg/kg Bi)	0.44	-	0.41	0.45	0.45	0.46	0.44
Maximum Cementation Time (min)	45	45	45	45	10	10	10
Actual Cementation Time (min)	21	45	45	45	10	10	10

Table continued with additional data - Revision R1

Test ID	BiC7	BiC8	BiC9	BiC10	BiC11	BiC12
Feed Solution	BiL10 Recovered PLS	BiL10 Recovered PLS				
Objective	As is FA	Pre-neut FA	FA optimization	Pre-neut FA	As Is FA Bulk validation	As Is FA Bulk validation
Feed Solids (if applicable)	-	-	-	-	-	-
Simulated Stage	1	1	1	1	1	1
Pre-Neut pH Limit	-	1.0	-	1.0	-	-
Acidity to Cementation (g/L HCl)	2.3	2.3	2.3	2.3	2.3	2.3
Iron Powder Source	Atomet 110	Atomet 110				
Assumed Iron Powder Dosage (% Bi)	100%	100%	104%	104%	100%	100%
Actual Iron Powder Dosage (% Bi)	100%	100%	104%	104%	101%	101%
Actual Iron Powder Dosage (% Ov.)	87%	87%	91%	91%	88%	88%
Cementation pH Limit	1.5	1.5	1.5	1.5	1.5	1.5
Final Pulp pH	0.64	1.03	0.61	1.24	0.55	0.51
Cementation Temperature (°C)	60	60	60	60	60	60
Fe Addition (kg/kg Bi)	0.40	0.40	0.42	0.42	0.40	0.40
Maximum Cementation Time (min)	20	20	20	20	10	20
Actual Cementation Time (min)	20	20	20	20	10	20

Table 8 Cementation solids departments

Solids Department (%)	BiC1	BiC2	BiC3	BiC4	BiC5	BiC6	RTBiC1
Bi	100.00	48	99.95	99.9	98.1	99.95	99.99
Co	76	11	12	27	4	1	69
Cu	98	8	82	90	29	84	100
Fe	3	1	2	2	4	0	3
Mg	1	1	2	2	2	1	2
Al	5	2	6	5	6	4	5
Cr	41	39	59	60	61	43	61
Mn	14	16	9	12	11	9	14
Ca	1	2	2	2	2	2	3
Na	0	0	0	0	0	0	0
As	97	16	-	97	90	89	94
Ag	57	59	94	93	85	95	97
Pb	23	9	10	12	15	9	13
Sb	99	42	98	98	98	97	97
Sn	35	33	21	31	37	35	57

Table continued with additional data - Revision R1						
Solids Department (%)	BiC7	BiC8	BiC9	BiC10	BiC11	BiC12
Bi	99.99	99.61	99.99	99.99	99.99	99.99
Co	1	2	12	38	17	17
Cu	70	31	99	100	98	98
Fe	0	1	1	1	0	0
Mg	1	1	1	2	2	2
Al	3	4	4	5	4	4
Cr	51	52	52	53	52	53
Mn	6	6	10	11	8	7
Ca	2	3	2	2	2	2
Na	0	0	0	0	0	0
As	87	72	97	94	92	96
Ag	94	85	94	94	94	94
Pb	4	5	9	20	11	11
Sb	100	96	99	99	98	98
Sn	33	33	33	33	34	34

Table 9 Cementation feed vs. filtrate difference efficiencies

Feed/Filtrate Cement. (%)	BiC1	BiC2	BiC3	BiC4	BiC5	BiC6	RTBiC1
Bi	100	4.1	100.0	99.9	98.0	99.9	100.0
Co	61	-77	-18	-1	-31	-54	51
Cu	96	5	84	90	27	81	100
Fe	-44	-8	-42	-44	-50	-67	-39
Mg	0	-2	5	2	4	-23	11
Al	1	-6	8	8	3	-24	12
Cr	2	-8	6	4	5	-91	0
Mn	-98	-12	-137	-127	-142	-198	-194
Ca	0	-1	5	4	-1	-25	12
Na	2	-2	3	8	1	-10	11
As	97	-66	97	97	90	88	96
Ag	1	50	96	94	84	94	98
Pb	22	-13	13	13	13	-1	19
Sb	98	13	98	99	97	97	98
Sn	1	-1	5	4	2	#VALUE!	61

Table continued with additional data - Revision R1						
Feed/Filtrate Cement. (%)	BiC7	BiC8	BiC9	BiC10	BiC11	BiC12
Bi	99.99	99.66	99.99	99.99	99.99	99.99
Co	-64	-59	-45	0	-38	-34
Cu	44	-31	98	99	97	97
Fe	-40	-39	-38	-37	-32	-35
Mg	-22	-21	-22	-19	-18	-19
Al	-24	-22	-23	-21	-20	-21
Cr	-25	-22	-23	-20	-24	-22
Mn	-213	-207	-214	-210	-208	-206
Ca	-9	-8	-8	-7	-6	-8
Na	9	7	9	9	12	13
As	86	70	97	94	91	96
Ag	94	85	94	94	94	94
Pb	8	10	14	35	16	19
Sb	100	96	99	99	98	98
Sn	1	-2	1	-2	1	2

Table 10 Cementation final solutions

Final Solution (mg/L)	BiC1	BiC2	BiC3	BiC4	BiC5	BiC6	RTBiC1
Bi	1	49800	24	68	1010	26	2
Co	5	22	15	13	17	20	6
Cu	0	217	37	38	266	62	0
Fe	67700	46900	65200	63700	65100	70500	63700
Mg	229	149	146	141	136	164	114
Al	342	240	219	216	225	254	175
Cr	16	9	8	8	7	14	8
Mn	49	17	38	38	40	41	42
Ca	504	265	264	258	265	313	230
Na	49500	50100	49900	47200	49600	53600	46200
As	3	163	3	3	11	13	8
S	0	0	0	0	0	0	0
S=	0	0	0	0	0	0	0
S0	0	0	0	0	0	0	0
Au	0	0	0	0	0	0	0
Ag	0	1	0	0	0	0	0
Pb	122	164	133	128	126	145	293
Sb	2	201	5	3	6	6	7
Sn	2	2	2	2	2	2	8

Table continued with additional data - Revision R1

Final Solution (mg/L)	BiC7	BiC8	BiC9	BiC10	BiC11	BiC12
Bi	6	178	6	6	6	3
Co	21	19	18	12	17	17
Cu	86	195	4	1	4	5
Fe	65600	63400	64600	62000	61800	63400
Mg	139	133	138	131	134	135
Al	210	200	208	198	203	205
Cr	10	9	9	9	9	9
Mn	39	37	39	37	38	38
Ca	261	252	260	249	255	260
Na	48800	48100	48400	47100	47200	46500
As	15	31	3	6	9	4
S	0	0	0	0	0	0
S=	0	0	0	0	0	0
S0	0	0	0	0	0	0
Au	0	0	0	0	0	0
Ag	0	0	0	0	0	0
Pb	142	135	133	97	129	125
Sb	1	9	2	2	4	4
Sn	2	2	2	2	2	2

Table 11 Cementation final solids

Final Solids (g/t)	BiC1	BiC2	BiC3	BiC4	BiC5	BiC6	RTBiC1
Bi	959,000	932,000	916,000	921,000	890,000	1,030,000	915,000
Co	296	56	38	94	13	4	233
Cu	108	386	3650	6380	1870	6020	2680
Fe	45200	5590	27000	23900	50800	2950	32600
Mg	40	40	40	40	40	40	40
Al	315	100	237	215	263	194	153
Cr	200	118	200	200	200	200	200
Mn	146	65	68	91	86	78	112
Ca	100	100	100	100	100	100	100
Na	100	182	306	203	393	100	100
As	1970	650	***	1920	1650	1980	2440
S	0	0	0	0	0	0	0
S=	0	0	0	0	0	0	0
S0	0	0	0	0	0	0	0
Au	0	0	0	0	0	0	0
Ag	2	27	44	26	22	26	72
Pb	675	339	242	316	391	278	701
Sb	2590	2950	4320	4350	4100	3880	5840
Sn	20	20	20	20	20	20	201

Bi assays > 100% - within the method estimated measurement uncertainty (MU) limits of 5%

Table continued with additional data - Revision R1						
Final Solids (g/t)	BiC7	BiC8	BiC9	BiC10	BiC11	BiC12
Bi	929000	939000	923000	906000	992000	995000
Co	4	8	49	149	67	65
Cu	4020	1830	5670	5430	5540	5530
Fe	6510	9080	15900	18300	5470	5780
Mg	40	40	40	40	40	40
Al	122	192	156	202	153	152
Cr	200	200	200	200	200	200
Mn	52	46	87	90	63	57
Ca	109	136	100	100	100	100
Na	153	263	285	406	100	200
As	2010	1660	2050	1940	1980	2070
S	0	0	0	0	0.04	0.02
S=	0	0	0	0	0	0
S0	0	0	0	0	0	0
Au	0	0	0	0	0	0
Ag	27	24	27	26	26	25
Pb	117	147	257	486	300	295
Sb	4140	4050	4150	3880	4200	4180
Sn	20	20	20	20	20	20
Cl	-	-	-	-	0.17	0.34

Sum of impurities						
Sample Analyses by:	BC11	BC12	BC11	BC12	BC11	BC12
	SGS		XPS		Average	
Metals, %	1.805	1.846				
Nonmetals, %	0.366	0.553				
Total impurities	2.172	2.399	1.848	1.800	2.010	2.099
Bi by diffe, %	97.828	97.601	98.152	98.200	97.990	97.901
Bi direct	99.200	99.500	n/a	n/a	n/a	n/a

Liquid solid separation

Direct discharge (i.e. un-thickened) cementation vacuum and pressure filtration tests were conducted on a bulk cementation discharge slurry sample.

The vacuum pressure filtration test conditions and results summary are provided in Table 12 with relevant dependencies illustrated in Figure 4. The pressure filtration test conditions and results summary are provided in Table 13, with relevant dependencies illustrated in Figure 5.

The results allowed to determine the optimized cake thickness and resulting belt-vacuum filter output. Vacuum filtration design criteria emerging from test 4 are summarized in the dedicated sub-section.

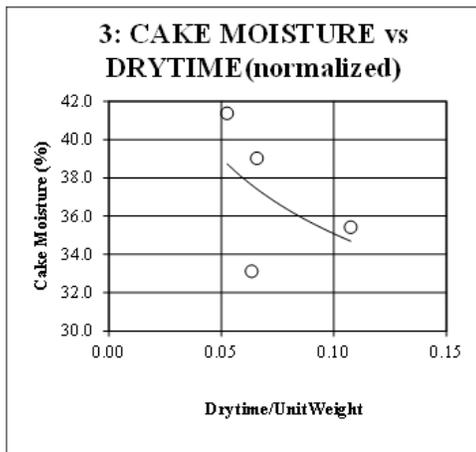
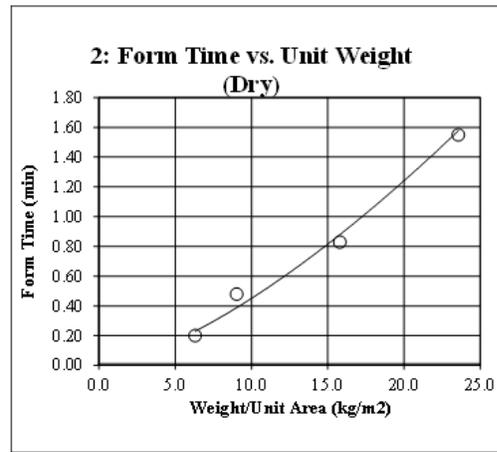
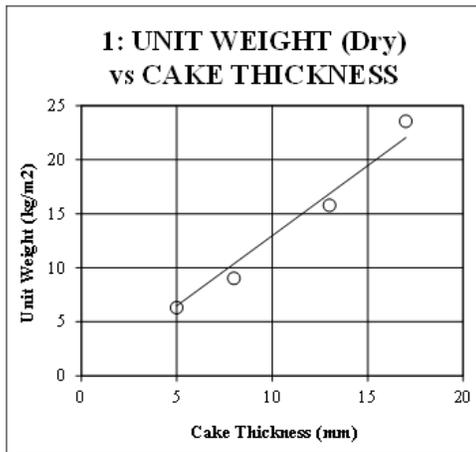
Table 12 Bismuth cement vacuum filtration test conditions and results summary

Company:	Fortune		Table No:	1 of 3
Sample:	BiC5 Final Pulp (direct discharge)		Date:	13-Nov-24
Project No.	11758-11		By:	MW
	0			
Filter cloth:	Micronics 861, Material: polypropylene, Permeability: 0.7 cfm, Weave: twill, Warp/Weft: multi/multi, Finish: calendered			
Filteraid:	None			
pH:	<1	Particle Size, d80:	39	µm
Feed Temp:	60 °C	Feed solids content (density):	4.2	% wt. solids
S.G. Solids:	9.22	Filter dia:	113	mm (for Pannevis Filter)
S.G. Liquid:	1.185			

Test ID	1	2	4	3
Actual cake thickness, m	0.005	0.008	0.013	0.017
Filter area m ²	0.0100			
Actual cake volume, L	0.050	0.080	0.130	0.170
Cake wet weight, kg	0.094	0.140	0.270	0.388
Cake dry weight, kg	0.063	0.091	0.158	0.236
Wet cake solids, % S	66.9	64.6	58.6	61.0
Wet cake density, kg/L	1.884	1.747	2.072	2.273
Dry cake density, kg/L	1.260	1.129	1.215	1.386
Filtrate volume, L	0.950	1.960	2.720	3.590
Filter loading, L/m ² , L= (Volume/Area)	99.7	203.4	284.2	375.0
Form time, h	0.003	0.008	0.014	0.026
Dry time, h	0.007	0.016	0.014	0.026
Form/Dry Time Ratio	0.50	0.49	1.00	1.00
Total filtration time (form+dry), h	0.010	0.024	0.028	0.052
Filtrate TSS, mg/L	<10	<10	<10	<10
*Calculated filter throughput versus test conditions				
Vacuum level, form, inches mercury	18			
Vacuum level, dry, inches mercury	7			
Feed solids, %wt	4.2			
Final wet cake moisture, %wt	33.1	35.4	41.4	39.0
Average - above, %	37.2			
Percent deviation - above, %	-11.1	-4.85	11.12	4.80
Dry solids capacity, kg/m ² h	630	374	571	456
Filter capacity, L/m ² h	9973	8418	10273	7258

* From raw test data / not for sizing any particular type of filter.
 Conversion: 0.098 bar = 2.896" Hg = 1 mH₂O = 3.28 ft H₂O = 1.42psi

Figure 4 Bismuth cement vacuum filtration relevant dependencies



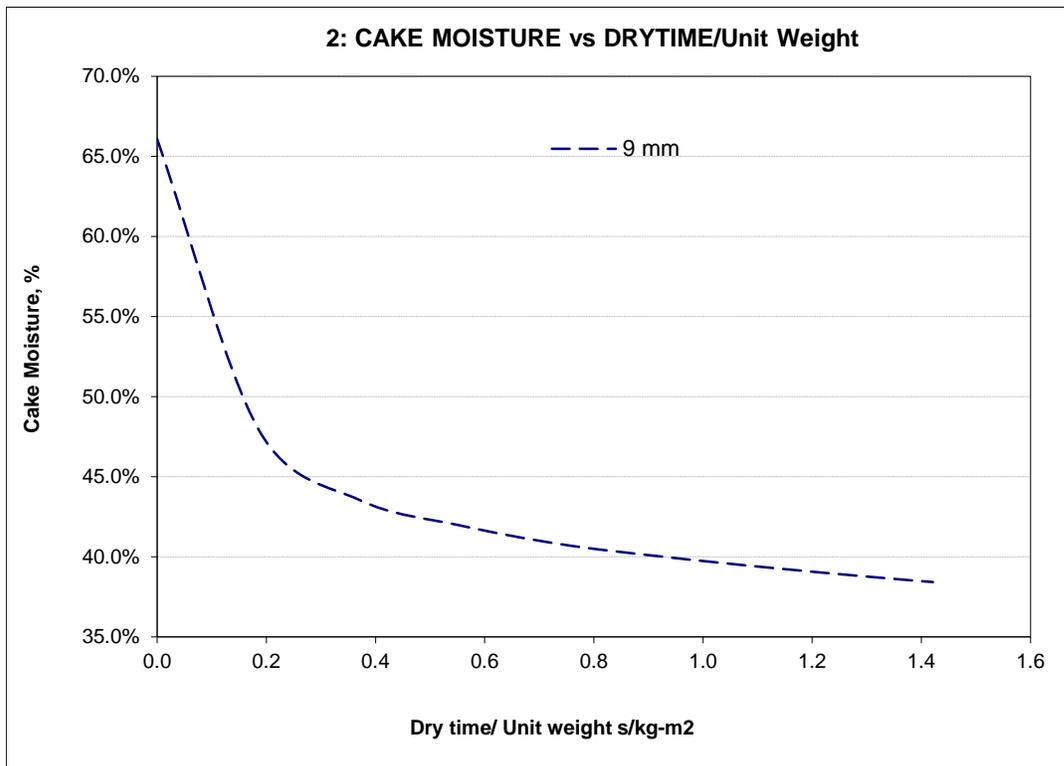
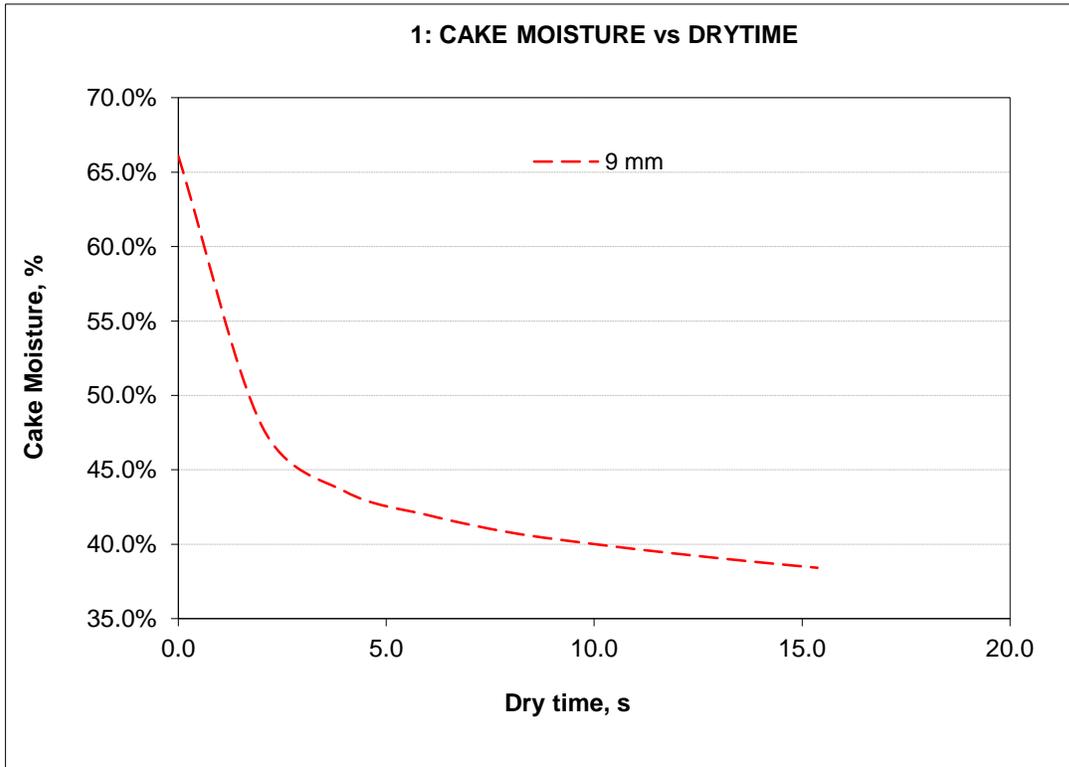
Relationships:

Unit Weight =	1.297	*	Cake Thickness
Form Time =	0.015549	*	Unit Weight ^ 1.462
Moisture =	24.598	*	(Drytime/Unit Weight) ^ -0.154

Table 13 Pressure filtration test conditions and results summary

Project No.: 11758-11		Date: Nov 12, 2024
Ref.: Fortune		
Sample I.D.: BiC5 Final Pulp (direct discharge)		
Particle Size, d_{80}: 39 μm	Pulp Temperature, $^{\circ}\text{C}$: 60	
Pulp pH: < 1		
Filter cloth: Micronics 861, Material: polypropylene, Permeability: 0.7 cfm, Warp/Weft: multi/multi Weave: twill, Finish: calendered		
Filter Aid, Type/Dosage, g/t: None		
Test ID	1	
¹ Actual cake thickness, m , H	0.0090	
Filter area m² , $A = (PI/4)*D^2$	0.0020	
Actual cake volume, L , $V=A*H$	0.018	
Cake wet weight, kg , CW	0.034	
Cake dry weight, kg , CD	0.021	
Wet cake solids, % , S	61.6	
Wet cake density, kg/L , $WD = CW/V$	1.952	
Dry cake density, kg/L , $DD=WD*CS/100$	1.202	
Filtrate volume, L , V	0.379	
Filter loading, L/m² , $L = (Volume/Area)$	202.2	
Form time, h	0.0103	
Dry time, h	0.0043	
Form/Dry	2.4177	
Total feeding time, h , T	0.0146	
Filtrate TSS, mg/L	7	
*Example of general filter throughput predictions versus test conditions		
Feed Pressure, bar	5.5	
Feed solids, %	4.2	
Final wet cake moisture, %	38.4	
² Dry solids capacity, kg/m²h , $KS=CD/(A*T)$	741	
³ Dry solids capacity, full cycle time, kg/m²h	60	
Filter capacity, L/m²h , $KL=L/T$	13,854	
* From raw test data / not for sizing any particular type of filter.		
¹ cake thickness is equivalent to half of the filter chamber thickness		
² Throughput (form and dry): including form and dry time only.		
³ Throughput (full cycle time): including 10 min of miscellaneous time including cake discharging and cloth washing and filter assembly in addition to form and dry time.		

Figure 5 Pressure filtration relevant dependencies



Bismuth leaching lixiviant regeneration

Parametric

The metallurgical response to ferric regeneration using controlled oxygen sparging and hydrogen peroxide addition was investigated at bench scale through batch tests. Classic agitated mixing tests were performed under controlled temperature, and with intermittent pH and oxidation potential (ORP) measurements. The test products were subjected to detailed characterization and subjected to chemical analyses. The tests outcome consisted of detailed metallurgical balances.

The test conditions are summarized in Table 14, and the results in Table 15 through Table 18. Relevant dependencies are illustrated in Figure 6.

In summary:

As expected, ferric chloride regeneration using oxygen was ineffective (~25% efficiency in 6 hours), whereas the hydrogen peroxide produced ~86% efficiency in 3 hours (Table 15).

Precipitated amount and losses were negligible, waiving the need for liquid-solid separation. Any solids formed would be redirected to leaching. However, the 204% peroxide addition vs. stoichiometry was deemed potentially uneconomical. For this reason, test FeOx3 was run at 125% total dosage (25% excess). It returned about 42% oxidation efficiency, out of which about 26% reported to solids formed during the process (corresponding to about 17% of overall iron precipitation). As a result, test FeOx4 was run using gaseous chlorine dosed over time to a 125% stoichiometric addition. It produced about 95% oxidation efficiency, during 3 hours of gas addition and one-hour additional holding time at 60 °C and without any solid precipitation.

About 1% of the stoichiometric dose of the chlorine was consumed to form acid (HCl), at a mass ratio of 0.008 t HCl/t ferric regenerated. Most of the chlorine excess was off-gassed.

Ferric chloride regeneration initial design criteria (i.e. unrevised R0) using hydrogen peroxide emerging from test FeOx2 were summarized in the dedicated sub-section of the previous unrevised release. They were deleted in this release. The regeneration criteria based on hydrogen peroxide were discarded for this revision (R1) and replaced with the design criteria emerging from test FeOx4 conducted with chlorine.

Table 14 Lixiviant regeneration conditions summary

Test ID	FeOx1	FeOx2	FeOx3	FeOx4
Objective	Base line	Base line	Base line	Base line
Feed Solution	BiC4 Filtrate	BiC4 Filtrate	Comp BiC11/12 Filtrate	Comp BiC11/12 Filtrate
Oxidant	O2	H2O2	H2O2	Cl2
Specific consumption, t PEq/t Fe oxidized*	4.71	0.72	0.92	0.84
Oxidant Addition, est. Stoich.	-	204%	125%	125%
Oxidant Addition, v/v/min (relative to feed vol.)	0.16	-	-	-
Oxidation ORP Target (mV)	N/A	650.0	-	-
Final pH Target	1.5	1.5	1.5	1.5
Neut. Reagent	H2SO4	H2SO4	H2SO4	*H2SO4
Specific consumption, t PEq/t Fe oxidized*	0.65	0.18	0.14	-
Oxidation Efficiency	25.3%	86.6%	41.8%	95.4%
Above as iron oxidized reporting to solution	23.2%	86.0%	26.4%	95.4%
Above as iron oxidized reporting to solids	2.1%	0.6%	15.4%	0.0%
Mass ratio of oxidized iron, Solids/Solution	0.1	0.007	0.581	0.000
Retention Time (h)	6.0	3.0	4.0	4.0
Temperature (°C)	60	60	60	60
Fe(II) / Fe (total) in Final Solution	0.73	0.02	0.54	0.0002

No H₂SO₄ was needed in test FeOx4. It produced FA ~3.2 g/L HCl pure equivalent from the excess of chlorine.
The difference to ~ 7 g/L HCl FA needed in the regenerated lixiviant will be completed using H₂SO₄ and NaCl top up.

Table 15 Lixiviant regeneration using hydrogen peroxide selected results – Test FeOx2

Ferric oxidative regeneration in solution, hydrogen peroxide, pH ~1.5, 60°C				
Time, hours	0	0.5	1.0	1.5
Iron oxidation - total, %	0.00	49.14	85.63	85.90
Iron oxidation in solution, %	0.00	48.79	85.01	85.28
t P.Eq. H2O2/t Fe oxidized	0	0.39	0.72	0.72
t P.Eq. H2SO4/t Fe oxidized		0.18		
H2O2 vs. Stoichiometry	0.00	111.34	203.51	203.51
ORP, mV Ag/AgCl	370	528	650	654
H2O2 grade, % wt.	50			
Technical grade H2O2, t/t Fe oxidized	1.43			
H2SO4 grade, % wt.	96			
Technical grade H2SO4, t/t Fe oxidized	0.19			
Technical grade H2O2, t/t purified Bi in cement	0.718			
Technical grade H2SO4, t/t purified Bi in cement	0.093			

Figure 6 Lixiviant – ferric regeneration using hydrogen peroxide kinetics

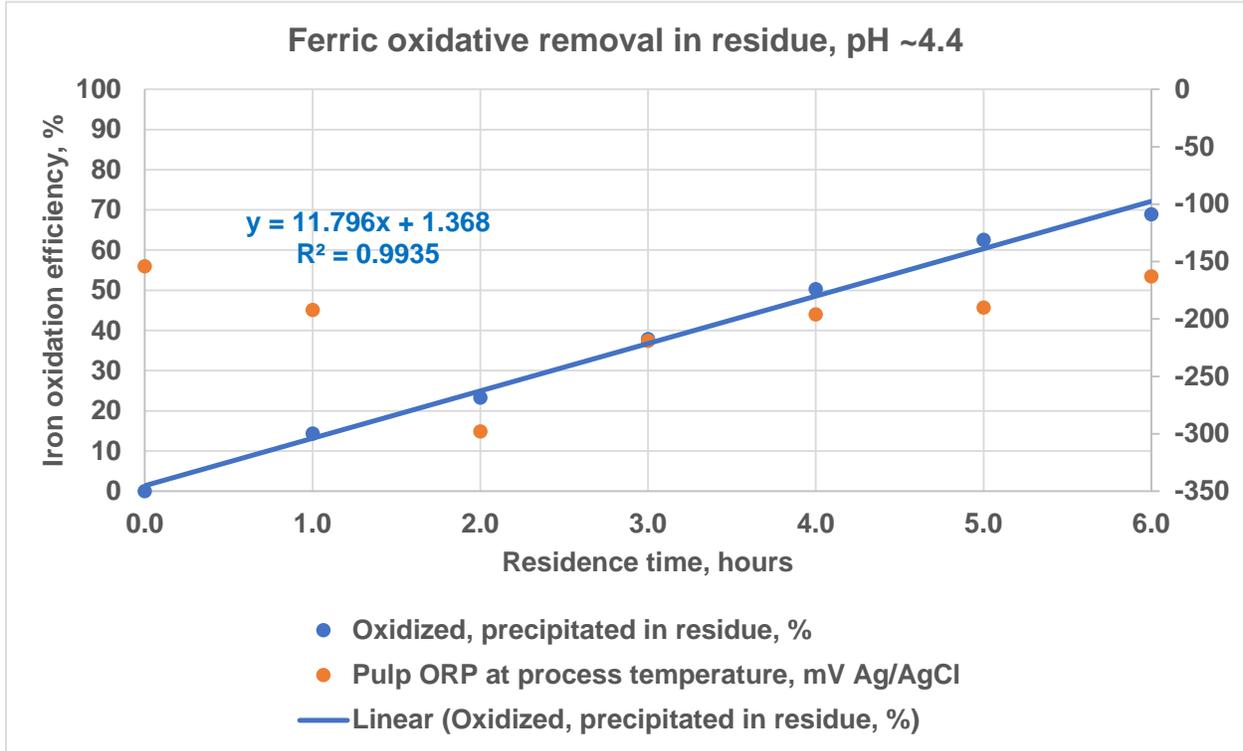


Table 16 Lixiviant regeneration precipitation losses

Precipitation (%)	FeOx1	FeOx2	FeOx3	FeOx4
Bi	0.3	0.1	9	No precipitation occurred
Co	0.2	0.0	1	
Cu	0.2	0.0	2	
Fe	2.1	0.7	17	
Mg	0.2	0.1	1	
Al	0.6	0.4	2	
Cr	1.2	0.7	21	
Mn	0.1	0.0	1	
Ca	0.2	0.1	1	
Na	0.4	0.0	0	
As	5.1	1.0	66	
Ag	5.1	2.4	34	
Pb	1.0	0.0	0	
Sb	17.6	1.1	13	
Sn	5.1	1.0	17	

Tests FeOx1 and FeOx2 with hydrogen peroxide - discarded. Test FeOx3 - with chlorine selected for design criteria.

Table 17 Lixiviant regeneration final residues

Final Solids (g/t)	FeOx1	FeOx2	FeOx3	FeOx4
Bi	42.0	85.0	20	No solids formed
Co	4.0	5.0	6	
Cu	13.0	5.0	5	
Fe	307,000.0	500,000.0	512000	
Mg	40.0	157.0	40	
Al	229.0	842.0	228	
Cr	17.0	68.0	97	
Mn	5.0	7.0	17	
Ca	100.0	221.0	131	
Na	32,100.0	493.0	1970	
As	30.0	31.0	285	
S	0.0	0.0	0	
S=	0.0	0.0	0	
S0	0.0	0.0	0	
Au	0.0	0.0	0	
Ag	2.0	2.0	2	
Pb	245.0	30.0	30	
Sb	40.0	23.0	30	
Sn	20.0	20.0	20	

Tests FeOx1 and FeOx2 with hydrogen peroxide - discarded. Test FeOx3 - with chlorine selected for design criteria.

Table 18 Lixiviant regeneration final solutions

Final Solutions (mg/L)	FeOx1	FeOx2	FeOx3	FeOx4
Bi	70.0	58.0	4	4.0
Co	13.5	12.1	17	17.8
Cu	38.9	32.8	5	4.8
Fe	61,600.0	54,300.0	52500	64100
Mg	139.0	120.0	133	137
Al	215.0	185.0	202	208
Cr	7.5	8.9	8	9.8
Mn	38.8	34.7	38	39.7
Ca	257.0	219.0	254	265
Na	47,100.0	42,600.0	46400	46500
As	3.0	3.0	3.0	7.0
S	0.0	0.0	0.00	0.00
S=	0.0	0.0	0.00	0.00
S0	0.0	0.0	0.00	0.00
Au	0.0	0.0	0.00	0.00
Ag	0.2	0.1	0.08	0.30
Pb	124.0	97.0	125	132
Sb	1.0	2.0	4	5
Sn	2.0	2.0	2	2
Fe(II)	44,700.0	1,320.0	28500	13

Tests FeOx1 and FeOx2 with hydrogen peroxide - discarded. Test FeOx3 - with chlorine selected for design criteria.

Bismuth leaching conclusions

Bismuth leaching extraction after liquid-solid separation and washing of 98.71 % based on the leach-only (99.5%) and wash only (99.21%) recoveries and PLS and wash vs. feed recovery. The washing maximizes the chloride removal from the washed cake to about 38 mg/L final Cl- tenor and the bismuth tenor in the combined PLS and wash (45,345 mg/L Bi tenor).

Cementation updated conditions produced 99.99% cementation efficiency vs. solids from the NICO feed, leaving from 3 to 6 mg/L Bi in the barren solution. The NICO cements consistently assayed above 99% Bi by direct assays and above 98% by difference vs. impurities. Most notably, both cement products contained less than 0.6 % Fe. Unoptimized conditions applied to the NICO/Rio Tinto feed blend produced 99.3% leaching efficiency (before liquid-solid separation and washing) and 99.99% cementation efficiencies, producing a cement assaying 91.5% Bi with about 3.2% Fe content.

Lixiviant regeneration efficiency using chlorine was about 95%, over 3 hours addition time, requiring 0.84 ton chlorine per ton of iron oxidized dosed at 125% stoichiometry excess. This regeneration option was selected.

Direct discharge of the bleed stream being permitted, regardless of its iron content, negating the need for the iron removal by oxidation. In this case, no oxygen is needed, and the lime would be used for pH adjustment only to ~4, bringing the specific consumption to 0.08 t/t lime as Ca(OH)₂ pure equivalent per ton of iron discarded.

PHASE 6A AND 6 B RESULTS

For manageable content streamlining, detailed results are provided for both Phases 6A and 6B in this section.

Fortune retained XPS Industry Relevant Solutions (XPS) to complete the bismuth smelting and refining test work for the pyrometallurgical circuits in the proposed Alberta refinery.

Phase 6A Initial smelting trials at XPS successfully converted the bismuth cement generated from hydrometallurgy at SGS into a crude bismuth product with a verified purity of 99%. The refining tests were carried out during the Milestone 3 work and validated the production of high-purity ≥99.99% “4N” bismuth metal ingots.

Phase 6B Results

Bismuth cement samples produced at SGS were successfully smelted to generate a crude ingot assaying approximately 99.8% bismuth (“2N”), which was then refined through sequential liquation, sulphuration, and chlorination steps. These refining stages achieved the target 99.99% (“4N”) bismuth grade, confirming the capability of the proposed flowsheet to produce high-purity bismuth ingots suitable for metal applications as well as for the manufacture of bismuth oxide and other chemicals. The pyrometallurgical investigation has reached completion, with operating conditions finalized and the issuance of a final XPS report containing the PDC required for incorporation into the Feasibility Study for the Alberta bismuth circuit.

Bismuth smelting and refining summary

Testing started with a 2-level design of experiments (high-low) that focused on the flux (Na₂CO₃) and reductant (Charcoal) dosages. Due to the presence of oxidized Bi species, it was decided that using increased reductant levels would have a more significant impact than flux ratio. A second set of tests was conducted introducing a midpoint between the high and low values for both flux and reductant, as well as the introduction of a second flux (Na₂SO₄) to help with slag fluidity. All the smelting tests were conducted in a muffle furnace with a ramp rate of 5°C/min. Setpoint was at 950°C with a 2 hour hold time for small scale (50g) tests and 2.5 hour hold time for larger scale. After the hold time, the furnace was allowed to cool overnight, and the samples were retrieved the next day for weighing and inspection. After inspection and weight of metal and slag was determined, selected samples for each run were sent for chemical analysis. MgO crucibles were used for all the smelting tests. Gas analysis measurements were also taken during the validation (250g x 2) and bulk testing (855g) to monitor gas evolution during smelting.

The four test conditions were narrowed down to three, as shown in Table 19 below. The best recipe was replicated in a slightly bigger scale (two tests at 250g feed) and once results showed good repeatability, a bulk test to convert all the Bi cement to metal (855g feed) was conducted.

Table 19 Design of Experiments

Date of Test	Test ID	Temp (°C)	Feed ID	Feed	Flux		Reductant		
					Na ₂ CO ₃	Na ₂ SO ₄	Activated Charcoal	Na ₂ CO ₃	Activated Charcoal
2025-02-03	1-A	950	BIME-10 PRODUCT 0003	50.00 g	25%		8.0%	12.50 g	4.00 g
2025-02-03	1-B	950	BIME-10 PRODUCT 0003	50.00 g	15%		8.0%	7.50 g	4.00 g
2025-02-03	1-C	950	BIME-10 PRODUCT 0003	50.00 g	15%		5.0%	7.50 g	2.50 g
2025-02-05	2-A	950	BIME-10 PRODUCT 0003	50.00 g	20%		6.5%	10.00 g	3.25 g
2025-02-05	2-B	950	BIME-10 PRODUCT 0003	50.00 g	18%	2%	6.5%	9.00 g	3.25 g
2025-02-05	2-C	950	BIME-10 PRODUCT 0003	50.00 g	20%		5.0%	10.00 g	2.50 g
2025-02-18	3-A1	950	BIME-10 PRODUCT 0003	250.00 g	25%		8.0%	62.50 g	20.00 g
2025-02-18	3-A2	950	BIME-10 PRODUCT 0003	250.00 g	25%		8.0%	62.50 g	20.00 g
2025-02-20	4	950	BIME-10 PRODUCT 0003	855.50 g	25%		8.0%	213.88 g	68.44 g

Smelting Results - First Cement Sample

Table 20 provides a summary of the weights of additives, initial sample and final products from the smelting results of the initial Bi cement sample.

Table 7 presents the results on the three tests sent for chemical analysis. Samples of the metal product were sent to two different labs for assay validation. Both samples were assayed by ICP-OES but there are some differences in the reported elements based on the analytical lab and their established methods. The Bi metal product of the bulk test 4 (855g feed) was not assayed given the same recipe as 3-A1 and 3-A2.

Only one lab was able to analyze the Bi at this concentration range, and this measurement was done by AAS. The second lab offered to do Bi purity by titration, but it would take them a few weeks to set up, and the analysis will be at a considerable cost.

Bi purity was calculated by impurity assay difference for all metal samples with Bi purity represented by the sum of the impurity concentrations (in ppm or %) subtracted from 100%.

Table 20: Initial Bi Cement Smelting Results

Date of Test	Test	Dry Bi cement sample (g)	Bi content (g)	Activated Charcoal (g)	Na ₂ CO ₃ (g)	Na ₂ SO ₄ (g)	Crucible Empty Weight (g)	Weight of Metal (g)	Weight of Matte + Slag (g)	Over-all %wt. loss	%wt.slag over charge	%wt.metal over charge	%wt.metal over feed	% wt.metal over Bi in feed
2025-02-03	1-A	50.0	36.3	4.02	12.51	0	155.66	37.08	14.59	22.4%	21.9%	55.7%	74.1%	102.2%
2025-02-03	1-B	50.0	36.3	4.01	7.51	0	157.76	37.65	9.16	23.9%	14.9%	61.2%	75.3%	103.8%
2025-02-03	1-C	50.0	36.3	2.51	7.52	0	178.96	35.65	12.51	19.8%	20.8%	59.4%	71.3%	98.3%
2025-02-05	2-A	50.0	36.3	3.25	10	0	157.9	36.1	13.8	21.1%	21.8%	57.1%	72.2%	99.6%
2025-02-05	2-B	50.0	36.3	3.25	9	1	155.1	36.1	13.6	21.4%	21.5%	57.1%	72.2%	99.6%
2025-02-05	2-C	50.0	36.3	2.5	10	0	147.8	36	14.9	18.6%	23.8%	57.6%	72.0%	99.3%
2025-02-18	3-A1	250.0	181.3	20	62.5	0	364.6	186.4	72.4	22.2%	21.8%	56.1%	74.6%	102.8%
2025-02-18	3-A2	250.0	181.3	20	62.5	0	313.7	176.3	83	22.0%	25.0%	53.0%	70.5%	97.3%
2025-02-20	4	855.5	620.2	68.4	213.9	0	865.2	643.7	250.1	21.4%	22.0%	56.6%	75.2%	103.8%

The metallurgical balance from the large-scale Smelting Test 3 is presented below. After the metal and slag were both assayed, a calculated head assay was established which is slightly higher than the Bi assay value measured by AAS. The calculated head assay is particularly useful in this analysis because the smelting in the lab was done in a closed system, and any fuming of Bi compounds is minimal.

Table 21 Metal Balance calculation for Smelting Test 3 using measured and calculated head assay

Reductive Smelting Stage	Feed	Metal (Product)	Slag (By-product)
Dry Weight, g	500	362,7	155.4
% Bi Assay (measured by AAS)	72.5%	98.6%	8.87%
Contained Bi, g	362.5	357.6	13.8
Metal balance based on measured head assay	102.5%	98.7%	3.8%
Calculated Bi contained in head material, g (sum of metal and slag)	371.4	357.6	13.8
Calculated % Bi assay for head material	74.3%	98.6%	8.87%
Metal balance based on calculated head assay	100%	96.3%	3.7%

Bi Cement Smelting – Second Bi Cement Samples

A second set of Bi cement samples were provided by the client (BiC11 and BiC12). The second Bi cement samples did not undergo purification via hydromet leaching but was produced under optimized cementation conditions. The cement went under smelting and refining process development, following the best recipe determined from work on the initial Bi cement.

Figure 7 Second Bi Cement Samples BIME-10 Sample 0002 (BiC11) and Bime-10 Sample 0003 (BiC12)



Test Plan and Methodology

All the smelting tests except 7-A & 7-B were conducted in a muffle furnace with a ramp rate of 5°C/min. Setpoint was at 950°C with a 2.5 hour hold time. After 2.5 hrs, the furnace was allowed to cool overnight, and the samples were retrieved the next day for weighing and inspection. MgO crucibles were used for all the smelting tests. 7-A & 7-B were direct melting tests and used the 10kW induction furnace set up.

Direct melting tests were performed as the samples looked more metallic than the purified Bi cement sample. Table 22 presents the recipes used for the second Bi Cement samples.

Table 22 Test Plan for Second Bi Cement Samples

Date of Test	Test ID	Temp (°C)	Feed ID	Feed	Na2CO3	Na2SO4	NaOH	Activated Charcoa	Flux	Reductant	
New cement samples (individual)											
2025-03-06	5-A	950	BIME-10 SAMPLE 0002	50.00 g	25%				8.0%	12.50 g	4.00 g
2025-03-06	5-B	950	BIME-10 SAMPLE 0003	50.00 g	25%				8.0%	12.50 g	4.00 g
2025-03-17	6-A	950	BIME-10 SAMPLE 0002	50.00 g	25%				8.0%	12.50 g	4.00 g
2025-03-17	6-B	950	BIME-10 SAMPLE 0003	50.00 g	25%				8.0%	12.50 g	4.00 g
2025-03-17	6-C	950	BIME-10 SAMPLE 0002	50.00 g	15%				5.0%	7.50 g	2.50 g
2025-03-17	6-D	950	BIME-10 SAMPLE 0003	50.00 g	15%				5.0%	7.50 g	2.50 g
2025-03-18	7-A	950	BIME-10 SAMPLE 0002	25.00 g			10%			2.50 g	
2025-03-18	7-B	950	BIME-10 SAMPLE 0003	25.00 g			10%			2.50 g	
New cement samples - blend											
2025-04-09	8-A	950	50:50 blend	50.00 g	15%				4.0%	7.50 g	2.00 g
2025-04-09	8-B	950	50:50 blend	50.00 g	13.5%	1.5%			4.0%	7.50 g	2.00 g
2025-04-25	9	950	50:50 blend	816.00 g	15%				4.0%	122.40 g	32.64 g

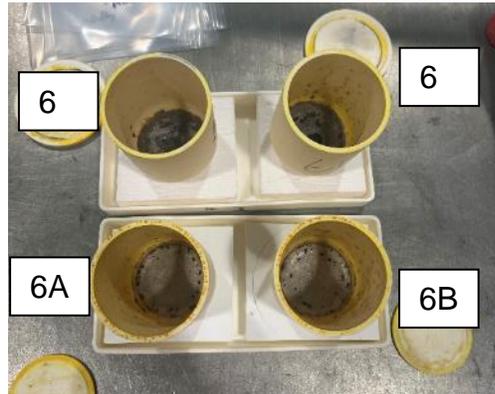
The validation tests of the developed recipe, Test 5A and 5B were declared failed tests due to the crucible for Test 5B breaking inside the furnace during the test. There was total loss of sample for Test 5B, while the crucible for Test 5A was fused to the catch tray and would not have been possible to get a reliable weight of the material post-smelt.

Figure 8 Test 5A and Test 5B (cracked) post-smelting were declared failed tests



New tests were set-up, Test 6A and 6B, using the same parameters. Two additional tests were also conducted following observations from Test 5A where an excess of flux and reductant was observed inside the crucible. Tests 6C and 6D were set up to have low reductant and low flux.

Figure 9 Test 6A to 6D post smelting



Test 7A and Test 7B were direct melting tests and used the 10kW induction furnace set up, as used in re-melting Bi metal and in the refining process development. 25g of sample was used in each test along with NaOH as flux. The samples were heated to 500°C, stirred manually and held at temperature for 15 minutes and then cooled to room temperature before unmolding.

Figure 10 Direct melting of second cement in 10kW induction furnace, with NaOH



Test 8A and 8B were set up to compare between two different fluxes (Na_2CO_3 only vs Na_2CO_3 - Na_2SO_4 90:10 blend) due to the low S content of the cement feed. The Na_2SO_4 is intended to produce a sulphide matte layer that will help improve the purity of the Bi metal produced. The Bi cement feed used in these tests are a 50:50 blend of the two cement samples submitted (BiC11 and BiC12). Total flux used in these tests are the same as in Test 6C and 6D (Low flux, 15%) but the reductant ratio was further reduced to 4%.

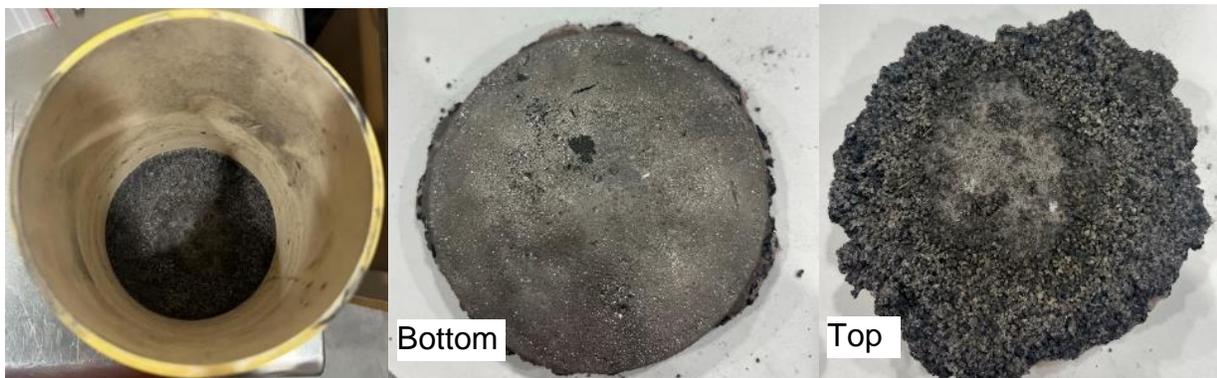
Both samples showed good recovery, and assay of the metal. From comparison of the physical aspects of the melt, Test 8A slag was lighter in colour (light grey) compared to Test 8B slag (dark grey). Test 8B slag also had a faint sulphide smell, which is consistent with the expectation that the Na_2SO_4 added in the flux recipe will produce a sulphide matte layer that will collect some of the impurities that would otherwise deport to the metal.

Figure 11 Setup and output of Smelting Tests 8A and 8B



Test 9 was a bulk test to validate 8A recipe (15% Na_2CO_3 , 4% charcoal) and produce metal for the refining stages. Figure 12 shows the smelting output prior to separating the slag from metal.

Figure 12 Test 9 Smelting Output



Results and Discussion

The second Bi samples were sent for analysis at two external labs, one of which that could provide Bi assays for high concentrates. The client had also shared their chemical assays on the material.

Table 23 presents the average chemical analysis from the two external labs and the client results and the average moisture content measured at XPS.

Table 23 Average Chemical and Moisture Analysis of BiC11 and BiC12

Analyte	ICP-OES Unit	BiC11 Average	BiC12 Average
		BIME-10 Sample 0002	BIME-10 Sample 0003
Ag	%	0.0024	0.0027
Al	%	0.0134	0.0135
As	%	0.2073	0.2167
B	%	0.0010	0.0010
Ba	%	0.0005	0.0005
Be	%	0.0000	0.0000
Bi	%	96.2500	96.0500
Ca	%	0.0100	0.0100
Cd	%	0.0006	0.0006
Ce	%		
Cl	%	0.1650	0.3200
Co	%	0.0069	0.0068
Cr	%	0.0137	0.0137
Cu	%	0.5467	0.5393
Fe	%	0.5357	0.5697
Ga	%	0.0010	0.0010
Hg	%	0.0001	0.0001
K	%	0.0047	0.0047
La	%	0.0010	0.0010
Li	%	0.0010	0.0010
Mg	%	0.0060	0.0060
Mn	%	0.0047	0.0044
Mo	%	0.0005	0.0005
Na	%	0.0101	0.0158
Ni	%	0.0243	0.0302
P	%	0.0040	0.0041
Pb	%	0.0285	0.0247
S	%	0.0350	0.0250
Sb	%	0.4050	0.3950
Sc	%	0.0001	0.0001
Se	%	0.0030	0.0030
Si	%		
Sn	%	0.0020	0.0020
Sr	%	0.0001	0.0001
Te	%	0.0136	0.0042
Th	%	0.0020	0.0020
Ti	%	0.0037	0.0037
Tl	%	0.0047	0.0045
U	%	0.0001	0.0001
V	%	0.0009	0.0009
W	%	0.0001	0.0001
Y	%	0.0001	0.0001
Zn	%	0.0029	0.0030
Zr	%	0.0001	0.0001
Total Assay		98.3124	98.2816
Total Impurities		2.0624	2.2316
% Bi by difference		97.9376	97.7684
Moisture	%	1.7	2.3

Bi Refining – Initial Bi Cement

Crude Bi metal was produced successfully from the smelting of Bi cement. Certain impurities reported to the matte and slag during the smelting reaction, but the metal produced still needed to undergo refining to reach 3N to 4N5 purity. These impurity elements can be removed selectively and sequentially in the different refining stages described in Figure 1 below. The metal from test 4 was re-melted along with the other crude Bi metal products from the small-scale tests to form the metal bath for Bi refining.

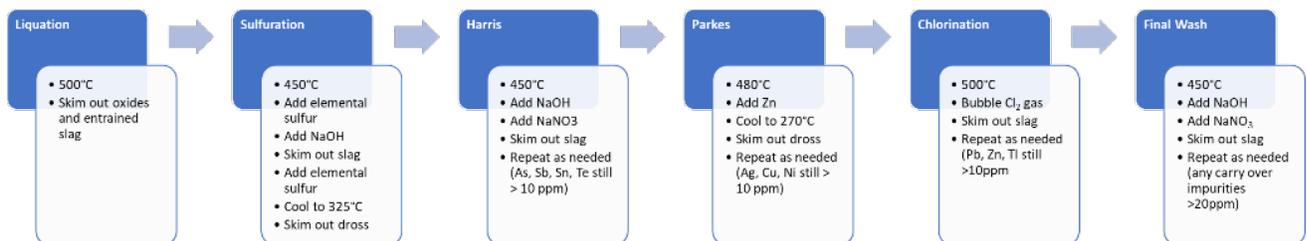
Figure 13 Bi refining process flow



Test Plan and Methodology

A refining kettle was simulated in the laboratory with a 10kW induction furnace, an overhead agitator and an alumina crucible. Temperature of the molten bath is controlled via the manual switching of the induction furnace and monitored by a sheathed thermocouple. A dedicated exhaust vent is provided to capture any fumes that may be released during the reactions. Details of target process conditions are described in Figure 14.

Figure 14 Bi Refining Process Conditions



For each of the refining stages, impurity removal was affected by either temperature control and/or chemical reaction with a reagent. Slag or dross are produced from the reaction which contains the impurity elements and are then skimmed and collected separately for characterization and metal

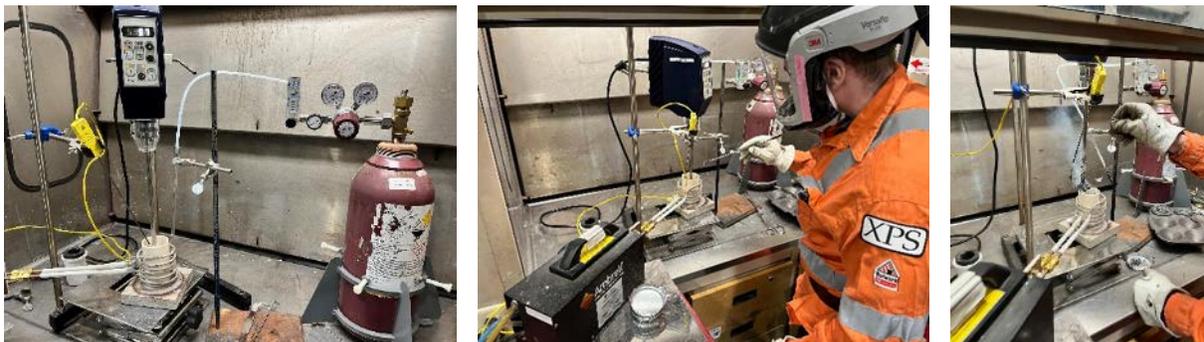
accounting. At the end of each refining stage, a sample of the molten metal was taken and sent for chemical analysis to confirm level of impurities

Figure 15 Skimming of Slag Floating on Top of the Molten Bismuth



For the planned chlorination stage and subsequent final caustic wash, the 10kW induction furnace was relocated inside a fume hood to help mitigate risks associated with chlorine usage.

Figure 16 Chlorination refining set-up inside fume hood



Results and Discussion

Chemical Analysis

Following the metal sample from smelting Test 3, the analytical lab reported all metal samples would require specialized sample preparation and analysis at significantly higher cost. XPS had proposed to the client (Fortune) to do in-house sample preparation of all the metal samples collected during the multiple steps of the refining process via acid digestion, for submittal to the analytical lab as liquid samples. Client agreed to this arrangement during the meeting and in writing. Details of the analytical process development and validation completed by XPS can be found Appendix 4.

Chemical Analysis

Most of the impurity elements in the metal produced from smelting are removed at the first step (liquation). The specific elements tracked and selectively removed in the subsequent steps are outlined in Table 24 below. Zn is not originally present in the Bi metal from smelting but is added as a reagent in Parkes and which will need to be removed in the next stage, Chlorination.

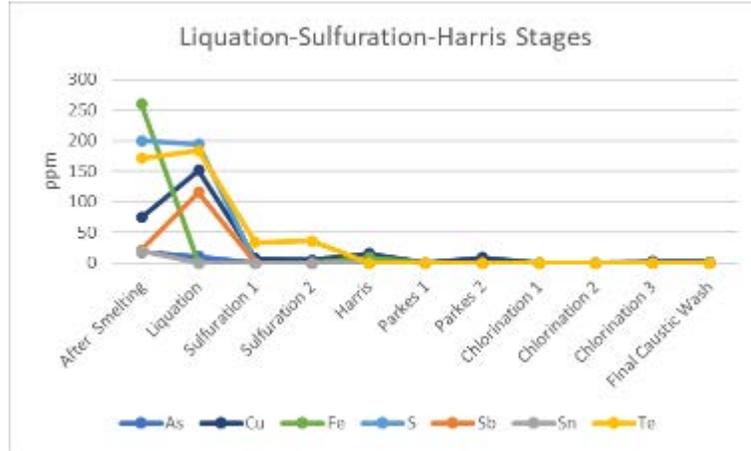
Table 24 Selective removal of impurity elements during refining

Refining Stage	Target elements for removal	Target concentration to reach 4N purity
Liquation	Entrained slag and other high melting point (>500°C) metals and compounds	Clean metal
Sulfuration	Cd, Cu	<10 ppm for each element, and <100 ppm in sum total of impurities
Harris	As, Cr, Fe, S, Sb, Sn, Te	
Parkes	Ag, Au, Cu, Ni	
Chlorination	Pb, Zn, Tl (tbd)	
Final Wash	Cl and any corrosion-related contamination (Cr, Fe)	

The specified elements are divided into three sub-groups based on where these elements will be completely removed or reduced in the 6 refining stages.

With the exception of Cu, the rest of the elements reported in the Liquation-Sulfuration-Harris group are effectively removed in this multi-stage grouping.

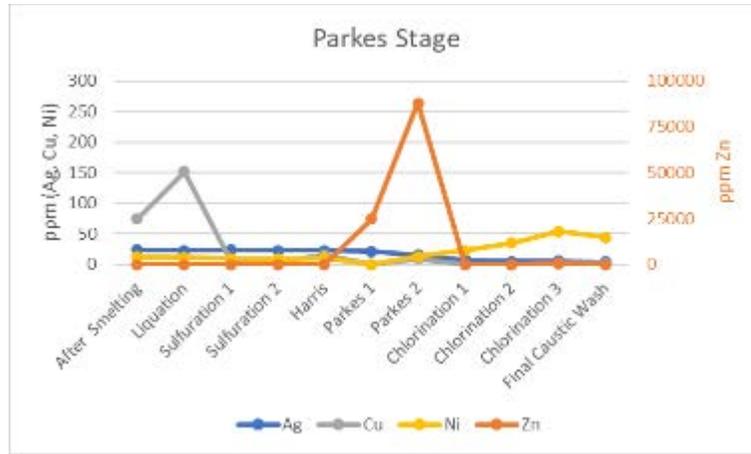
Figure 17 Impurity concentration levels for target elements at the Liquation-Sulfuration-Harris refining stages



Cu is further reduced in concentration in the next stage, Parkes, along with Ag and other precious metals if present, and Ni. Zn metal is added to the bath which will form intermetallic compounds with the said elements and form a crust that will float on top of the Bi bath that can be skimmed out. However, at this eutectic point, ~3% of Zn will also dissolve into the molten Bi. The increase in Zn concentration is evidenced in Figure 18 below.

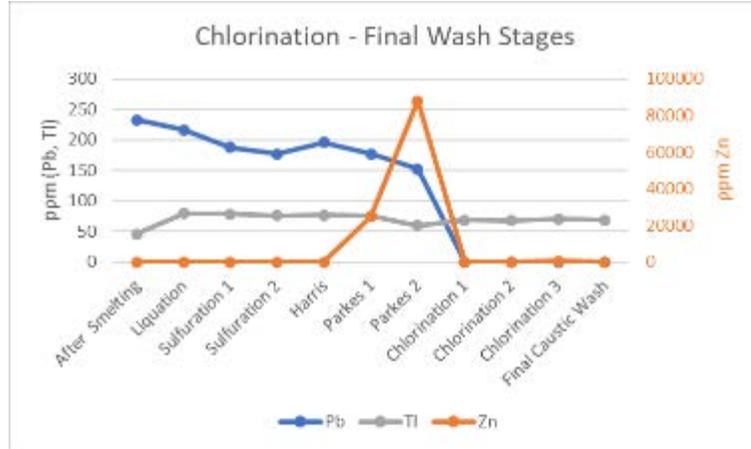
Ag, Cu and Ni concentrations in the metal were all effectively reduced after the Parkes process, but assays showed Ni levels started to increase again during chlorination. This will be discussed at a later section (Test Observations).

Figure 18 Impurity concentration levels for target elements at the Parkes refining stage



As shown in Figure 19 below, both Zn and Pb were successfully removed by reacting with chlorine gas. The Thallium concentration however remained unchanged.

Figure 19 Impurity concentration levels for target elements at the Chlorination and Final Wash refining stages

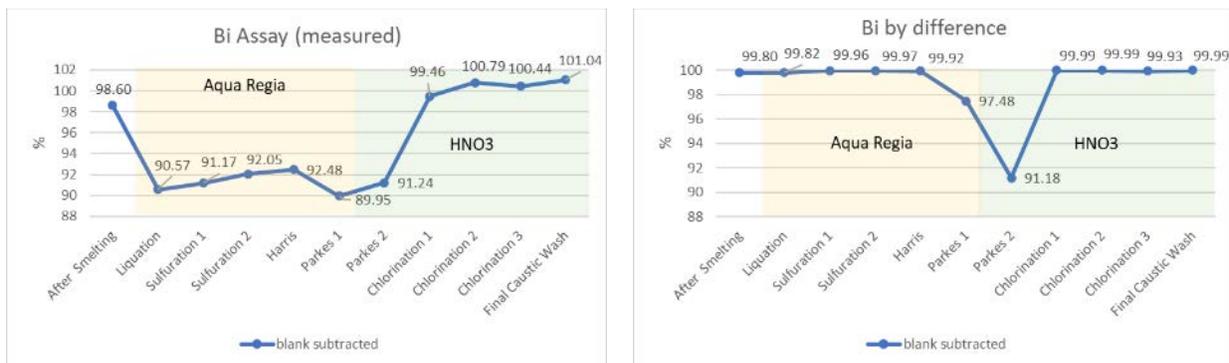


Based on literature and previous plant experience, Tl can be removed with $ZnCl_2$. For the refining development, it was assumed that any $ZnCl_2$ formed during chlorination would then react with the Tl in the melt and produce $TlCl_3$ which can then be removed along with the Pb and Zn chlorides. Additional Cl gas will be required as Zn will dissolve back into the Bi melt. In this test, chlorine dosing was tracked over time and metal samples were taken at three points in time (at stoichiometric dose, at +25% excess, at +50% excess). For all three samples, Tl remained at the same concentration.

There was an increase in Zn assay for the third chlorine sample (50% excess Cl dosing) which may indicate the reaction as described above starting, but since overdosing with Cl will lead to Bi losses from BiCl₃ formation and fuming off, it was not in the plan to go beyond 50% excess.

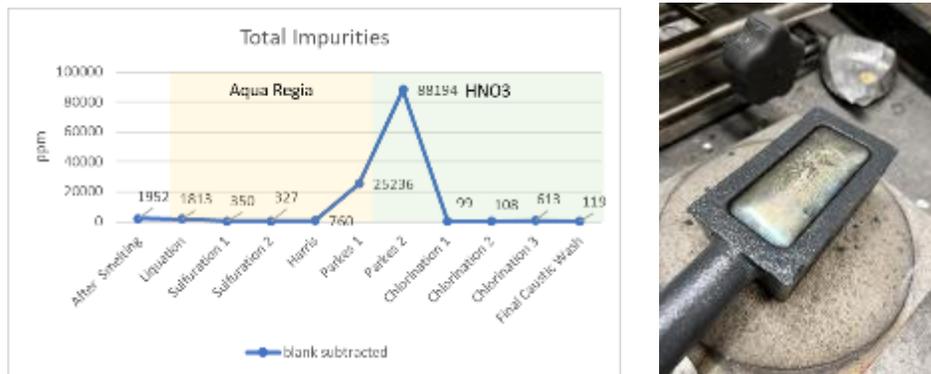
XPS reports the assay of the Bi metal over the course of the whole refining process. The sample IDs are indicated on the 1st row while the 2nd row details the method of assay preparation. The first half of samples were prepared by aqua regia as discussed in Appendix 4, while the second half of samples which were generated at a later date, were prepared with HNO₃. This was to avoid the issues encountered with high solute concentrations in the samples submitted. The HNO₃ prepped samples came back with Bi assays way below expected range and were sent back to the lab for repeat analysis. The updated assays are presented in Figure 24 below, showing Bi assays very close to 100%.

Figure 20 Comparison of Bi assay (measured vs calculated by difference) through the different refining stages



By tracking the impurities in the metal, we can see the decreasing level over the course of the refining process. Based on the assays, 4N purity was reached at the first chlorination sample (stoichiometric Cl dose).

Figure 21 Sum of impurities in the Bi metal through the different refining stages and photo of the cast ingot after final wash



Bi Refining – Second Bi Cement Samples

Crude Bi metal produced from smelting tests of the second set of Bi Cement samples were subjected to the same refining stages as the initial Bi Cement sample.

Test Plan and Methodology

A refining kettle was simulated in the laboratory with a 10kW induction furnace, an overhead agitator and an alumina crucible. Temperature of the molten bath is controlled via the manual switching of the induction furnace and monitored by a sheathed thermocouple. An addition of temperature datalogger was introduced to capture the temperature of the molten bath through the various stages. A dedicated exhaust vent is provided to capture any fumes that may be released during the reactions. Process conditions for each stage are shown again in Figure 22.

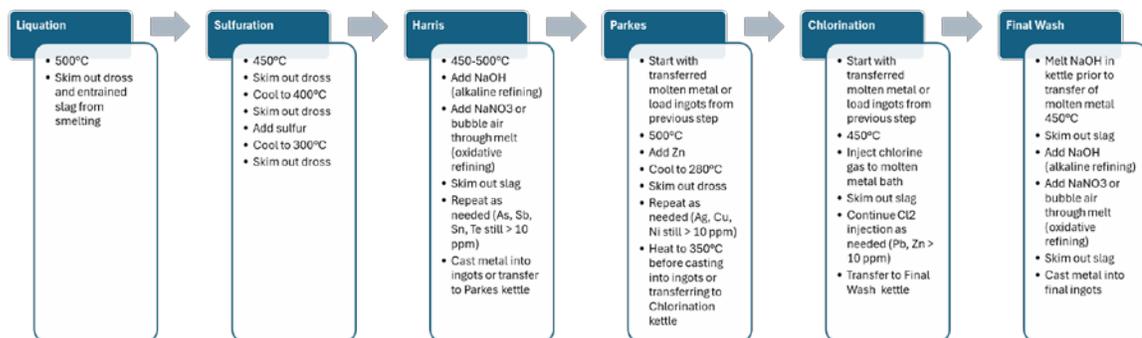
For liquation, the metal recovered from Smelting Test 9 was melted first and after removing the dross, sampled for analysis. This sample aims to review the quality of the metal from smelting, before and after liquation. The rest of the metal samples from previous small-scale tests (Tests 6A to 7B) were then added to the melt and the liquation step was repeated. The sample taken after the 2nd liquation drops were removed will be used as the metal assay feed to the next step.

Sulfuration, Parkes and Chlorination steps were all set to be done in one step to compare with the first refining batch. Stoichiometric dosages for the zinc and chlorine gas were applied to the melt for impurity removal. Please refer to Table 9 for the different stages and the target elements for removal in each stage. Sampling was also limited to 5g samples to avoid losing metal volume during the process.

Part of the learnings from the first batch of refining was the high possibility of contamination with the small-scale refining. For the 2nd refining batch, the following adjustments were made:

Transfer molten metal to a new crucible for Parkes, Chlorination and Final Wash. Agitators and the ceramic sheath protecting the thermocouple will also be replaced with clean and new versions. Sampling spoons and stirrers will also be changed out for each crucible transfer.

Figure 22 Bi Refining Process Conditions



Results and Discussion

Chemical Analysis Preparation

All of the metal samples from the smelting tests were prepared in XPS by acid digestion prior to submittal and solution samples to analytical laboratory ICP- OES analysis. All samples, both metal and slags were prepared using aqua regia (3:1 HNO₃ to HCl). A blank sample was also prepared and analyzed with the group of samples for blank correction. A Bi metal sample was sourced commercially and prepared along with the test samples to serve as a check sample.

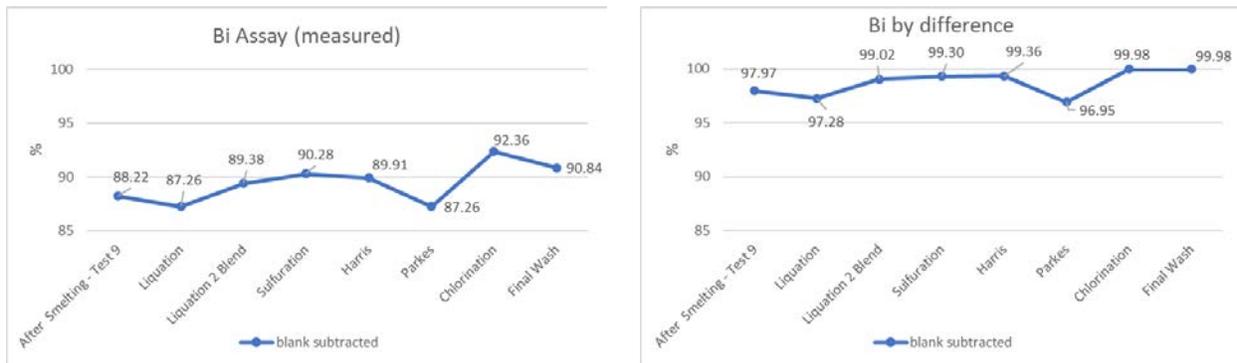
From the 5-gram metal samples taken during the refining, the small ingots were crushed manually with a hammer to get 0.5 g samples for digestion.

For the slag samples however, the entire sample was digested to avoid sampling errors with respect to Bi entrainment. Volume to weight ratios for the slag samples were kept between 10-20x to avoid high dissolved solids content in the digested solutions that could lead to precipitation.

Chemical Analysis

The assays for the metal samples showed similar response to previous refining batch samples that were prepared by aqua regia digestion – measured Bi assays are slightly lower than expectation. Bi by difference, however, shows Bi close to 100%.

Figure 23 Bi assays for 2nd refining test (measured vs by assay difference)



The Bi assay trend showed the progression from 98% Bi metal produced from smelting to 99.98% Bi after refining. The Bi metal assay from smelting (97.97%) is lower than what was measured from the smaller scale tests (Test 8A/8B were at 98.7 to 99.1% Bi). Additional tests are recommended to further optimize the recipe for the new cement. The impurity content of the Bi metal seems to have increased after the first liquation step, as seen in Figure 24 below. These may be due to contamination in the metal sample, possibly from slag, as the elements that contributed to the higher total assay of impurities were As, Fe and Na as presented in Figure 44. The numbers decreased again by the second liquation sample. The final wash removed relatively insignificant impurities after chlorination.

Figure 24 Sum of impurities in the Bi metal through the different refining stages

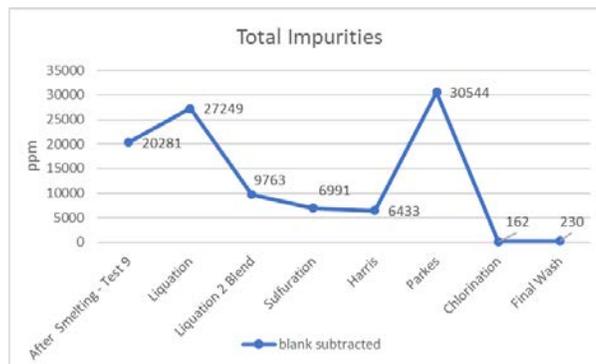
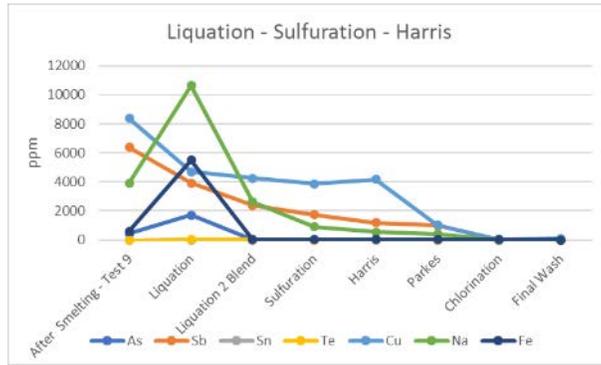
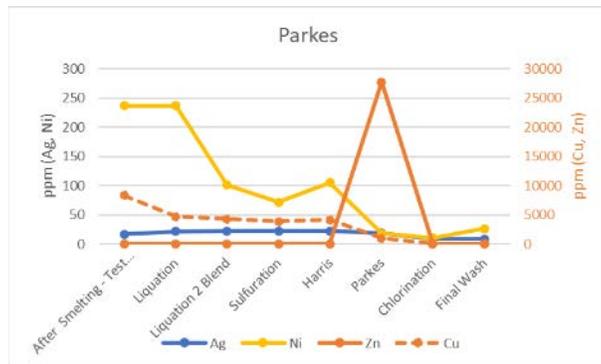


Figure 25 Impurity concentration levels for target elements at the Liquation-Sulfuration-Harris refining stages



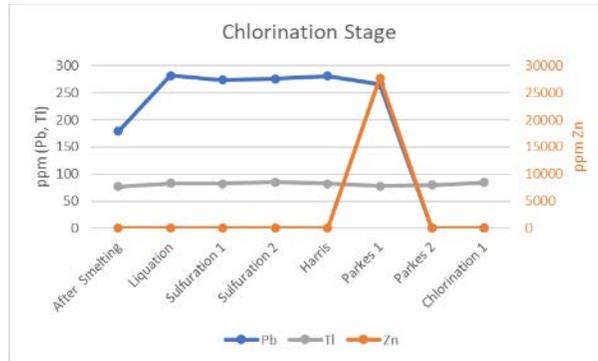
Cu was not effectively decreased during sulfuration, likely a result of under-dosing of sulfur. Doing the two-step sulfuration process may be required to ensure Cu is brought down considerably before Parkes. Sb was much higher in the metal than was expected, so it was not sufficiently removed during Harris. The Sb that slipped through was removed at the end with the final caustic wash. As, Na and Fe were reduced to target levels as the refining process progressed.

Figure 26 Impurity concentrations for target elements at the Parkes stage



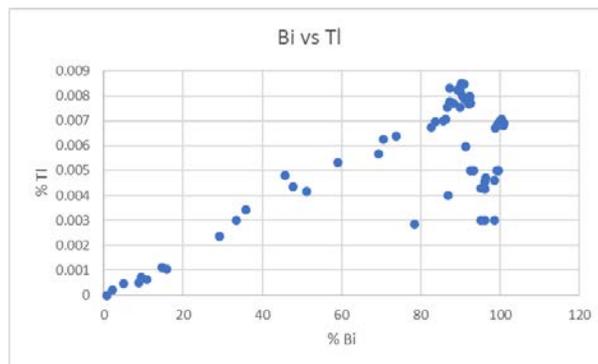
All the elements behaved as expected during the Parkes process where Ag, Cu, and Ni were brought down to low levels. However, both Cu and Ni show a slight uptick in concentration after the final wash, despite our precautions for cross-contamination. Zn was present in stoichiometric ratio to the Bi.

Figure 27 Impurity concentrations for target elements at the Chlorination stage



Pb and Zn were sufficiently removed during chlorination. Thallium, however, stayed almost constant throughout the multiple refining steps. Thallium assay verification is recommended to ascertain the validity of the reported assays for this element. If true, additional testing will be required to remove the Thallium from the Bi metal. Bi and Tl assays for all the samples assayed in this test program are plotted in Figure 28 below as a preliminary start to investigating the phenomenon.

Figure 28 Plotting Bi and Tl assays to show possible interference effect



Metallurgical Balance and Recovery

Over-all Bi recovery for the 2nd refining (metal from new cement) is at 57%, slightly lower than the first batch (60%). The final assay for the Bi metal is at 99.98% versus the first batch attaining 99.99%. These are presented in Figure 29.

Both of these can be attributed to higher impurities in the incoming metal that needed to be removed. There was more slag generated, and therefore increased chance for Bi losses due to entrainment. These are shown in Table 25.

Figure 29 Bi Purity vs Recovery through the whole refining process

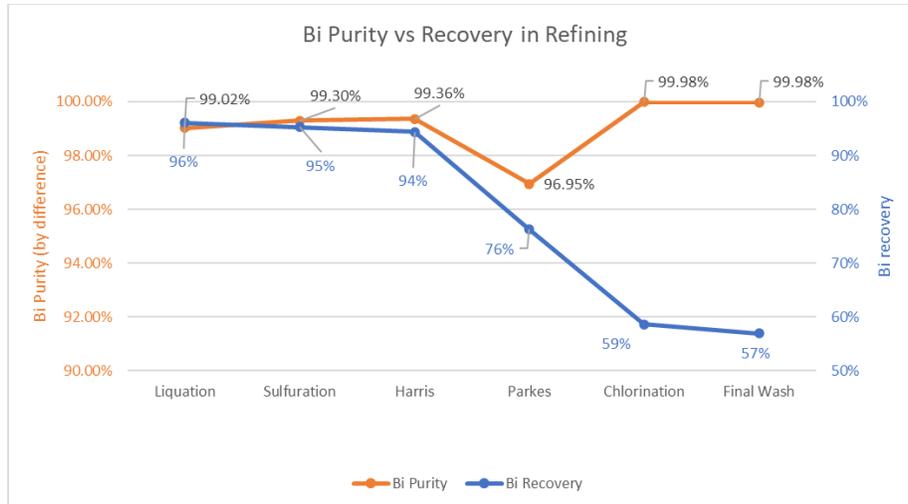


Table 25 Slag and Dross Generated (Comparison of Refining Batch 1 and 2)

Stage	Slag/Dross from Refining Batch 1			Slag/Dross from Refining Batch 2		
	Weight, g	Bi Assay (average)	Contained Bi, g	Weight, g	Bi Assay (average)	Contained Bi, g
Liquation				44.58	39.86%	17.77
Sulfuration	39.13	2.10%	0.82	9.70	47.83%	4.64
Harris	15.20	5.07%	0.77	20.61	0.67%	0.14
Parkes	200.61	83.93%	168.37	222.65	83.98%	186.98
Chlorination	125.68	52.53%	66.02	167.78	74.06%	124.25
Final Wash	84.40	24.14%	20.37	58.83	18.02%	10.60
Total	464.92	55.14%	256.34	524.15	65.70%	344.37

Table 26 Over-all Bi Recovery in Refining

Stage	Feed			Metal Product			Metal Samples			Slag/Dross from Refining Batch 2			Bi Recovery Over-all
	Weight, g	Bi Assay (by difference)	Contained Bi, g	Weight, g	Bi Assay (by difference)	Contained Bi, g	Weight, g	Bi Assay (by difference)	Contained Bi, g	Weight, g	Bi Assay (actual)	Contained Bi, g	
Liquation	1129.11	99.97%	1128.79	1051.77	99.36%	1045.00	5.00	97.28%	4.86	26.30	35.79%	9.41	94%
5.04							99.02%	4.99	18.28	45.72%	8.36		
5.11							99.30%	5.07	9.70	47.83%	4.64		
Harris	1051.77	99.36%	1045.00	861.75	96.95%	835.43	5.23	99.36%	5.20	20.61	0.67%	0.14	76%
Parkes							5.13	96.95%	4.97	222.65	83.98%	186.98	
Chlorination	861.75	96.95%	835.43	631.28	99.98%	631.18	4.99	99.98%	4.99	81.16	69.37%	56.30	59%
Final Wash										631.28	99.98%	631.18	
										23.28	29.09%	6.77	57%
										35.55	10.76%	3.83	

The biggest drop in Bi recovery is still in the Parkes and Chlorination processes. A plan to incorporate these drosses back into the circuit to recover those Bi units need to be prepared as part of the flowsheet.

Bismuth smelting and refining conclusions and recommendations

XPS was able to successfully convert the two types of Bi cement provided by Fortune to Bi metal with >99.9% purity. A smelting recipe has been developed for the two cement samples, with some difference in the ratio of flux and reductant, depending on the composition of the feed. It is recommended to conduct further optimization on the smelting recipe especially if the material quality is highly variable or coming from different sources. A treatment and/or disposal plan for the smelting slag and baghouse dust also needs to be developed. XPS recommends pursuing treatment of these materials to recover the Bi units that may be lost from these by-products.

The metal produced from the smelting trials were successfully refined to produce ingots assaying from >99.98% to 99.99 % Bi metal. Recoveries decreased as the level of purification increased, ranging from 57% to 60% in the purified ingots versus the bismuth contained in the feed. These results are reflective of the artefact of the small-scale laboratory experiments. Industrial implementation of the process is expected to yield overall recoveries generally exceeding 90% after recycling of the intermediate streams. Further optimization is required to ensure impurity specifications are met and Bi losses are minimized. It is recommended to develop a plan for treating the slags and drosses generated from the refining stages in order to recover the Bi units, as well as other elements that may be of interest to other industries (e.g. Te, Sb, Ag).

PHASE 5B AND PHASE 7 RESULTS

For manageable content streamlining, detailed results are provided for both Phase 5B and Phase 7 in this section.

POX test work on cobalt concentrate blended with bismuth leach residue at SGS achieved cobalt recoveries of up to 97% and copper recoveries of up to 79%. Pre-neutralization of the POX discharge effectively removed iron and arsenic impurities, generating a PLS suitable for downstream copper and cobalt recovery and confirming the selected autoclave operating conditions for inclusion in the refinery design basis.

Gypsum precipitation test work demonstrated that sulphate removal and gypsum formation were technically successful; however, arsenic incorporation in the gypsum by-product could not be reduced to levels considered acceptable for the proposed commercial application in soil stabilization (10 ppm). Further work will be required to determine whether specific industries can accept gypsum containing elevated arsenic or, alternatively, to define arsenic criteria that would allow the gypsum residue to be classified as non-hazardous for disposal.

The 2025 primary neutralization test work showed that under optimized conditions at approximately pH 2.5, effective removal of 99% of the arsenic and 80–92% of the iron was achieved without oxygenation or heating, using either limestone or lime and a short residence time of 0.5 hours. Copper cementation test work confirmed that copper in the barren solution could be reduced to below 0.3 mg/L, meeting the project target for feed to the subsequent cobalt purification steps. Secondary oxidative neutralization tests were also completed, providing additional performance data on residual iron and arsenic removal for incorporation into the final design criteria.

The purified cobalt-bearing solution was then treated through manganese precipitation using a sulphur dioxide/oxygen gas mixture, followed by solvent extraction (SX) to remove trace metals including nickel, zinc, and copper. The resulting purified solution was subjected to evaporation and crystallization to produce a high-purity cobalt sulphate heptahydrate product suitable for battery-grade applications. All test work data, reagent optimization results, and operating parameters generated during Phase 5B have been compiled and used to finalize the hydrometallurgical process design criteria for the cobalt circuit in the Alberta Refinery design.

Gold recovery test work completed under Phase 7 confirmed robust gold performance across multiple NICO process streams and validated the assumptions carried forward from earlier phases. Monitoring of gold deportment during ferric chloride leaching of bismuth concentrate verified that gold remains in the leach residue and does not report to solution, supporting downstream cyanidation-based recovery strategies. CIL cyanidation tests on bismuth leach residues demonstrated that direct gold recovery from this stream is technically viable. Additional CIL tests on blended cobalt concentrate and bismuth leach residue from the POX discharge achieved gold recoveries of up to 98.5%, confirming that the primary gold circuit can meet or exceed design expectations. The results also showed that gold recovery from lime- and limestone-neutralized residues differed by less than 1%, indicating that separation of the POX discharge prior to neutralization is not required.

Redundancy testing to address potential autoclave downtime demonstrated that intensive cyanidation of the bismuth ferric leach residue can achieve up to 95% gold recovery within 48 hours, providing a viable contingency option for maintaining gold production during an unplanned autoclave shutdown. Complementary CIL testing on bulk scavenger flotation tails confirmed additional gold recovery potential from this stream, while ensuring that the resulting residues remained compatible with the planned residue management strategy.

Downstream residue characterization work has been finalized. TCLP tests on gold plant discharge residues produced under both lime- and limestone-neutralization conditions, as well as on POX residues, verified that the iron/arsenic precipitates meet criteria for safe disposal in a government-approved Class 2 landfill. TCLP testing of residues generated from CIL treatment of bulk scavenger flotation tails similarly confirmed their suitability for potential Class 2 landfill disposal. Liquid–solid separation tests on these residues were

successfully completed, providing design data for dewatering, handling, and long-term waste management. Acid-Base Accounting (ABA) testing was also completed on key residue streams and confirmed that the residues exhibit acceptable long-term geochemical stability, providing a sound basis for future environmental management planning.

Cyanide destruction optimization tests were concluded on selected leach solutions and tailings, establishing operating conditions capable of reducing cyanide concentrations to meet the project's detoxification and discharge criteria. All Phase 7 work has now been completed, and the results from the gold recovery, cyanide destruction, and residue stability programs have been consolidated into the final test reports used to support detailed engineering design and the updated feasibility study for the NICO Project Alberta Refinery.

Test work program objectives outline

The test work objectives consisted of:

1. Confirmation of the POX conditions and results.
2. Selective gypsum precipitation test work.
3. Primary non-oxidative neutralization test work.
4. Copper cementation test work.
5. Secondary oxidative neutralization test work.
6. Polishing test work.
7. Cobalt sulphate heptahydrate test work.
8. Gold recovery from the primary neutralization residue and from selected tailings samples.
9. Cyanide destruction, liquid-solids separation and environmental testing of the gold recovery residues.

Confirmation of the pressure oxidation conditions and results

The test work aimed to confirm the POX conditions and results on freshly produced cobalt concentrate and bismuth concentrate ferric chloride leach residues. Most of the cobalt concentrate sample available was produced during the 2024 flotation mini-pilot campaign and it was about half the grade compared to the "design" grade concentrate tested during the 2008 hydrometallurgical pilot campaign. This test sample was identified as "Off-spec". A small amount of design specification sample (with respect to cobalt content) was produced by a subsequent bench-scale batch sequence in 2025 using a higher cobalt grade ore. This test sample was identified as "On-spec". One intermediate sample produced during the benchwork was identified as "Near on spec".

The discharge liquors and residues produced from the off-spec sample were used for baseline and optimization tests. The discharge PLS sample and subsequent purified solution samples produced from the on-spec sample were used to validate the optimized conditions and produce the final hepta product.

POX test conditions

The initial POX conditions were confirmed at bench scale under identical conditions for all samples: 180°C, one hour residence time and 6.8 atmospheres oxygen overpressure. Four small scale (2-L Parr) tests were conducted, and two bulk scale (20-L Parr) test were conducted. Bulk test POX 4 was done on the feed containing the “On-spec” cobalt concentrate. Except for test POX1, the solids content of the test-feed was estimated based on the autothermic requirement.

POX test results

Bulk test POX3 (off-spec 1.4% Co concentrate with BiL residue) produced 95% Co, 79% Cu, with 20% Fe and 6% As, as notable co-extractions. Bulk test POX4 (on-spec 3.81% Co concentrate with BiL residue) produced 97% Co 74% Cu, with 19% Fe and 5% As as notable co-extractions. Bulk test POX4 pregnant leach solution tenors (before dilution with filtrate) assayed 5.6 g/L Co, 0.6 g/L Cu, 8.7 g/L Fe and 0.6 g/L As. For comparison, representative 2008 pilot PLS contained on average 5.5 g/L Co, 0.7 g/L Cu, 3.2. g/L Fe and 0.9 g/L As.

A key differentiating factor between the tests conducted with On-spec (POX 3) and Off spec (POX 4) samples feeds was the ferric to arsenic ratios of 23 vs. 14, respectively, hinting to the possibility of subsequent increased arsenic removal even without the use of oxygen.

The residues were also comparable except for the sulphur as sulphide which assayed 0.6% S²⁻ in POX 4 residue (3.2% total sulphur), compared to 2.4 S²⁻ in the pilot residue (4.9% total sulphur), indicative of potentially more advanced oxidation in the current work, or/and, due to the nature of the current sample. Of note, the 2008 pilot sample consisted of only cobalt concentrate, not blended with bismuth ferric chloride leach residue.

All in all, no surprises, with all previous work (bench and pilot), and with both autoclave feeds (cobalt only and blend) successfully validated. The existing PDC package submitted in November 2024 remains in effect.

Table 27 Pressure oxidation test conditions

Test ID	POX1	POX2	POX3	POX4	POX5	POX6
Purpose	2 L Scoping Cobalt Conc	2 L Scoping	20 L Bulk Cobalt Conc	20 L Bulk Blended	21 L Bulk Cobalt Conc	22 L Bulk Cobalt Conc
Main Feed	Cake Pail 2 (PP- 205)	F18 Rougher Scav Tail	Cake Pail 2 (PP- 205)	F19/20/21 Co Conc	Cake Pail 2 (PP- 205)	Cake Pail 2 (PP- 205)
Co Con spec - primary feed	Off	Near on	Off	On	Off	Off
Secondary Feed	BiL6 Residue					
Primary : Secondary Feed mass ratio, dry, <->: 0.097						
POX feed solids content, % wt.	10%	14%	18%	14%	18%	18%
Temperature (°C)	180					
Oxygen Overpressure (kPa)	689					
Acidification (kg/t)	35					
Retention Time (minutes)	60					
Lignosol Addition (kg/t)	3					
Final Acidity (g/L)	26	30	40	38	37	38

Table 28 Pressure oxidation leaching efficiencies

Extraction (%)	POX1	POX2	POX3	POX4	POX5	POX6
Bi	2	0	1	1	1	0
Co	93	95	95	97	92	95
Cu	70	60	79	74	70	76
Fe	17	12	20	19	18	16
Mg	30	32	39	36	36	35
Al	31	39	45	59	41	42
Cr	19	11	33	31	27	27
Mn	35	34	44	43	42	40
Ca	26	20	21	26	13	15
Na	17	11	16	15	22	17
As	2	2	6	5	4	4
S	0	0	0	0	0	0
S=	0	0	0	0	0	0
S0	-	-	-	-	-	-
Au	0	-	-	0	0	0
Ag	18	16	13	15	8	6
Pb	26	10	16	14	10	11
Sb	4	3	5	4	1	1
Sn	58	53	47	52	34	34
Weight loss	-3%	-3%	0%	3%	4%	2%

Table 29 Pressure oxidation final solutions

Final PLS (mg/L)	POX1	POX2	POX3	POX4	POX5	POX6
Bi	24	1	37	12	15	7
Co	1570	4350	2750	5620	2700	2730
Cu	555	557	1110	629	990	1060
Fe	5520	5570	10700	8670	10700	9630
Mg	504	838	1130	746	1070	992
Al	642	1170	1630	1210	1470	1490
Cr	9	30	28	78	24	22
Mn	44	71	98	76	92	88
Ca	895	783	960	973	745	911
Na	62	45	111	48	126	110
As	238	287	895	599	832	975
S	0	0	0	0	0	0
S=	0	0	0	0	0	0
S0	0	0	0	0	0	0
Au	0	0	0	0	0	0
Ag	0	0	0	0	0	0
Pb	2	2	2	2	2	2
Sb	1	1	2	2	1	1
Sn	2	2	2	2	2	2
Fe (II)	-	-	863	358	1170	1300
Fe (III) / Fe(II) ratio	-	-	11.4	23	8	6
Fe (III) / As ratio	-	-	11.0	14	11	9

Table 30 Pressure oxidation final residues

Final Residue (g/t)	POX1	POX2	POX3	POX4	POX5	POX6
Bi	15000	7240	14700	6310	14800	14700
Co	1030	1630	750	1280	1170	742
Cu	2240	2430	1460	1580	2150	1730
Fe	244000	268000	231000	275000	256000	252000
Mg	11000	11600	8840	9570	9750	9660
Al	13000	11800	9720	6120	10900	10700
Cr	354	1560	288	1280	323	314
Mn	745	894	615	750	673	680
Ca	26600	29900	26100	25800	26800	26900
Na	2830	2460	2980	1960	2330	2800
As (%)	12	8	12	9	10	11
S (%)	2.7	3.8	3.4	3.2	3.3	3.3
S= (%)	0.7	1.3	0.3	0.6	0.9	0.8
S0 (%)	0	0	0	0	0	0
Au	72	0	0	95	91	90
Ag	5	5	5	5	5	6
Pb	80	200	98	128	98	85
Sb	339	331	351	395	349	348
Sn	20	20	20	20	20	20

Table 31 Pressure oxidation feed samples

Test feed, g/t	POX1	POX2	POX3	POX4	POX5	POX6
Bi	13924	6820	13893	5339	13876	13876
Co	13503	28304	13489	35395	13481	13481
Cu	6580	5656	6610	5247	6626	6626
Fe	267255	288896	267087	293445	266998	266994
Mg	13959	15964	13964	12691	13966	13966
Al	16899	18241	16879	12788	16868	16867
Cr	395	1382	395	1582	394	394
Mn	1031	1230	1030	1121	1030	1030
Ca	31785	34499	31772	28954	31766	31766
Na	3307	2796	3305	2315	3304	3304
As (%)	10	0	10	9	10	10
S (%)	15.5	2.5	15.5	17.7	15.5	15.5
S= (%)	12.9	12.5	12.8	14.6	12.8	12.8
S0 (%)	0	0	0	0	0	0
Au	67	57	69	98	70	70
Ag	6	5	6	5	6	6
Pb	88	101	88	122	88	88
Sb	320	313	323	346	324	324
Sn	20	20	20	20	20	20

Selective gypsum precipitation test work conditions and results

Selective gypsum precipitation test work investigating whether or not the gypsum produced according to the 2008 and 2017 current flowsheet contains sufficiently low arsenic levels as to allow for its safe disposal (as non-registered waste) or even potential commercial uses

Gypsum test conditions

Classic endpoint tests were conducted, investigating gypsum precipitation / acid trimming by neutralization on POX liquor. The iron and arsenic co-precipitation were quantified and assessed with a view to their selective precipitation. The resulting precipitate compositions were determined and evaluated.

Gypsum test results

Arsenic co-precipitation, while subdued, still resulted in measurable concentrations in the potential product. The tests produced removals from 21% to 83% of the free acid while producing gypsum residues containing from 30 ppm As (0.02% removal) to 199 ppm As (1.25% removal), respectively, and within the pH ranging from about 1.1 to 1.4, respectively.

The predicted gypsum output on a more confident basis would range from 13k tpy (64 g/t As) to 19 tpy (199 g/t As) on an equivalent dry basis. The predicted gypsum market size for 2030 is about 938 Mtpy at 10-15 USD per metric tonne, equivalent dry basis. Hence no apparent economic impact on the NICO Project (100-300 k USD\$ per year).

It is recommended that an assessment of industries able to use and accept arsenic in gypsum be undertaken. If no industries can be identified, then additional testing and treatment of gypsum material will need to be undertaken to meet non-hazardous classification. No commercial production is recommended until the marketing is study complete, and as such, no design criteria has been issued.

Table 32 Gypsum precipitation test conditions

Test ID	GP1	GP2	GP3	GP4	GP5	GP6
Purpose	Scoping	Scoping	Scoping	Scoping	Scoping	Scoping
Feed	POX2 Filtrate	GP1 Filtrate	Diluted/Spiked POX3 Filtrate	Diluted/Spiked POX3 Filtrate	Diluted/Spiked POX3 Filtrate	Diluted/Spiked POX3 Filtrate
Reagent	Ca(OH) ₂	Ca(OH) ₂	Ca(OH) ₂	Ca(OH) ₂	Ca(OH) ₂	Ca(OH) ₂
Target pH/Acidity	1	5 g/L H ₂ SO ₄	variable	9 g/L	14 g/L	4 g/L
Temperature (°C)	85	85	85	85	85	85
Retention Time (h) ¹	2	2	0.5	2.0	2.0	2.0
Final Pulp pH	1.11	1.31	4.12	1.03		
Final Free Acid (g/L H ₂ SO ₄)	23.9	8.0	-	11.6	16.1	6.5
Reagent Addition (kg/m ³)	4.5	13.6	53.1	20.2	16.2	24.2
Calc. Acid Neutralization	21%	69%	98%	70%	58%	83%
Calcium/Gypsum Conversion (g/L)	8.7	22.8	49.5	34.6	28.6	41.0

Table 33 Gypsum precipitation efficiencies

Precipitation (%)	GP1	GP2	GP3	GP4	GP5	GP6
Bi	7	18	92	51	46	48
Co	0	0	4	0	0	0
Cu	0	0	80	0	0	0
Fe	0	0	99	0	0	1
Mg	0	0	2	0	0	0
Al	0	0	95	0	3	2
Cr	1	13	98	30	23	32
Mn	0	0	1	0	0	1
Ca	51	82	96	87	85	88
Na	0	4	20	5	27	16
As	0.0	0.3	99.0	0.5	0.2	1.2
S	-	-	-	-	-	-
S=	-	-	-	-	-	-
S0	-	-	-	-	-	-
Au	-	-	-	-	-	-
Ag	5	16	62	33	18	44
Pb	18	49	49	16	12	17
Sb	2	37	49	9	6	9
Sn	2	9	39	16	12	17

Table 34 Gypsum precipitation final solutions

Final PLS (mg/L)	GP1	GP2	GP3	GP4	GP5	GP6
Bi	1	1	1	14	11	10
Co	4410	4590	4800	5710	5800	5560
Cu	555	589	23	668	665	632
Fe	5630	5840	8	9010	9250	8600
Mg	844	968	991	947	934	931
Al	1200	1270	1	1310	1360	1250
Cr	31	31	0	16	16	16
Mn	73	76	66	70	71	68
Ca	617	628	512	895	838	798
Na	62	52	72	72	60	59
As	288	304	3	608	617	585
S	-	-	-	-	-	-
S=	-	-	-	-	-	-
S0	-	-	-	-	-	-
Au	-	-	-	-	-	-
Ag	-	-	-	-	-	1
Pb	2	2	2	2	2	2
Sb	1	1	2	2	2	2
Sn	2	2	2	2	2	2
Fe (II)	977	0	0	0	0	0

Table 35 Gypsum precipitation final residues

Final Solids (g/t)	GP1	GP2	GP3	GP4	GP5	GP6
Bi	36	22	177	430	405	252
Co	23	38	2080	41	53	120
Cu	8	20	4460	16	20	36
Fe	612	490	65300	361	770	1810
Mg	40	40	157	56	97	107
Al	100	100	9810	100	1740	715
Cr	200	200	118	200	200	200
Mn	5	5	8	5	10	15
Ca	284000	281000	188000	271000	311000	254000
Na	100	100	195	100	926	319
As	30	40	4500	80	64	199
S (%)	-	-	-	-	-	-
S= (%)	-	-	-	-	-	-
S0 (%)	-	-	-	-	-	-
Au	-	-	-	-	-	-
Ag	2	4	2	2	2	30
Pb	200	200	30	20	20	20
Sb	10	60	30	10	10	10
Sn	20	20	20	20	20	20
Fe (II)	0	0	0	0	0	0

GP2 residue assays of 40 g/t As suspicious.

Table 36 Gypsum precipitation feed solutions

Feed, (mg/L)	GP1	GP2	GP3	GP4	GP5	GP6
Bi	1	1	37	23	23	37
Co	4350	4410	2750	5640	5640	2750
Cu	557	555	1110	642	642	1110
Fe	5570	5630	10700	8840	8840	10700
Mg	838	844	1130	816	816	1130
Al	1170	1200	1630	1270	1270	1630
Cr	30	31	28	16.1	16.1	28
Mn	71	73	98	66.4	66.4	98
Ca	783	617	960	636	636	960
Na	45	62	111	66	66	111
As	287	288	895	589	589	895
S	-	-	-	-	-	-
S=	-	-	-	-	-	-
S0	-	-	-	-	-	-
Au	-	-	-	-	-	-
Ag	-	-	-	<0.08	<0.08	-
Pb	2	2	2	<2	<2	2
Sb	1	1	2	<2	<2	2
Sn	2	2	2	<2	<2	2
Fe (II)	0	977	863	600	600	863

Non-oxidative preneutralization test work conditions and results

Objectives:

Non-oxidative preneutralization test work aiming to investigate iron/arsenic removal on the entire unseparated POX slurry discharge while taking advantage of the oxidation potential of the ferric ion contained in the POX-PLS;

Comparing limestone vs. lime as neutralization reagents to allow for process design and cost flexibility;
Producing residue and liquor samples for the gold recovery and cementation test works, respectively.

Primary neutralization test conditions

- The tests were similar to the gypsum precipitation tests but at set final pH value of 2.5.
- Tests PN1 and PN4 consisted of pre-neutralization of the entire ACD slurry.
- Limestone was the neutralizing reagent used in tests PN1 and PN3.
- Lime was the neutralizing reagent used in tests PN2 and PN4.
- The Final Solids Ratios were calculated directly for tests PN1 and PN4.
- Tests PN2 and PN3 consisted of pre-neutralization of the ACD-PLS only. The Final Solids Ratios were calculated indirectly for these tests.
- Feed for PN2 was generated from off-spec Co con (POX3) by diluting and spiking the PLS to simulate POX 4 D/C PLS.

Key parameters included: no air or oxygen sparging, simple precipitation tests of 0.5 hours residence time, pH ~ 2.5, no heat applied, initial / final ORP values as “existing in feed / produced during the test” and ranging from 552 to 661 mVAg/AgCl, and from 398 to 664 mVAg/AgCl. Of note, the maximum values from each range occurred in tests PN3 run with the “On Spec” sample.

Non-oxidative preneutralization test results

Arsenic precipitation was 99% in all tests, along with iron precipitation ranging from 80% to 92% and without any cobalt losses by co-precipitation and entrapment.

All other metallic species precipitated as expected at the tested pH of 2.5, and with the iron removal comparable with the results produced with PLS-sample during the 2017 test work but using oxygen and heated. However, the arsenic removal in 2017 ranged from 56% to 91%, inferior compared with the current results produced without separating the residue from the PLS.

Overall, the results of the 2025 primary neutralization test work provided that effective removal of 99% of the arsenic and from 80% to 92% of iron under optimized conditions was possible at pH ~ 2.5, without oxygenation and heating, using either limestone or lime and with a very short residence time of 0.5 hours.

The results were used as design criteria (listed in the dedicated section). Nonoxidative preneutralization discharge liquid solid separation. The results of the 2017 dynamic testing and subsequent filtration test works were relied upon. While comparable to the Pockock work (2008) the results are conservative because the solids contain heavy gangue (POX residue) at a ratio of ~1.5/1 vs. the gypsum component and the actual pH is lower (2.5 vs. 4.6). Dewatering of the final moisture content should be amenable to extrapolation to a squeeze-pressure-filter target. The results were used as design criteria (listed in the dedicated section).

Table 37 Primary neutralization test conditions

Test ID	PN1	PN2	PN3	PN4	
Purpose	Scoping	Scoping	Scoping	Scoping	
Feed	POX5 Pulp	Diluted/Spiked POX3 Filtrate	POX4 Filtrate	POX6 Pulp	
Feed Background	Off-spec Conc.	On/Off-spec Conc.	On-spec Conc.	Off-spec Conc.	
Reagent	CaCO ₃	Ca(OH) ₂	CaCO ₃	Ca(OH) ₂	
Target pH/Acidity	2.5	2.5	2.5	2.5	
Temperature (°C)	85	85	85	85	
Retention Time (h) ¹	0.5	0.5	0.5	0.5	
Final Pulp pH	2.50	2.50	2.50	2.49	
Initial Pulp ORP, mV Ag/AgCl	556	614	661	552	
Final Pulp ORP, mV Ag/AgCl	413	426	664	398	
Initial Free Acid (g/L H ₂ SO ₄)	37.2	35.7	38.4	37.8	
Final Free Acid (g/L H ₂ SO ₄)	1.7	0.0	0.0	1.6	
Tech. Reagent Addition (kg/m ³ PLS)	58.6	43.9	55.2	42.2	
Tech. Reagent Addition (kg/t conc.)	292	310	390	213	
kg CaO equiv. / m3 PLS	31	33	29	31	
kg CaO equiv. / t H ₂ SO ₄ neut.	875	912	756	868	
kg CaO equiv. / t Fe removed	3451	4046	3683	4046	
kg CaO equiv. / t Co Conc.	154	230	205	158	
kg CaO equiv. / t Blend POX feed	140	209	187	144	
Final Solids	g/g Co Conc.	1.56587	1.66	1.64	1.58
Ratio	g/g Blend. POX Feed	1.42	1.51	1.49	1.44

Table 38 Primary neutralization efficiencies

Reagent used	CaCO ₃	Ca(OH) ₂	CaCO ₃	Ca(OH) ₂
Precipitation (%)	PN1/POX 5	PN2/POX 3	PN3/POX 4	PN4/POX 6
Bi	92	96	93	84
Co	0	0	0	0
Cu	1	1	0	0
Fe	84	92	92	80
Mg	0	1	7	0
Al	28	23	18	27
Cr	87	83	74	85
Mn	0	1	1	0
Ca	14	97	97	32
Na	27	13	28	21
As	99	99	99	99
S	-	-	-	-
S=	-	-	-	-
S0	-	-	-	-
Au	-	-	-	-
Ag	0	68	67	0
Pb	0	46	45	0
Sb	0	46	29	0
Sn	0	46	45	0
Fe (II)	100	-	-	100
POX 4 / PN 3 - On spec AC Feed				

Table 39 Primary neutralization final solutions

Reagent used	CaCO ₃	Ca(OH) ₂	CaCO ₃	Ca(OH) ₂
Final PLS (mg/L)	PN1/POX 5	PN2/POX 3	PN3/POX 4	PN4/POX 6
Bi	1	1	1	1
Co	2420	5240	5350	2560
Cu	870	605	583	977
Fe	1550	708	663	1730
Mg	1580	940	1260	1020
Al	934	908	945	992
Cr	3	2	14	3
Mn	85	65	74	86
Ca	569	574	560	569
Na	82	60	40	80
As	5	3	4	11
S	0	0	0	0
S=	0	0	0	0
S0	0	0	0	0
Au	0	0	0	0
Ag	0	0	0	0
Pb	2	2	2	2
Sb	1	1	2	1
Sn	2	2	2	2
Fe (II)	0	0	0	0
POX 4 / PN 3 - On spec AC Feed				

Table 40 Primary neutralization final residues

Reagent used	CaCO ₃	Ca(OH) ₂	CaCO ₃	Ca(OH) ₂
Final Solids (g/t)	PN1/POX 5	PN2/POX 3	PN3/POX 4	PN4/POX 6
Bi	9350	265	162	9310
Co	704	73	13	461
Cu	1390	69	31	1130
Fe	180000	88400	91200	175000
Mg	6530	60	1150	6130
Al	8100	3160	2640	8030
Cr	268	108	500	258
Mn	422	5	5	424
Ca	89400	230000	232000	89500
Na	1660	100	191	1920
As	69200	6600	7700	80400
S (%)	0	0	0	0
S= (%)	0	0	0	0
S0 (%)	0	0	0	0
Au	54	0	0	57
Ag	4	2	2	5
Pb	64	20	20	58
Sb	225	10	10	227
Sn	20	20	20	20
POX 4 / PN 3 - On spec AC Feed				

Table 41 Primary neutralization feed solutions

Reagent used	CaCO ₃	Ca(OH) ₂	CaCO ₃	Ca(OH) ₂
Feed PLS (mg/L)	PN1/POX 5	PN2/POX 3	PN3/POX 4	PN4/POX 6
Bi	15	23	12	7
Co	2700	5640	5620	2730
Cu	990	642	629	1060
Fe	10700	8840	8670	9630
Mg	1070	816	746	992
Al	1470	1270	1210	1490
Cr	23.5	16.1	78.3	22.3
Mn	92.1	66.4	76.4	88
Ca	745	636	973	911
Na	126	66	48	110
As	832	589	599	975
S				
S=				
S0				
Au				
Ag	<0.08	<0.08	<0.08	<0.08
Pb	<2	<2	<2	<2
Sb	<1	<2	<2	<1
Sn	<2	<2	<2	<2
Fe (II)	1170	600	358	1300
POX 4 / PN 3 - On spec AC Feed				

Table 42 POX discharge solids used (i.e. fed) into the PN tests

Reagent used	CaCO ₃	Ca(OH) ₂	CaCO ₃	Ca(OH) ₂
POX discharge solids used (i.e. fed) into the tests				
Feed Solids (g/t)	PN1/POX 5	PN2/POX 3	PN3/POX 4	PN4/POX 6
Bi	14800	-	-	14700
Co	1170	-	-	742
Cu	2150	-	-	1730
Fe	256000	-	-	252000
Mg	9750	-	-	9660
Al	10900	-	-	10700
Cr	323	-	-	314
Mn	673	-	-	680
Ca	26800	-	-	26900
Na	2330	-	-	2800
As	99900	-	-	111000
S	3.34	-	-	3.32
S=	0.88	-	-	0.76
S0	-	-	-	-
Au	90.5	-	-	90.2
Ag	5	-	-	6
Pb	98	-	-	85
Sb	349	-	-	348
Sn	<20	-	-	<20
POX 4 / PN 3 - On spec AC Feed				

Table 43 POX discharge solids not used (i.e. not fed) into the PN tests

Composition of the POX discharge solids NOT used (i.e. NOT fed) into the tests				
Reagent used	CaCO ₃	Ca(OH) ₂	CaCO ₃	Ca(OH) ₂
Feed Solids (g/t)	PN1/POX 5	PN2/POX 3	PN3/POX 4	PN4/POX 6
Bi	-	14700	6310	-
Co	-	750	1280	-
Cu	-	1460	1580	-
Fe	-	231000	275000	-
Mg	-	8840	9570	-
Al	-	9720	6120	-
Cr	-	288	1280	-
Mn	-	615	750	-
Ca	-	26100	25800	-
Na	-	2980	1960	-
As	-	11.7	9.23	-
S	-	3.35	3.21	-
S=	-	0.33	0.55	-
S0	-	0	0	-
Au	-	0	94.9	-
Ag	-	<5.00	<5.00	-
Pb	-	<98	<128	-
Sb	-	<351	<395	-
Sn	-	<20	<20	-
POX 4 / PN 3 - On spec AC Feed				

Table 44 Composition of the feed into the precursor POX tests

Composition of the feed into the precursor POX tests				
Reagent used	CaCO ₃	Ca(OH) ₂	CaCO ₃	Ca(OH) ₂
Feed Solids (g/t)	POX5/PN1	POX 3/PN2	POX4/PN3	POX6/PN4
Bi	13,876	13,893	5,339	13,876
Co	13,481	13,489	35,395	13,481
Cu	6,626	6,610	5,247	6,626
Fe	266,998	267,087	293,445	266,994
Mg	13,966	13,964	12,691	13,966
Al	16,868	16,879	12,788	16,867
Cr	394	395	1,582	394
Mn	1,030	1,030	1,121	1,030
Ca	31,766	31,772	28,954	31,766
Na	3,304	3,305	2,315	3,304
As	102,825	102,932	91,384	102,821
S	155,466	155,280	177,103	155,474
S=	127,980	128,174	146,346	127,972
S0	0	0	0	0
Au	70	69	98	70
Ag	6	6	5	6
Pb	88	88	122	88
Sb	324	323	346	324
Sn	20	20	20	20
Co/Bi Ratios in the POX feed				
By mass, dry eq/nt	0.10	0.10	0.10	0.10
By Bi content	0.02	0.02	0.05	0.02
POX 4 / PN 3 - On spec AC Feed				

Table 45 First stage Iron Arsenic Removal - HDS Thickening 2017 Work

First stage Iron Arsenic Removal - HDS Thickening 2017 Work								
Undiluted thickener feed		Diluted thickener feed		Specific Unit Area m ² /(t/day)	Net loadings		Underflow Pulp kg/L	
Pulp kg/L	Solids % wt.	Pulp kg/L	Solids % wt.		Solids t/m ² /h	Hydraulic m ³ /m ² /h		
1.072	9.0	1.026	2.5	0.7	0.060	2.26	1.28	
Particle size: K ₈₀ = 6.7 microns, Actual Solids SG = 2.79, Solution SG = 1.01, pH ~ 2.5Temp.: 75 °C.								
Maximum HDS predicted underflow solids content 37%. Vane yield could not be measured. Rheopectic.								
Flocculant: BASF Magnafloc 338 high molecular weight, anionic polyacrylamide. Dosed at 30 g/t as 0.25 g/L stock solution.								
"As produced" bulk discharge feed predicted solids content ~ 6.4% wt., to be thickened by recycling the underflow ("HDS") then autodiluted in the feedwell to about 2.5% wt. solids.								
CCRV CSD = ~36% solids, corresponding to ~ 19 Pa unsheared and 25 Pa sheared yield stress. Rheopectic.								
Residence times, hours								
Bed Height mm	Versus the entire equipment volume				³ Solids phase	Versus OF and UF volumes		
	¹ Feed slurry		² Liquid phase			⁴ Liquid vs. OF		
	Undiluted	Diluted	Undiluted	Diluted		Undiluted	Diluted	
180.0	0.46	0.12	0.48	0.12	13.39	0.17	0.05	
All values calculated without safety factor.								
¹ The actual volume of the test-unit divided by the undiluted and diluted feed slurry flowrates.								
² The actual volume of the test-unit divided by the undiluted and diluted liquid phase flowrates.								
³ The actual volume of the test-unit divided by the dry solids volumetric flowrate estimated based on SG.								
⁴ The actual volume occupied by the overflow divided by the undiluted and diluted liquid phase flowrates.								
⁵ The actual volume of the underflow divided by the dry solids volumetric flowrate estimated based on SG.								

Table 46 First stage Iron Arsenic Removal - Dewatering 2017 Work

First stage Iron Arsenic Removal design criteria - Dewatering 2017 Work				
Particle Size, d_{80} : 6.7 μm		Pulp Temperature: 75°C, Pulp pH ~ 2.89		
Solids SG: 2.79. Solution SG: 1.01.				
*Vacuum - Filter cloth: Testori P 6125 CQ Material: polypropylene, Permeability: 10.25 cfm, Warp/Weft: Mono/Mono, Weave: Two Plies, Finish: calendered, Weight: 435 g/m ²				
**Pressure Filter cloth: Micronics 861, polypropylene; warp/weft: multi/multi; weave: twill, finish: calendered, permeability: 0.70 cfm, 661.24 g/m ²				
Filter Aid, Type/Dosage, g/t: Magnafloc 333 at 125 g/t (contained in the underflow)				
Actual cake thickness, mm	30.5	20.0	0.031	0.0323
Form time, h	0.006	0.005	0.001	0.00
Dry time, h	0.0007	0.047	0.026	0.03
From/Dry	8.8	0.1	0.05	0.043
Filtrate TSS, mg/L	<10	<10	125	65
Wet cake solids, % S	42.04	48.38	60.2	62.7
Wet cake density, kg/L , WD = CW/V	1.41	1.24	1.08	1.0
Dry cake density, kg/L , DD=WD*S/100	0.59	0.60	0.65	0.6
Method	Vacuum		Pressure	
Feed / form pressure, bar	0.74	0.74	5.5	6.9
Dry / form pressure, bar	0.74	0.74	5.5	6.9
Feed solids, %			32.00	
Final wet cake moisture, %	58.0	52	40	37
Dry solids capacity, kg/m ² h, KS=CD/(A*T)	2649	231	750	788
Filter capacity, L/m ² h, KL=L/T	6514	645	2365	2236
* From raw test data / not for sizing any particular type of filter.				
¹ Throughput is calculated using cycle time which includes form and dry times only.				
Conversion: 0.098 bar = 2.896" Hg = 1 mH ₂ O = 3.28 ft H ₂ O = 1.42psi				
Refer to note for users (test details) for suggested steps to convert data into equipment sizing criteria.				

Copper cementation test work conditions and results

The objective was to investigate copper cementation using both coarse and fine iron powder. Testing of the coarse iron powder attempted to replicate the best results to date as per Pilot, 2008, [1] but without the negative implication of filtration. Testing of the fine iron powder was the fallback fail-safe method. In more detail, it was necessary to determine whether or not cementation was sufficiently robust to consistently ensure residual copper, ideally below the analytical limit of 0.3 mg/L Cu in the barren. As a result, the grade of the cement produced was not a distinct priority, since it would have called for lower iron powder additions.

Copper cementation test conditions

The first two test were conducted within the context of gypsum precipitation test work using barren (CuC1) and POX-PLS (CuC2) as test-feeds. All other test were conducted on primary neutralization liquor samples.

The test work was conducted at various initial acidity levels, iron powder additions, retention times and with or without pH control. Most importantly, tests CuC1 through CuC6 were conducted using a coarse iron powder, Atomet 110, which is only ~3% passing 100 mesh (150 microns). This was done in an effort to improve the filtration of the resulting copper cement slurry. Tests CuC7 and CuC8 were conducted using a fine iron powder, Atomet 29, with ~95% passing 100 mesh (150 microns). These tests aimed to replicate the 2008 pilot conditions on semi-depleted feed solutions from tests CuC5 and CuC6 (scavenging conditions). The resulting solutions were subjected to secondary neutralization, as described in the next section. These scavenging tests replicated the 2008 Pilot conditions: pH adjusted to 2.1 with sulfuric acid, no subsequent acid addition, 60°C temperature, iron addition add Fe at 1.45 vs. Cu and free acidity, one hour retention time

Copper cementation test results

The relevant test results validated the pilot 2008 results by achieving 0.3 mg/L Cu or less (tests CuC7, CuC8 and bulk validation test CuC9). No arsine generation risk during copper cementation was detected during the pilot validation (2008). Copper cementation discharge liquid-solid separation. According to the 2017 liquid solid separation data, pressure filtration produced dry solids capacity as low as 11 kg/m2h at 6.9 bar pressure, for about 50% wt. moisture content cake. That test work was performed on cement of similar particle size to Pilot 2008 / Test Cu7/CuC8. These results are applicable but preferably by inserting a thickener and “seeding” the feedwell by recycling some underflow to allow for increased solids content in the filtration feed. In addition, the final moisture content should be amenable to extrapolation to a squeeze-pressure-filter target. Centrifugation could be considered as a last resort option due to its cost (capex primarily). Validation of this approach was realized by performing the dynamic test on the CuC9 bulk test discharge. There was no possibility of performing the preliminary static tests due to the very low solids content of the thickener feed (0.3% wt.) In addition, the settling rate was extremely high, instantly producing a 58% wt. underflow.

All in all, the existing and validated Pilot 2008 results were used as design criteria, and, in conjunction with the 2025 liquid-solid separation generated criteria (listed in the dedicated section).

Table 47 Copper cementation test conditions

Test ID	CuC1	CuC2	CuC3	CuC4	CuC5	CuC6	CuC7	CuC8	CuC9
Purpose	Scoping	Scoping	Scoping	Scoping	Scoping	Scoping	Scoping	Scoping	Validation
Feed	GP2 Filtrate	Diluted/Spiked POX3 Filtrate	PN2 Filtrate	PN2 Filtrate	PN2 Filtrate	PN3 Filtrate	CuC5 Filtrate	CuC6 Filtrate	Synthetic SN Discharge Liquor
Acidity to Cementation (g/L H2SO4)	8.0	35.7	0.00	0.00	0.00	0.00	0.88	1.1	2.5
Iron Powder Source	Atomet 110	Atomet 110	Atomet 110	Atomet 110	Atomet 110	Atomet 110	Atomet 29	Atomet 29	Atomet 29
Actual Iron Powder Dose (Overall FA, Cu)	119%	103%	120%	114%	120%	121%	140%	140%	55%
Final Pulp pH	1.74	0.74	1.99	2.74	2.55	2.59	3.38	3.50	3.59
Final pulp ORP, mV ^{Ag/AgCl}	nd	-221.00	368.50	253.00	364.00	341.00	-394.00	-432.00	-196
Cementation Temperature (°C)	60	60	60	60	60	60	60	60	60
Fe Addition (kg/kg Cu)	30.6	95.0	1.8	1.7	1.8	1.8	8.0	20.2	12.1
Maximum Cementation Time (min) ¹	20	10	10	10	20	20	60	60	60

¹Passing 100 mesh (150 microns): Atomet 110 ~3%, Atomet 29: 95%.

Table 48 Copper cementation efficiencies

Solids Department (%)	CuC1	CuC2	CuC3	CuC4	CuC5	CuC6	CuC7	CuC8	CuC9
Bi	23	95	0	0	2	1	0	0	3
Co	1	1	0	0	0	0	0	0	1
Cu	94	99	0	0	51	76	100	100	100
Fe	48	79	0	0	31	6	0	0	48
Mg	0	0	0	0	0	0	0	0	0
Al	0	2	0	0	1	1	0	0	11
Cr	10	53	0	0	15	3	0	0	87
Mn	12	45	0	0	1	0	0	0	0
Ca	0	1	0	0	0	0	0	0	0
Na	12	27	0	0	0	1	0	0	6
As	31	83	0	0	12	16	0	0	2
S	-	-	0	0	-	-	-	-	-
S=	-	-	0	0	-	-	-	-	-
S0	-	-	0	0	-	-	-	-	-
Au	-	-	0	0	-	-	-	-	-
Ag	33	81	0	0	2	2	0	0	4
Pb	9	63	0	0	2	1	0	0	1
Sb	9	41	0	0	1	1	0	0	1
Sn	11	41	0	0	1	1	0	0	2

Table 49 Copper cementation final solutions

Final PLS (mg/L)	CuC1	CuC2	CuC3	CuC4	CuC5	CuC6	CuC7	CuC8	CuC9
Bi	1	1	1	1	1	1	1	1	1
Co	4700	5830	4690	5460	5400	5620	5700	5390	2190
Cu	15	3	202	92	279	122	0	0	0.4
Fe	10300	16600	1110	1530	1270	1620	3250	3150	1300
Mg	1010	851	867	965	1040	1330	994	1260	248
Al	1290	1320	828	909	919	971	809	802	367
Cr	32	17	2	2	2	14	2	11	0
Mn	85	93	62	65	70	77	68	77	31
Ca	645	679	532	591	569	586	580	561	497
Na	52	69	58	60	64	41	62	75	18
As	161	99	3	3	3	3	3	3	3
S	0	0	0	0	0	0	0	0	0
S=	0	0	0	0	0	0	0	0	0
S0	0	0	0	0	0	0	0	0	0
Au	0	0	0	0	0	0	0	0	0
Ag	0	0	0	0	0	0	0	0	0
Pb	2	2	2	2	2	2	2	2	2
Sb	1	1	1	1	1	1	1	1	1
Sn	2	2	2	2	2	2	2	2	2
Fe (II)	0	0	0	0	0	0	0	0	0

Table 50 Copper cementation final residues

Final Solids (g/t)	CuC1	CuC2	CuC3	CuC4	CuC5	CuC6	CuC7	CuC8	CuC9
Bi	30	308	0	0	20	20	0	0	20
Co	3250	678	0	0	1470	4780	0	0	11100
Cu	21500	10200	0	0	286000	522000	198000	43300	85900
Fe	948000	942000	0	0	576000	145000	0	0	790000
Mg	40	40	0	0	40	307	0	0	55
Al	444	328	0	0	10400	19800	0	0	30500
Cr	350	279	0	0	350	682	0	0	448
Mn	1160	1120	0	0	802	435	0	0	84
Ca	100	100	0	0	1240	795	0	0	100
Na	675	377	0	0	298	490	0	0	745
As (%)	7200	7050	0	0	393	794	0	0	39
S (%)	0	0	0	0	0	0	0	0	0
S= (%)	0	0	0	0	0	0	0	0	0
S0 (%)	0	0	0	0	0	0	0	0	0
Au	0	0	0	0	0	0	0	0	0
Ag	4	5	0	0	2	2	0	0	2
Pb	20	50	0	0	40	20	0	0	20
Sb	10	10	0	0	10	16	0	0	10
Sn	24	20	0	0	21	20	0	0	21
Fe (II)	0	0	0	0	0	0	0	0	0

Table 51 Copper cementation feed solutions

Feed solutions.(mg/L)	CuC1	CuC2	CuC3	CuC4	CuC5	CuC6	CuC7	CuC8	CuC9
Bi	1	37	1	1	1	1	1	1	1
Co	4590	2750	5240	5240	5240	5350	5400	5620	2210
Cu	589	1110	605	605	605	583	279	122	178
Fe	5840	10700	708	708	708	663	1270	1620	447
Mg	968	1130	940	940	940	1260	1040	1330	244
Al	1270	1630	908	908	908	945	919	971	391
Cr	31	28	2	2	2	14	2	14	0
Mn	76	98	65	65	65	74	70	77	31
Ca	628	960	574	574	574	560	569	586	481
Na	52	111	60	60	60	40	64	41	19
As	304	895	3	3	3	4	3	3	3
S	-	0	0	0	0	0	0	0	
S=	-	0	0	0	0	0	0	0	
S0	-	0	0	0	0	0	0	0	
Au	-	0	0	0	0	0	0	0	
Ag	-	0	0	0	0	0	0	0	0
Pb	2	2	2	2	2	2	2	2	2
Sb	1	2	1	1	1	2	1	1	1
Sn	2	2	2	2	2	2	2	2	2

Secondary neutralization test work conditions and results

Secondary oxidative neutralization test work aiming to investigate the maximum removal of the remaining iron and arsenic, critical for the purity of the final cobalt sulphate heptahydrate product.

Secondary neutralization test conditions

Quicklime used as reagent. The test-feed-samples consisted of both the limestone and quicklime generated feed solutions. Oxygen and air were compared as oxidizing agents. As opposed to the previous 2017 test work, specific tests were conducted at an initial temperature of 60°C and allowed to cool naturally (except for test SN1 and SN2), as to be amenable for a Pachuca type of tank design.

Tests SN1 and SN2 were conducted using filtrate produced by test CuC5, which in turn was produced from the off-spec POX-feed through neutralization using limestone. The aim of these tests was to investigate if a combined secondary neutralization and polishing step could be possible (pH ~ 6.4) while held at 60°C throughout.

Tests SN3 and SN4 were conducted using synthetic filtrate resembling a real discharge produced by the worst-case cementation with coarse iron powder (~280 mg/L Cu). These tests were constricted to classic secondary neutralization conditions (pH ~ 4.6). The tests were differentiated by air sparging (SN3) vs. oxygen sparging (SN4).

Tests SN5 and SN6 were conducted using cementation barren solutions from CuC7 and CuC8. The original precursors of these samples consisted of the bulk pressure oxidation conducted with “Off spec” (POX3) and “On spec” (POX4), feeds, respectively, preneutralized with quicklime (PN2) and limestone (PN3), respectively. These tests were conducted at an initial temperature of 60°C and allowed to cool during the test.

Tests SN7 and SN8 consisted of hydrogen peroxide polishing tests conducted on SN 5 and SN 6 discharge solutions. Tests SN 9 and SN10 consisted of Pachuca reactor amenability tests with oxygen and air, respectively, conducted on CuC9 copper cementation validation tests discharge solution.

Secondary neutralization test results

Tests SN1 and SN2 results produced unacceptable cobalt loss, indicating that a combined secondary neutralization and polishing step was not possible (pH ~ 6.4).

Tests SN3 and SN4 results produced the design criteria at 60°C initial feed temperature, without heating. The results indicated the possibility of completely removing the iron from the cementation discharge barren (<70 mg./L Fe with air, 10 mg/L with oxygen) and arsenic (<3 mg/L, both oxidants),

Whilst tests SN3 and SN4 produced the design criteria in this original (unrevised) version, they were ultimately rejected due to the unfavorable response during the subsequent manganese removal stage.

Tests SN5 and SN6 did not improve the iron precipitation, needing tests SN 7 and SN 8 to remove the iron quantitatively by hydrogen peroxide polishing through tests SN 7 and SN8.

Pachuca tests SN 9 and SN10 also produced dismay iron removal efficiencies (30% and 18%, respectively), requiring hydrogen peroxide polishing (test SN 11 and SN 12) to achieve residual iron below 1 mg/L.

Overall, the efforts toward improving the secondary neutralization by removing the heating, reducing residence time, avoiding polishing, and using Pachuca reactors failed. As a result, the design criteria were reverted to the 2017 conditions, listed in the dedicated section.

Secondary oxidative neutralization discharge liquid-solid separation

The results of the 2017 dynamic testing and subsequent filtration test works were relied upon as they were conducted under similar conditions.

The results are applicable if needed but preferably by recycling the thickener underflow to the actual process (“seeding”) to allow for increased solids content in the filtration feed. Short-route recycling to the feedwell is also an option. Dewatering of the final moisture content should be amenable to extrapolation to a squeeze-pressure-filter target. Centrifugation could be considered as a last resort option due to its cost (capex primarily).

The results of the 2017 liquid solid separation were used as design criteria (listed in the dedicated section). The design criteria is common for both the limestone and quicklime streams since only quicklime is used in this stage.

Table 52 Secondary neutralization test conditions

Test ID	SN1	SN2	SN3	SN4	SN5	SN6	SN7	SN8	SN9	SN10	SN11	SN12
Purpose	Scoping	Scoping	Scoping	Scoping	Scoping	Scoping	Scoping	Scoping	Scoping Pachuca	Polishing of	Pachuca d/c	
Feed	CuC5 Filtrate	CuC5 Filtrate	Synthetic SN Feed Liquor (May 2025)	Synthetic SN Feed Liquor (May 2025)	CuC7 Filtrate	CuC8 Filtrate	SN5 Filtrate	SN6 Filtrate	CuC9 Decant	CuC9 Decant	SN9 Filtrate	SN9 Filtrate
Feed Background	Off-spec Conc.	Off-spec Conc.	Synthetic	Synthetic	Off-spec Conc.	On-spec Conc.	On-spec Conc.	On-spec Conc.	Synthetic	Synthetic	Synthetic	Synthetic
Neut. Reagent	Ca(OH)2	Ca(OH)2	Ca(OH)2	Ca(OH)2	Ca(OH)2	Ca(OH)2	Ca(OH)2	Ca(OH)2	Ca(OH)2	Ca(OH)2	Ca(OH)2	Ca(OH)2
Target pH/Acidity	Variable	Variable	4.6	4.6	4.6	4.6	0	0	4.6	4.6	4.6	4.6
Oxidant	None	H2O2	Air	Oxygen	Air	Oxygen	H2O2	H2O2	Oxygen	Air	H2O2	H2O2
Oxidant Flow (v/v/min)	-	-	0.25	0.25	0.25	0.25	-	-	0.1 - 0.165	0.1 - 0.166	-	-
Stoich. H2O2 Added (%)	-	163%	-	-	-	-	493%	354%	-	-	107%	110% to 140%
Temperature (°C)	60	60	60	60	60 to 40	60 to 40	60	60	60	60	60	60
Retention Time (h)1	1.0	1.0	3	3	5	3	2	2	6	6	2	1
Final Pulp pH	6.40	6.45	4.67	4.66	4.59	4.61	4.61	4.61	4.58	4.63	4.58	4.68
Tech. Reagent Addition (kg/m3 PLS)	13.0	11.7	5.8	6.3	3.4	3.2	5.0	2.3	2.4	2.0	1.2	1.4
kg CaO equiv. / m3 PLS	9.7	8.7	4.3	4.7	2.5	2.4	3.7	1.7	1.8	1.5	0.9	1.0
kg CaO equiv. / t H2SO4 neut.	16432	14721	7335	7954	4260	4015	6243	2955	1650	1377	1466	1734
kg CaO equiv. / t Fe removed	8563	7273	2853	3162	3844	1671	1530	1009	5305	7355	2007	1957
kg CaO equiv. / t Co Conc.	77	69	34	37	20	19	29	14	14	12	7	8
kg CaO equiv. / t Blend POX Feed	70	63	31	34	18	17	27	13	13	11	6	7

Table 53 Secondary neutralization efficiencies

Precipitation (%)	SN1	SN2	SN3	SN4	SN5	SN6	SN7	SN8	SN9	SN10	SN11	SN12
Bi	32	32	30	29	18	23	7	16	6	4	2	2
Co	83	84	3	4	3	6	1	1	3	2	0	1
Cu	100	100	60	70	85	71	59	40	73	62	60	67
Fe	100	100	95	99	23	45	100	100	30	18	99.9	99.8
Mg	6	5	1	1	0	1	0	0	0	0	0	0
Al	100	100	100	100	96	96	82	92	94	93	98	98
Cr	92	98	84	98	96	99	89	95	86	80	60	70
Mn	28	40	1	0	0	0	30	19	0	0	5	5
Ca	87	86	86	86	74	79	49	68	45	35	8	17
Na	6	7	7	15	7	10	2	2	2	3	1	1
As	24	28	69	67	10	20	1	3	1	1	0	0
S	-	-	-	-	-	-	-	-	-	-	-	-
S=	-	-	-	-	-	-	-	-	-	-	-	-
S0	-	-	-	-	-	-	-	-	-	-	-	-
Au	-	-	-	-	-	-	-	-	-	-	-	-
Ag	37	37	35	34	22	27	9	19	7	5	2	3
Pb	19	19	18	17	10	13	33	9	3	2	1	1
Sb	42	32	10	9	10	13	2	9	3	2	1	1
Sn	19	19	18	17	10	13	4	9	3	2	1	1
Fe (II)	-	-	0	0	0	0	-	-	-	-	-	-

Table 54 Secondary neutralization final solutions

Final PLS (mg/L)	SN1	SN2	SN3	SN4	SN5	SN6	SN7	SN8	SN9	SN10	SN11	SN12
Bi	1	1	1	1	1	1	1	1	1	1	1	1
Co	749	750	5100	5070	5060	5100	5060	5030	2120	2070	2140	2010
Cu	0	0	122	89	0	0	0	0	0	1	0	0
Fe	0	0	73	10	2260	1730	0	0	807	961	0.2	0.8
Mg	891	898	1210	1240	971	1300	987	1280	256	246	267	259
Al	0	1	3	3	30	34	3	3	11	11	0	0
Cr	0	0	0	0	0	0	0	0	0	0	0	0
Mn	44	37	79	80	68	76	46	61	33	31	30	31
Ca	496	502	494	502	520	552	528	527	523	485	535	511
Na	67	66	47	47	59	48	70	41	19	20	20	20
As	3	3	3	3	3	3	3	3	3	3	3	3
S	0	0	0	0	0	0	0	0	0	0	0	0
S=	0	0	0	0	0	0	0	0	0	0	0	0
S0	0	0	0	0	0	0	0	0	0	0	0	0
Au	0	0	0	0	0	0	0	0	0	0	0	0
Ag	0	0	0	0	0	0	0	0	0	0	0	0
Pb	2	2	2	2	2	2	2	2	2	2	2	2
Sb	1	1	2	2	1	1	2	1	1	1	1	1
Sn	2	2	2	2	2	2	2	2	2	2	2	2
Fe (II)	0	0	73	10	2260	1710	0	0	0	0	0	0

Table 55 Secondary neutralization final residues

Final Solids (g/t)	SN1	SN2	SN3	SN4	SN5	SN6	SN7	SN8	SN9	SN10	SN11	SN12
Bi	20	20	20	20	20	20	20	20	20	20	20	20
Co	104000	109000	6240	10900	13800	20000	7090	7190	20100	18700	5690	7540
Cu	6780	6560	8420	10100	53	32	37	7	353	398	195	174
Fe	30300	30700	70500	73500	60700	95200	192000	170000	113000	104000	322000	316000
Mg	1370	1140	455	491	398	484	85	103	379	444	50	42
Al	23000	22900	52800	53600	67600	52800	2480	3270	55300	75800	11500	9570
Cr	46	200	24	200	200	603	200	200	200	200	200	200
Mn	449	666	25	16	14	16	3000	1440	19	17	1270	955
Ca	151000	138000	141000	150000	132000	133000	136000	119000	139000	131000	60400	93100
Na	100	129	157	420	394	338	167	100	134	307	102	100
As	40	50	300	300	30	50	10	10	10	10	10	10
S (%)	0	0	0	0	0	0	0	0	0	0	0	0
S= (%)	0	0	0	0	0	0	0	0	0	0	0	0
S0 (%)	0	0	0	0	0	0	0	0	0	0	0	0
Au	0	0	0	0	0	0	0	0	0	0	0	0
Ag	2	2	2	2	2	2	2	2	2	2	2	2
Pb	20	20	20	20	20	20	253	20	20	20	20	20
Sb	30	20	10	10	10	10	10	10	10	10	10	10
Sn	20	20	20	20	20	20	20	20	20	20	20	20

Table 56 Secondary neutralization feed solutions

Feed solutions.(mg/L)	SN1	SN2	SN3	SN4	SN5	SN6	SN7	SN8	SN9	SN10	SN11	SN12
Bi	1	1	1	1	<1	<1	<1	<1	<1	<1	<1	<1
Co	5400	5400	5310	5310	5700	5390	5060	5100	2090	2090	2120	2120
Cu	279	279	286	286	0.3	0.3	0.1	0.2	3.0	3.0	0.4	0.4
Fe	1270	1270	1330	1330	3250	3150	2260	1730	1310	1310	807	807
Mg	1040	1040	1210	1210	994	1260	971	1300	237	237	256	256
Al	919	919	956	956	809	802	30	34	275	275	11	11
Cr	2	2	0	0	2	11	0	0	0	0	0	0
Mn	70	70	80	80	68	77	68	76	31	31	33	33
Ca	569	569	470	470	580	561	520	552	475	475	523	523
Na	64	64	46	46	62	75	59	48	21	21	19	19
As	3	3	5	5	3	3	3	3	3	3	3	3
S	0	0	0	0	0	0	0	0	0	0	0	0
S=	0	0	0	0	0	0	0	0	0	0	0	0
S0	0	0	0	0	0	0	0	0	0	0	0	0
Au	0	0	0	0	0	0	0	0	0	0	0	0
Ag	0	0	0	0	<0.08	<0.08	<0.08	<0.08	<0.08	<0.08	<0.08	<0.08
Pb	2	2	2	2	<2	<2	<2	<2	<2	<2	<2	<2
Sb	1	1	2	2	<1	<1	<1	<1	<1	<1	<1	<1
Sn	2	2	2	2	<2	<2	<2	<2	<2	<2	<2	<2
Fe (II)	-	-	1110	1110	3250	3150	2260	1710	0	0	0	0

Table 57 Second stage Iron Arsenic Removal - HDS Thickening 2017 Work – in lieu of SN

Second stage Iron Arsenic Removal design criteria - HDS Thickening 2017 Work							
Undiluted thickener feed		Diluted thickener feed		Specific Unit Area m ² /(t/day)	Net loadings		Underflow Pulp kg/L
Pulp kg/L	Solids % wt.	Pulp kg/L	Solids % wt.		Solids t/m ² /h	Hydraulic m ³ /m ² /h	
1.015	1.8	1.006	0.7	2.2	0.019	2.47	1.11
Particle size: K80 = 17 microns, Actual Solids SG = 3.95, Solution SG = 1.0, pH ~ 4.4Temp.: 75 °C.							
Maximum HDS predicted underflow solids content 11% at 17 Pa yield stress by vane.							
Flocculant: BASF Magnafloc 333 high molecular weight, nonionic polyacrylamide dosed at 45 g/t as 0.25 g/L stock solution.							
"As produced" bulk discharge feed predicted solids content ~ 0.9% wt., to be thickened by recycling the underflow ("HDS") then autodiluted in the feedwell to about 0.7% wt. solids.							
CCRV CSD 5.6% solids, corresponding to ~ 36 Pa unsheared and 10 Pa sheared yield stress.							
Residence times, hours							
Bed Height mm	Versus the entire equipment volume				³ Solids phase	Versus OF and UF volumes	
	¹ Feed slurry		² Liquid phase			⁴ Liquid vs. OF	
	Undiluted	Diluted	Undiluted	Diluted		Undiluted	Diluted
150.0	0.28	0.11	0.28	0.11	92.49	0.13	0.05
All values calculated without safety factor.							
¹ The actual volume of the test-unit divided by the undiluted and diluted feed slurry flowrates.							
² The actual volume of the test-unit divided by the undiluted and diluted liquid phase flowrates.							
³ The actual volume of the test-unit divided by the dry solids volumetric flowrate estimated based on SG.							
⁴ The actual volume occupied by the overflow divided by the undiluted and diluted liquid phase flowrates.							
⁵ The actual volume of the underflow divided by the dry solids volumetric flowrate estimated based on SG.							

Table 58 Second stage Iron Arsenic Removal design criteria - Dewatering 2017 Work – in lieu of SN

Second stage Iron Arsenic Removal design criteria - Dewatering 2017 Work		
Particle Size, d Pulp Temperature: 75°C		Pulp pH: 4.4
Solids SG: 3.95 Solution SG: 1.0		Unit: Pannevis
Vacuum Filter cloth: Testori P 6125 CQ Material: polypropylene, Permeability: 10.25 cfm, Warp/Weft: Mono/Mono, Weave: Two Plies, Finish: calendered, Weight: 435 g/m ²		
Pressure Filter cloth: Micronics 861, polypropylene; warp/weft: multi/multi; weave: twill, finish: calendered, permeability: 0.70 cfm, 661.24 g/m ²		
Filter Aid, Type/Dosage, g/t: Magnafloc 333 at 65 g/t (contained in the underflow)		
Actual cake thickness, m, H	17	15
Form time, h	0.05	0.031
Dry time, h	0.01	0.021
From/Dry	5	1.51
Filtrate TSS, mg/L	60	<10
Wet cake solids, % S	25.8	34.7
Wet cake density, kg/L, WD = CW/V	1.23	1.29
Dry cake density, kg/L, DD=WD*S/100	0.32	0.45
Method	Vacuum, Hg"	Pressure
Feed / form pressure, bar	0.67	6.9
Dry / form pressure, bar	0.56	6.9
Feed solids, %		14.00
Final wet cake moisture, %	74.2	65
Dry solids capacity, kg/m ² h, KS=CD/(A*T)	95	129
Filter capacity, L/m ² h, KL=L/T	589	802
* From raw test data / not for sizing any particular type of filter.		
¹ Throughput is calculated using cycle time which includes form and dry times only.		
Conversion: 0.098 bar = 2.896" Hg = 1 mH ₂ O = 3.28 ft H ₂ O = 1.42psi		
Refer to note for users (test details) for suggested steps to convert data into equipment sizing criteria.		

Polishing test work conditions and results

Polishing test work aimed to investigate the maximum removal of the trace copper left over from cementation, critical for the purity of the final cobalt hepta product. The reason for running these tests was as a redundancy for the situation whereby the copper cementation could not produce copper levels below 1 mg/L Cu.

Copper polishing test conditions

Samples were collected from both PSN tests for liquor analysis, and the pulps were combined and sent to LSR. At the very low indicated % solids, they were only able to do some limited floc scoping work. No assays could be performed with these quantities. Quicklime used as reagent. The test-feed samples consisted of secondary (oxidative) neutralization generated feed solutions.

PSN1 – neutralization of SN3 liquor to either pH 6.5 or ~100% stoichiometry of lime addition relative to SN3 feed Cu tenor.

PSN2 – neutralization of SN4 liquor to the same conditions as used in PSN1

Detailed results of the polishing test work conditions and results will be included in the formal SGS report.

Polishing test results

The test work resulted in copper tenors of 7 mg/L Cu (PSN1) and 7.7 mg/L Cu (PSN2).

Of note, the 2017 polishing performed following the manganese removal produced more substantial results. While chemically effective, the polishing discharge appeared impractical to thicken and filter: 4% wt. underflow requiring 3 m²/(t/day) specific thickener unit area, and 18 kg/m²h dry solids capacity at 9.9 bar pressure, for about 84% wt. moisture content cake. These results are applicable if needed but preferably by recycling the thickener underflow to the actual process (“seeding”) to allow for increased solids content in the filtration feed. Centrifugation could be considered as a last resort option due to its cost (capex primarily). However, the results of the copper cementation, with residual copper tenors well below 1 mg/L Cu under the right conditions negated the need for the polishing. As a result, no design criteria was issued for polishing.

Gold recovery and environmental test work conditions and results

Main test items consisted of gold recovery by carbon in leach (“CIL”) from the residue produced by combined POX/primary neutralization instead of the POX-leach residue on its own. This is necessary to validate that the gold recovery is not affected. Cyanide destruction, static settling, thickening, and filtration of the gold recovery residue aimed for final disposal were performed to validate the entire design criteria. The difference between the two primary neutralization samples was that PN1 was conducted with limestone, and PN4 with lime.

Secondary test item: intensive cyanidation test was performed on the bismuth leach residue on its own. This was necessary to ascertain gold recovery for the situation of a temporary shutdown of the pressure oxidation autoclave.

Gold recovery test conditions

Intensive cyanidation of the bismuth ferric leach residue: ambient temperature, no carbon, 72 hours duration with kinetics, 20 g/L NaCN maintained, pH 10.5 -11, maintained, 20-25 ppm dissolved oxygen by aeration. Classic ambient temperature carbon in leach (“CIL”) bottle roll tests on all other samples: 24 hours duration, 1 g/L NaCN , maintained, pH 10-10.5, 33% wt. Solids, 10 g/L Carbon, no kinetics.

Gold recovery test results

Intensive cyanidation on the bismuth ferric leach residue produced 91% and 95% gold recovery after 4 and 48 hours, respectively. The 48-hours specific consumptions were 99.5 kg/t NaCN and 8.9 kg/t CaO.

Carbon in leach cyanidation on the pressure oxidation discharge residue produced 98.5% gold recovery after 24 hours, with 98.1% adsorbed onto carbon at 4.425 kg/t Au. The specific consumptions were 0.64 kg/t NaCN and 11.6 kg/t CaO. Carbon in leach cyanidation on the PN1 discharge residue (limestone) produced 97.3% gold recovery after 24 hours, with 96.9% adsorbed onto carbon at 2.605 kg/t Au. The specific consumptions were 1.31 kg/t NaCN and 23.2 kg/t CaO. Carbon in leach cyanidation on the PN 4 discharge residue (lime) produced 97.5% gold recovery after 24 hours, with 97.1 % adsorbed onto carbon at 2.705 kg/t Au. The specific consumptions were 1.14 kg/t NaCN and 20.6 kg/t CaO. Overall, the recovery differential between the pressure oxidation discharge and the limestone and lime neutralized residues were within about one percent.

The test work included static settling, thickening and filtration of the gold recovery residue after cyanide destruction. The results are summarized below. They indicated insignificant differences in terms of preneutralization agent used, allowing them to establish a common design criterion (listed separately).

Thickening results are conservative due to the static testing. They can be improved by recycling some of the thickener underflow to the feedwell, to allow for increased solids content in the filtration feed (“short-route recycling”). After dewatering, the final moisture content should be amenable to extrapolation to a squeeze-pressure-filter target. Centrifugation could be considered as a last resort option due to its cost (capex primarily). The results were used as design criteria (listed in the dedicated section).

Table 59 Gold recovery summary – from Bi leach residue and primary neutralization residues

Test ID		CN1	CN2	CN3	CN4
Feed		BiL6 Residue	POX4	PN1/CaCO ₃	PN4/Ca(OH) ₂
Initial Pulp Density (% Solids)		10	33	33	33
Target NaCN (g/L)		20	1	1	1
Carbon Concentration (g/L)		N/A	10	10	10
pH Range		10.5-11.0	10.5-11	10.5-11	10.5-11
Preconditioning Time		-	1h	1h	1h
Cyanidation Time		72h	24h	24h	24h
Oxygen Applied		Yes	No	No	No
Average DO (mg/L)		21.5	-	-	-
NaCN Addition (kg/t)		258.4	2.14	2.58	2.45
NaCN Consumption (kg/t)		99.5	0.64	1.31	1.14
CaO Addition (kg/t)		10.2	11.7	23.2	20.7
CaO Consumption (kg/t)		8.88	11.6	23.2	20.6
Final Free CN (mg/L)		9098	389	324	329
Gold	Final Residue (g/t)	40.0	1.4	1.5	1.5
	Calc. Head (g/t)	814.3	94.2	55.7	58.0
	Extraction (%)	95.1	98.5	97.3	97.5
Loaded Carbon Assay (g/t Au)		-	4425	2605	2705
Consumptions expressed vs. initial POX Leach Feed					
CaO Consumption kg/t POX leach feed		4.8	11.3	33.0	29.6
NaCN Consumption (kg/t) POX leach feed		54.0	0.6	1.9	1.6
Intensive cyanidation CN 1: no carbon, 72 hours, kinetics, 20 g/L NaCN, pH 10.5 -11, 20-25 ppm DO. CIL tests CN2 through CN4: 24 hours, 1 g/L NaCN , pH 10-10.5, 10 g/L Carbon, no kinetics.					
Weigh losses: Bi leaching: 45.7%, POX: 2.8%. Gains vs. POX feed: PN1: 142% and PN 4: 144%.					
CIL feed samples PN1 and PN4 consisted of pre-neutralization of the entire ACD slurry, using CaCO ₃ and Ca(OH) ₂ , respectively.					

Table 60 Summary of Static Settling Tests – gold plant discharge

Settling Test Number	8	9
Sample Identification	CND-1 Pulp (CaCO₃)	CND-2 Pulp (Ca(OH)₂)
Raked, Yes/No	Yes	Yes
Particle Size, d80, microns	15	17
Dry Solids Specific Gravity, <->	3.00	2.89
Pulp pH	8.3	8.5
Pulp Temperature, °C	20	20
Solution Density, kg/L	1.000	1.000
Process Discharge Solid Content, % w/w	33.0	33.0
Thickener Feed Solid Content, % w/w	5.0	5.0
BASF Magnafloc 10, g/t dry	50	30
U/F Solids Density, % w/w	38	48
Thickener Unit Area, m ² /(t/day)	0.25	0.12
Thickener Hydraulic Unit Area, m ² /(m ³ /day)	0.04	0.02
Initial Settling Rate, m ³ /m ² /day	493	889
Supernatant Clarity, 5 minutes	Clear	Hazy
TSS, 5 minutes, mg/L	11	39
Supernatant Clarity, 30 minutes	Clear	Hazy
Supernatant Clarity, Final	Clear	Clear
All values calculated without safety factor.		
BASF Magnafloc 10 is a very high molecular weight slightly anionic polyacrylamide flocculant.		

Table 61 Pressure Filtration Summary - CND-1 Underflow (CaCO₃) - 6.9 bar

Pressure Filtration Summary - CND-1 Underflow (CaCO₃) - 6.9 bar			
Project No.: 11758-11	Date: April 28, 2025		
Ref.: Fortune			
Sample I.D.: CND-1 Underflow			
Particle Size, d80: 14.5 µm	Pulp Temperature, °C:	20	
Pulp pH: 8.3			
Filter cloth: Testori P 6583 TC, Material: polypropylene, Permeability: 1.23 cfm, Pore size: 8 µm, Weave: satin, Warp/Weft: multi/spun yarn, Finish: heat-set/ calendered			
Filter Aid, Type/Dosage, g/t: None			
Test ID	611	610	609
¹ Actual cake thickness, m, H	0.0250	0.0200	0.0150
Filter area m ² , A = (PI/4)*D ²	0.0020	0.0020	0.0020
Actual cake volume, L, V=A*H	0.049	0.039	0.029
Cake wet weight, kg, CW	0.067	0.052	0.037
Cake dry weight, kg, CD	0.044	0.034	0.025
Wet cake solids,%, S	65.6	66.7	68.5
Wet cake density, kg/L, WD = CW/V	1.367	1.314	1.250
Dry cake density, kg/L, DD=WD*CS/100	0.897	0.876	0.856
Filtrate volume, L, V	0.041	0.034	0.025
Filter loading, L/m ² , L= (Volume/Area)	45.8	37.4	27.6
Form time, h	0.0030	0.0022	0.0013
Dry time, h	0.0219	0.0156	0.0112
Form/Dry	0.1379	0.1395	0.1153
Total feeding time, h, T	0.0249	0.0177	0.0124
Filtrate TSS, mg/L	31	41	36
*Example of general filter throughput predictions versus test conditions			
Feed Pressure, bar	6.9		
Feed solids, %	40.0		
Final wet cake moisture, %	34.4	33.3	31.5
² Dry solids capacity, kg/m ² h, KS=CD/(A*T)	901	989	1032
³ Dry solids capacity, full cycle time, kg/m ² h	117	95	72
Filter capacity, L/m ² h, KL=L/T	1839	2109	2220
* From raw test data / not for sizing any particular type of filter.			
¹ cake thickness is equivalent to half of the filter chamber thickness			
² Throughput (form and dry): including form and dry time only.			

Table 62 Pressure Filtration Summary - CND-1 Underflow (CaCO₃) – 5.5 bar

Pressure Filtration Summary - CND-1 Underflow (CaCO₃) - 5.5 bar			
Project No.: 11758-11	Date: April 28, 2025		
Ref.: Fortune			
Sample I.D.: CND-1 Underflow			
Particle Size, d80: 14.5 µm	Pulp Temperature, °C:		20
Pulp pH: 8.3			
Filter cloth: Testori P 6583 TC, Material: polypropylene, Permeability: 1.23 cfm, Pore size: 8 µm, Weave: satin, Warp/Weft: multi/spun yarn, Finish: heat-set/ calendered			
Filter Aid, Type/Dosage, g/t: None			
Test ID	614	613	612
¹ Actual cake thickness, m, H	0.0260	0.0210	0.0150
Filter area m ² , A = (PI/4)*D ²	0.0020	0.0020	0.0020
Actual cake volume, L, V=A*H	0.051	0.041	0.029
Cake wet weight, kg, CW	0.072	0.058	0.041
Cake dry weight, kg, CD	0.046	0.037	0.027
Wet cake solids, % S	63.6	63.5	64.5
Wet cake density, kg/L, WD = CW/V	1.409	1.414	1.395
Dry cake density, kg/L, DD=WD*CS/100	0.895	0.898	0.900
Filtrate volume, L, V	0.041	0.033	0.024
Filter loading, L/m ² , L= (Volume/Area)	47.0	37.9	27.1
Form time, h	0.0043	0.0030	0.0018
Dry time, h	0.0279	0.0216	0.0157
Form/Dry	0.1524	0.1394	0.1147
Total feeding time, h, T	0.0322	0.0247	0.0175
Filtrate TSS, mg/L	49	116	52
*Example of general filter throughput predictions versus test conditions			
Feed Pressure, bar	5.5		
Feed solids, %	40.0		
Final wet cake moisture, %	36.4	36.5	35.5
² Dry solids capacity, kg/m ² h, KS=CD/(A*T)	724	765	773
³ Dry solids capacity, full cycle time, kg/m ² h	117	99	73
Filter capacity, L/m ² h, KL=L/T	1460	1536	1549
* From raw test data / not for sizing any particular type of filter.			
¹ cake thickness is equivalent to half of the filter chamber thickness			
² Throughput (form and dry): including form and dry time only.			

Table 63 Pressure Filtration Summary - CND-2 Underflow (Ca(OH)₂) - 6.9 bar

Pressure Filtration Summary - CND-2 Underflow (Ca(OH)₂) - 6.9 bar			
Project No.: 11758-11	Date: April 25, 2025		
Ref.: Fortune			
Sample I.D.: CND-2 Underflow			
Particle Size, d₈₀: 17 µm	Pulp Temperature, °C: 20		
Pulp pH: 8.3			
Filter cloth: Testori P 6583 TC, Material: polypropylene, Permeability: 1.23 cfm, Pore size: 8 µm, Weave: satin, Warp/Weft: multi/spun yarn, Finish: heat-set/ calendered			
Filter Aid, Type/Dosage, g/t: None			
Test ID	605	604	603
¹ Actual cake thickness, m, H	0.0250	0.0200	0.0150
Filter area m ² , A = (PI/4)*D ²	0.0020	0.0020	0.0020
Actual cake volume, L, V=A*H	0.049	0.039	0.029
Cake wet weight, kg, CW	0.085	0.067	0.049
Cake dry weight, kg, CD	0.058	0.046	0.034
Wet cake solids,% S	67.5	68.6	69.7
Wet cake density, kg/L, WD = CW/V	1.741	1.703	1.666
Dry cake density, kg/L, DD=WD*CS/100	1.176	1.169	1.161
Filtrate volume, L, V	0.040	0.033	0.026
Filter loading, L/m ² , L= (Volume/Area)	45.6	36.7	28.2
Form time, h	0.0128	0.0087	0.0049
Dry time, h	0.0188	0.0161	0.0136
Form/Dry	0.6810	0.5388	0.3577
Total feeding time, h, T	0.0317	0.0248	0.0185
Filtrate TSS, mg/L	34	48	56
*Example of general filter throughput predictions versus test conditions			
Feed Pressure, bar	6.9		
Feed solids, %	45.0		
Final wet cake moisture, %	32.5	31.4	30.3
² Dry solids capacity, kg/m ² h, KS=CD/(A*T)	929	943	941
³ Dry solids capacity, full cycle time, kg/m ² h	148	122	94
Filter capacity, L/m ² h, KL=L/T	1441	1479	1521
* From raw test data / not for sizing any particular type of filter.			
¹ cake thickness is equivalent to half of the filter chamber thickness			
² Throughput (form and dry): including form and dry time only.			

Table 64 Pressure Filtration Summary - CND-2 Underflow (Ca(OH)₂) – 5.5 bar

Pressure Filtration Summary - CND-2 Underflow (Ca(OH)₂) - 5.5 bar			
Project No.: 11758-11	Date: April 25, 2025		
Ref.: Fortune			
Sample I.D.: CND-2 Underflow			
Particle Size, d₈₀: 17 µm	Pulp Temperature, °C:		20
Pulp pH: 8.3			
Filter cloth: Testori P 6583 TC, Material: polypropylene, Permeability: 1.23 cfm, Pore size: 8 µm, Weave: satin, Warp/Weft: multi/spun yarn, Finish: heat-set/ calendered			
Filter Aid, Type/Dosage, g/t: None			
Test ID	608	607	606
¹ Actual cake thickness, m, H	0.0260	0.0200	0.0150
Filter area m ² , A = (PI/4)*D ²	0.0020	0.0020	0.0020
Actual cake volume, L, V=A*H	0.051	0.039	0.029
Cake wet weight, kg, CW	0.089	0.069	0.050
Cake dry weight, kg, CD	0.059	0.046	0.033
Wet cake solids,% S	66.7	66.3	66.9
Wet cake density, kg/L, WD = CW/V	1.741	1.747	1.694
Dry cake density, kg/L, DD=WD*CS/100	1.162	1.159	1.134
Filtrate volume, L, V	0.041	0.031	0.023
Filter loading, L/m ² , L= (Volume/Area)	46.7	35.7	26.8
Form time, h	0.0143	0.0105	0.0060
Dry time, h	0.0177	0.0136	0.0103
Form/Dry	0.8061	0.7734	0.5834
Total feeding time, h, T	0.0321	0.0241	0.0163
Filtrate TSS, mg/L	38	10	86
*Example of general filter throughput predictions versus test conditions			
Feed Pressure, bar	5.5		
Feed solids, %	45.0		
Final wet cake moisture, %	33.3	33.7	33.1
² Dry solids capacity, kg/m ² h, KS=CD/(A*T)	942	961	1042
³ Dry solids capacity, full cycle time, kg/m ² h	152	121	93
Filter capacity, L/m ² h, KL=L/T	1458	1481	1644
* From raw test data / not for sizing any particular type of filter.			
¹ cake thickness is equivalent to half of the filter chamber thickness			
² Throughput (form and dry): including form and dry time only.			

Gold recovery from tailings and environmental test work conditions and results

The test work included carbon in leach (“CIL”) bottle roll tests. According to historic data (2007 Pilot) from about 4 to about 15% of the overall gold reported to the bulk scavenger tailings. According to the 2024 mini- pilot data, about 6% of the overall gold reported to the clean scavenger tailings.

Gold recovery test conditions

Classic ambient temperature carbon in leach (“CIL”) bottle roll tests of 48 hours duration, 0.5 g/L NaCN , maintained, pH 10-10.5, 33% wt. Solids, 10 g/L Carbon, no kinetics at about 3% mass pull.

Gold recovery test results

Gold extractions insensitive to aging, averaging ~65% for the bulk rougher tailings and 83% for the clean scavenger tailings. These findings were consistent with the general consensus whereby gold extractions are directly proportional to the head grades (averaging ~1.15 g/t Au bulk rougher, 5.6 g/t Au bulk scavenger), at least for the 48 hour duration leaches.

Discharge residue grades consistent with previous work as well, ~0.4 g/t Au bulk rougher, 0.9 g/t Au bulk scavenger).

Table 65 Table Gold recovery summary – fresh and aged tailings

Sample	Test #	Reagent Addition		Reagent Consumption		Extraction, Au, %
		NaCN, kg/t	CaO, kg/t	NaCN, kg/t	CaO, kg/t	
PP109 Ro Tailing (Fresh)	CN1	1.05	0.72	0.15	0.64	64.6
PP109 Ro Tailing (Aged)	CN3	1.04	0.51	0.15	0.44	64.4
PP109 Scav Tailing (Fresh)	CN2	1.16	2.02	0.36	1.97	81.4
PP109 Scav Tailing (Aged)	CN4	1.30	1.56	0.41	1.50	85.3
Sample	Carbon Au, g/t	Residue Au, g/t		Calculated Head Au, g/t		
-		Au, g/t A	Au, g/t B	Au, g/t Avg		
PP109 Ro Tailing (Fresh)	35.6	0.41	0.41	0.41	1.16	
PP109 Ro Tailing (Aged)	34.1	0.40	0.39	0.40	1.11	
PP109 Scav Tailing (Fresh)	216	1.01	1.06	1.04	5.56	
PP109 Scav Tailing (Aged)	229	0.82	0.82	0.82	5.56	
Ambient temperature carbon in leach (“CIL”) bottle roll tests of 48 hours duration, 0.5 g/L NaCN, maintained, pH 10.5-11, maintained, 33% wt. solids, 10 g/L Carbon, no kinetics.						

Gold discharge streams environmental analyses results

TCLP results were produced by recent comparative tests on CND1 and CND2 residues generated by preneutralization of the entire (unseparated) autoclave discharge slurry with limestone and lime, respectively, followed by cyanidation for gold recovery and final cyanide destruction. The TCLP results were also produced on 2024 mini-pilot CN4 residue generated by cyanidation of the cleaner scavenger tailings.

The Ontario Schedule 4 (Canadian) limits are given in each report and show that all samples are not leachate toxic. Modified acid base accounting tests (ABA) results in duplicate on the above three samples indicated that all displayed negative Net Neutralization Potentials (NNPs), implying that they are Potentially Acid Generating (PAG).

Table 66 TCLP results summary

Parameter	Unit	US RCRA Limits	Ontario Schedule 4 Limits	CND1 Residue CA19873-APR25	CND2 Residue CA19873-APR25	CN4 Cleaner tails
LIMS	<->			19873-APR25	19873-APR25	19004-MAY25
Final pH	<->			5.09	5.12	5.17
Hg	mg/L	0.2	0.1	< 0.00001	< 0.00001	< 0.00001
Ag	mg/L	5.0	5.0	< 0.00005	0.00009	0.00006
Al	mg/L			0.035	0.148	4.96
As	mg/L	5.0	2.5	0.691	2.05	1.92
Ba	mg/L	100.0	100.0	0.00979	0.017	0.212
Be	mg/L			0.000021	0.000032	0.000667
B	mg/L		500.0	0.047	0.576	0.235
Bi	mg/L			0.00318	0.118	0.0672
Ca	mg/L			949	980	183
Cd	mg/L	1.0	0.5	0.0006	0.00048	0.00034
Co	mg/L			0.208	0.172	0.737
Cr	mg/L	5.0	5.0	0.00117	0.00517	0.00173
Cu	mg/L			0.163	0.226	0.017
Fe	mg/L			0.086	2.44	14.7
K	mg/L			1.73	1.74	49
Li	mg/L			0.0033	0.003	0.0166
Mg	mg/L			2.98	2.63	9.37
Mn	mg/L			0.106	0.0446	1.83
Mo	mg/L			0.0097	0.0202	0.0014
Na	mg/L			1310	1390	1360
Ni	mg/L			0.0174	0.0159	0.0475
P	mg/L			0.02	0.02	0.02
Pb	mg/L	5.0	5.0	0.00606	0.00669	0.0313
Sb	mg/L			0.0788	0.0806	0.028
Se	mg/L	1.0	1.0	0.0337	0.0632	0.00677
S	mg/L			63.4	76.3	14.6
Sn	mg/L			0.00023	0.00028	0.00014
Sr	mg/L			8.12	0.458	0.164
Th	mg/L			< 0.0001	< 0.0001	0.0001
Ti	mg/L			0.0008	0.0153	0.0015
Tl	mg/L			0.0001	0.00008	0.00039
U	mg/L		10.0	0.00258	0.00306	0.115
V	mg/L			0.00015	0.00059	0.00019
W	mg/L			0.0002	0.0006	0.0003
Y	mg/L			0.00647	0.00723	0.00985
Zn	mg/L			0.029	0.021	0.081

Table 67 Modified ABA results summary

Parameter	Unit	CND1 Residue	CND2 Residue	CN4 Cleaner tails
LIMS 1	<->	19002-MAY25	19002-MAY25	19003-MAY25
LIMS 2	<->	14296-JUN25	14296-JUN25	14297-JUN25
Paste pH	<->	8.24	8.31	8.11
Fizz Rate	<->	2	2	2
Sample weight	g	2.0	2.0	2.03
HCl added	mL	20	20	24.8
HCl	Normality	0.1	0.1	0.1
NaOH	Normality	0.1	0.1	0.1
Vol NaOH to pH=8.3	mL	18.1	17.81	18.69
Final pH	no unit	1.74	1.7	1.7
NP	t CaCO ₃ /1000 t	4.7	5.5	15
AP	t CaCO ₃ /1000 t	54.1	62.6	15.1
Net NP	t CaCO ₃ /1000 t	-49.4	-57.09	-0.09
NP/AP	ratio	0.09	0.09	0.99
S	%	7.9	8.1	0.603
Acid Leachable SO ₄ -S	%	6.12	6.11	0.12
Insoluble SO ₄ as S	%	<0.02	<0.02	<0.02
Sulphide	%	1.73	2	0.48
C	%	0.197	0.191	0.108
CO ₃	%	0.15	0.04	0.4
CO ₃ NP	t CaCO ₃ /1000 t	2.49	0.664	6.64
Net CO ₃ NP	t CaCO ₃ /1000 t	-52	-62	-8
CO ₃ NP/AP	ratio	0.05	0.01	0.44

Cobalt sulphate heptahydrate test work conditions and results

Cobalt sulphate heptahydrate test work: manganese removal, cobalt solvent extraction and crystallization test works aiming to prove the possibility of producing the low sodium hepta product according to the client's specifications. The parameters were based on existing PDC package submitted in November 2024. Source: Metallurgical Process Opinion No. 3_ 241101 Unrevised R0 [5]. Feed to circuit: secondary neutralization discharge solution. Of note, all test feeds originated from actual NICO feed concentrates pressure oxidation feed.

Manganese removal test conditions and results

Conditions:

- Feed samples: SN5, SN 7 and SN 8 discharge liquors;
- temperature ~50°C, 4 hour total retention time with gas flow;

- gas flow is 2% SO2 in O2, running at 0.04 v/v/min;
- target pH of 4.5 using lime to control; and
- hourly kinetic liquor samples to track actual Mn removal.

Results

- SN5 feed too high in iron, results discarded (Test MnR1);
- tests MnR2 and MnR 3 identical, Test MnR3 bulk validation; and
- manganese removal efficiency approximatively 98%, leaving well below 1 mg/L Mn in the discharge solution.

Table 68 Key manganese removal test work results summary

MnR2 test results summary									
Sample & Quant.	Assay Units	SN7 Filtrate	1 h Sample Filtrate	2 h Sample Filtrate	3 h Sample Filtrate	4 h Sample Filtrate	4 h Sample Residue	Precipitation	Account.
(mL or g)		1500	64	60	45	1335	2	%	out/in %
Bi	mg/L, g/t	<1	<1	<1	<1	<1	<20	3	103
Co	mg/L, g/t	5060	4980	4930	4940	4760	94000	3	97
Cu	mg/L, g/t	<0.1	0.1	<0.1	<0.1	<0.1	145	68	313
Fe	mg/L, g/t	<0.2	<0.2	<0.2	<0.2	<0.2	1260	90	1024
Mg	mg/L, g/t	987	952	965	973	958	118	0	97
Al	mg/L, g/t	3.2	2.5	2.5	1.6	1.1	2330	73	145
Cr	mg/L, g/t	<0.1	<0.1	<0.1	<0.1	<0.1	<200	75	394
Mn	mg/L, g/t	46.4	22.3	1.68	0.49	0.15	27200	97	88
Ca	mg/L, g/t	528	564	625	612	561	172000	31	80
Na	mg/L, g/t	70	69	72	71	69	444	1	100
As	mg/L, g/t	<3	<3	<3	<3	<3	<30	1	102
Zn	mg/L, %	27.5	28.2	28.8	27.8	27.6	642	100	34341
Pb	mg/L, g/t	<2	<2	<2	<2	<2	<20	1	102
Sb	mg/L, g/t	<2	<1	<1	<1	<1	<10	1	51
Sn	mg/L, g/t	<2	<2	<2	<2	<2	<20	1	102
Bulk MnR4 test results summary									
Sample & Quant.	Assay Units	SN8 Filtrate			2 h Sample Filtrate	4 h Sample Filtrate	4 h Sample Residue	Precipitation	Account.
(mL or g)		9600			38	9735	23	%	out/in %
Bi	mg/L, g/t	<1			<1	<1	<20	5	107
Co	mg/L, g/t	5030			5040	4910	69800	3	103
Cu	mg/L, g/t	<0.1			<0.1	<0.1	<5	11	114
Fe	mg/L, g/t	<0.2			<0.2	<0.2	258	75	414
Mg	mg/L, g/t	1280			1370	1350	96	0	107
Al	mg/L, g/t	3			<3	<3	701	36	158
Cr	mg/L, g/t	<0.1			<0.1	<0.1	<200	83	585
Mn	mg/L, g/t	61.3			0.31	1.03	25600	98	103
Ca	mg/L, g/t	527			689	563	191000	45	96
Na	mg/L, g/t	41			48	47	106	1	117
As	mg/L, g/t	<3			<3	<3	<30	2	104
Zn	mg/L, %	17.1			15.4	15.5	238	99.7	33728
Pb	mg/L, g/t	<2			<2	<2	<20	2	104
Sb	mg/L, g/t	<1			<1	<1	<20	5	107
Sn	mg/L, g/t	<2			<2	<2	<20	2	104

Inaccurate account closing due to very low solids concentrations rejected. Solution values valid.

Solvent extraction test conditions and results

Extraction:

- 10-15% Cyanex 272 in diluent (we'll use Exxsol D80)
- 55°C, pH near 5-5.2 controlled with soda ash 10% Na₂CO₃; and
- A/O of 2/1.

Scrubbing and stripping:

- Scrubbing at 1/1 with pH controlled to 4-4.4 with sulfuric acid 10% H₂SO₄;
- stripping at pH 2.7-3.3, same reagent; and
- A/O of 1:1.

The main goal of the test work was to prove the possibility of producing adequate specification cobalt sulphate heptahydrate product exclusively from NICO concentrate. For this reason, there was no synthetic Co in the strip feed, which in turn limited the Co-extraction hence its overall recovery efficiencies below 90% in some of the tests instead of the > 99% already proven during the 2012 pilot campaign. Also, the Mg tenor in the strip was higher than it should be since the scrubbing could not be conducted under steady state continuous conditions with maximum cobalt loading in order to allow for the magnesium to be "pushed out by cobalt".

The analytical summary of the strip solution produced from the unpolished and Zn-polished manganese removal circuit discharge fed directly to solvent extraction are summarized comparatively in Table 69.

Qualitatively, the main positive difference brought by polishing was the reduction on Zn to <1 ppm compared to >4 mg/L Zn. While small values, these are key to meeting or missing the specifications.

On the downside, the Ca tenor increased from <4 mg/L to 47 mg/L since the polishing was done after SX. This fact allowed for ignoring the Ca assays in the final HEPTA produced from the polished stream. Same goes for the Na which increased from <4 mg/L to 41 mg/L. Since the Zn-polishing was done before SX in the validated process, these abnormal impurity build-ups have been flagged as non-representative.

Table 69 Strip solutions comparative analyses

Test	CoS 1-1 unpolished		CoS 2-1 unpolished		CoS 3-1 Zn-polished	
Species	mg/L	Co/M Ratio	mg/L	Co/M Ratio	mg/L	Co/M Ratio
Ag	<0.08	246250	<0.08	240000	<0.08	173750
Al	1.2	16417	2.4	8000	0.2	69500
As	<3	6567	<3	6400	<3	4633
Ba	0.02	985000	0.02	960000	0.02	695000
Be	0.05	394000	0.002	9600000	0.002	6950000
Bi	<1	19700	<1	19200	<1	13900
Ca	3.6	5472	3.3	5818	47	296
Cd	0.22	89545	0.2	96000	0.1	139000
Co	19700	1	19200	1	13900	1
Cr	<0.1	197000	<0.1	192000	<0.1	139000
Cu	0.3	65667	0.1	192000	0.2	69500
Fe	0.3	65667	1.8	10667	0.2	69500
K	<1	19700	<1	19200	<1	13900
Li	<2	9850	<2	9600	<2	6950
Mg	130	152	77.8	247	90.9	153
Mn	1.37	14380	4.62	4156	0.99	14040
Mo	<0.6	32833	<0.6	32000	<0.6	23167
Na	4	4925	2	9600	41	339
Ni	3.3	5970	3	6400	1	13900
P	6	3283	5	3840	10	1390
Pb	<2	9850	<2	9600	<2	6950
Sb	<3	6567	<3	6400	<3	4633
Se	<3	6567	<3	6400	<3	4633
Sn	<2	9850	<2	9600	<2	6950
Sr	0.01	1970000	0.007	2742857	0.034	408824
Ti	<0.2	98500	<0.1	320000	<0.0	347500
Tl	<3	6567	<3	6400	<3	4633
V	<0.2	98500	<0.2	96000	<0.2	69500
Y	<0.03	656667	<0.02	960000	<0.02	695000
Zn	8.7	2264	4.3	4465	0.7	19857

Evaporation – crystallization test conditions and results:

- The combined strip liquor was simmered until the total volume reduced to target ~150 g/L Co or near;
- allowed natural cooling under agitation; and
- vacuum filtered, acetone washed on filter, air dried and submitted for analyses.

While the purity of the cobalt sulphate heptahydrate produced from the unpolished stream was 99.51% (20.86% Co), exceeding the battery grade minimum of 97.81% (20.5% Co), and beating the low-sodium (<20 ppm Na vs. 20-50 ppm Na), sum of heavies (As/Cd/Pb <3 ppm vs. 5-10 ppm). However, it failed the Zn (37 ppm Zn > 10 ppm Zn).

Fixing required polishing implying a loss of as about >2.45% Co indefinitely, reason whereby it was intentionally avoided in the first place.

The analytical summary of the cobalt sulphate heptahydrate crystals produced from the polished stream are summarized comparatively versus battery grade specifications in Table 70.

The cobalt sulphate heptahydrate produced from the polished stream exceeded 99.99% (20.959% Co), slightly exceeding the battery grade minimum of 99.99% (20.958% Co). All heavy metal impurities assay results were either below or between the limits imposed by the battery grade specifications. Magnesium specifications will be met through the SX engineering optimization work as detailed further herein.

Of note, Mg limit exceeded due to the test conditions consisting of small batches. Magnesium should be reduced by maximizing cobalt loading, adding the third scrub stage and lowering the pH profile in the scrub stages. A comprehensive continuous pilot exercise is needed to optimize this aspect, while leaving room for final optimization during the startup of the industrial plant.

In conclusion, downstream solution purification results confirmed the validity of the established secondary neutralization sequence by producing a HEPTA feed which was subjected to manganese removal, solvent extraction followed by the evaporation - crystallization sequence. The design criteria remain valid but with the inclusion of the slight changes in the SX circuit, detailed in the next section. The data are suitable to be refined by SysCad modeling for the purpose of the feasibility study.

Table 70 HEPTA crystals analyses vs. specifications

Element	Assay unit	2025 Battery grade Specs	2025 Bench
		Specifications	Representative
Ni	g/t	<10–50	<6
Fe	g/t	<5–20	<1
Cu	g/t	<5–10	1
Mn	g/t	<10–50	25
*Zn	g/t	<5–10	<7
Ca	g/t	<10–20	4
*Mg	g/t	<5–10	n/a
Na	g/t	<20–50	<10
K	g/t	<10–20	<10
As	g/t	<5 (in heavy metals)	<1
Cd	g/t	<5 (in heavy metals)	<1
Pb	g/t	<5–10 (in heavy metals)	<1
As+Cd+Pb	g/t	<5–10	<3
	g/t	<100	53
Si	g/t	<5–10	<5
*Insol's	% wt.	<0.01	<0.01
*Co	% wt.	>20.958	20.959
	% wt.	>99.990	99.995

*Mg - engineering optimization work in progress.

Specific SX recommendations for battery grade HEPTA specifications

Important notes for the application of the data for SX modeling and design taken from the 2012 short pilot exercise are included in this section.

There appears to be room for further improvement in keeping magnesium out of the cobalt product. Firstly, as mentioned, the upper limits of cobalt loading must be found. Secondly, there may be some room for optimizing the pH profile in extraction by lowering the pH in E4. However, a more effective strategy may be to establish a pH profile in the scrub stages by adding pH modifier. In a longer campaign, this would have been done. Lower pH across the scrub stages may help displace more Mg back upstream with the scrub liquor. Addition of a third stage of Co scrubbing is another measure that may improve the Mg separation and help with some other trace impurities as well. A systematic program to adjust and optimize operating conditions in the production plant should enable these measures to be implemented as long as the plant has been designed to allow the recommended actions to be carried out, specifically pH control in scrubbing using acid and a third scrub stage.

Two scrub stages were used in the pilot plant circuit. A third stage is recommended. No pH control was used but it is recommended that 150 g/L H₂SO₄ be used in all three stages to establish a pH profile in the scrub stages. A starting point for the profile would be pH 4.4, 4.2, 4.1 in stages B1, B2 and B3 respectively. This profile needs to be optimized but much longer run-times are needed. Optimization can be done operating a continuous circuit pilot plant but more likely, it will need to be done in the industrial plant. The only accurate way to judge how effectively scrubbing is working will be to look at the strip liquor analyses under each set of conditions set in scrubbing and it is necessary to wait until the circuit stabilizes to any changes in scrubbing before meaningful data can be obtained. Mg is a good element with which to track scrubbing efficiency.

In designing the commercial circuit, it is important to take into consideration the volume of pH modifier that is required in stripping as this is of the same order of magnitude as the actual strip feed flow. It may be possible to reduce the dilution by pH modifier by switching to a stronger acid, greater than the 200 g/L used, but the pH must remain well-controlled as localized low pH in the mixers can lead to contamination of the strip liquor product when unwanted impurities the organic is holding are also inadvertently stripped off. It is also likely that there is room to increase the concentration of cobalt in the strip liquor product further by adjusting the advance flows (advance O/A ratio) and the pH modifier concentration, should this be desirable.

Two zinc strip stages were used in the plant. If the industrial plant also receives a feed that is essentially free of zinc, it is still recommended that two stages be used as this may allow for some segregation of zinc from the other impurities. There was insufficient operating time to quantify acid consumption in the zinc strip stages but this is expected to be very low. It may also be advisable to operate these stages at higher acidity than the normal pH for zinc stripping, simply to ensure that no other elements like iron build up on the organic. ZS2 could be operated at 200 g/L acid and the ZS2 liquor could be used to act as a pH modifier in ZS1 to keep the pH there at a target of 0.6-0.8. Alternatively, provision should be made to treat a very small bleed of the stripped (recycle) organic, perhaps 0.5% of the flow, with 200 g/L acid to ensure that no impurity build up would occur.

The major impurities in the strip liquor were manganese and magnesium with lesser amounts of calcium and nickel. Magnesium should be reduced by maximizing cobalt loading, adding the third scrub stage and lowering the pH profile in the scrub stages. Nickel and calcium should also be lowered by more efficient scrubbing.

As a result, cobalt sulphate heptahydrate purity validation triggered a slight change in the solvent extraction PDC – summarized in Draft Metallurgical Process Opinion No. 3_251114 Revision 3.

Flowsheet schematics and narratives summary

The entirety of all NICO project hydrometallurgical process flowsheet descriptions and associated schematics are summarized in this section as follows:

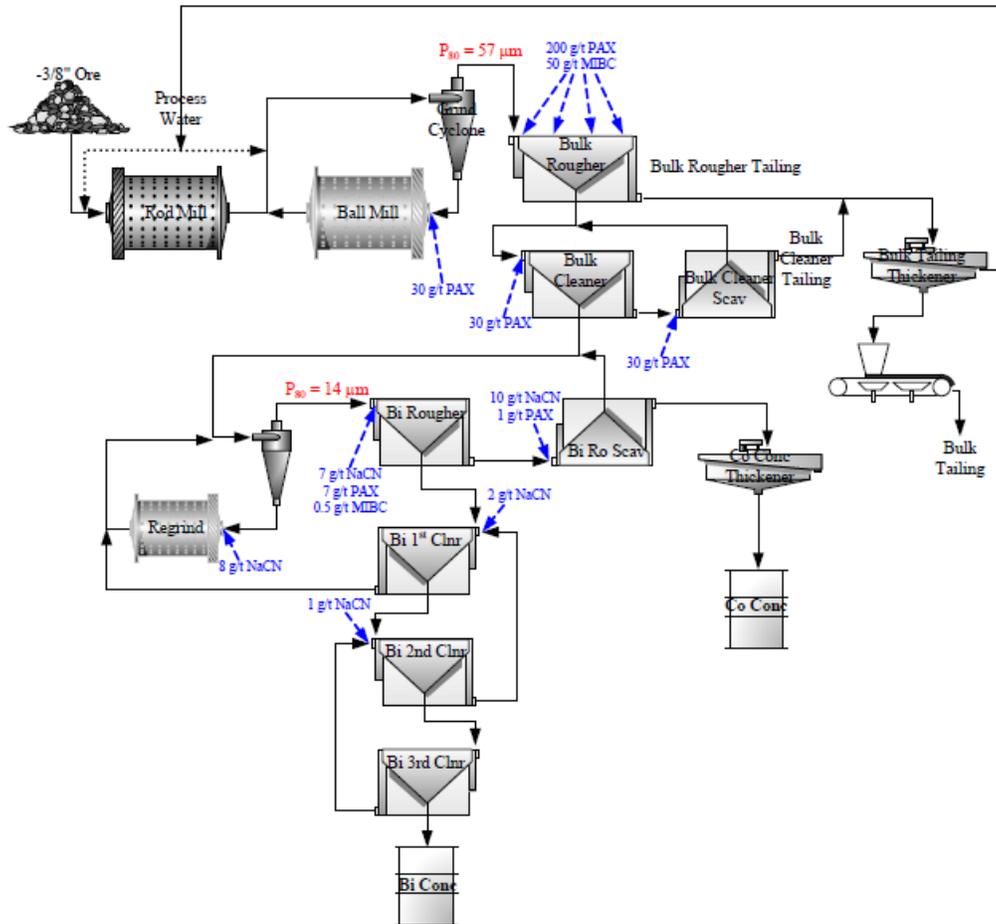
1. Mineral processing producing the selective cobalt and bismuth concentrates;
2. Pressure oxidation of the cobalt concentrate and bismuth leach residue blend, including purification;
3. Manganese removal from the purified pregnant leach solution;
4. Solvent extraction, evaporation and crystallization of the manganese-free discharge solution; and
5. Bismuth recovery from the bismuth concentrate.

Mineral processing flowsheet schematic process description

The feed is subjected to primary grinding targeting K80 ~ 70 microns, then reduced to 54 microns to increase bulk recoveries. The ground feed is subjected to bulk flotation which includes the recycling of the bulk cleaner scavenging stream. The flowsheet is depicted in Figure 30.

The bulk cleaner concentrate was subjected to reground to target K80 ~ 14 microns and subjected to selective flotation that recovered the bismuth and cobalt concentrates inclusive of the recycled scavenger and cleaner streams.

Figure 30 Mineral processing concentrator flowsheet schematic



Pressure oxidation and autoclave discharge flowsheet schematic process description

The pressure oxidation feed consisting of cobalt concentrate, bismuth residue and recycled cobalt sulphide streams is repulped at about 14% solids (as determined by sulphide content) and directed continuously into the autoclave. POX is carried out for 1-hour residence time at 180°C-process temperature, 759 kPa (>110 psi) oxygen µm overpressure. The flowsheet is depicted in Figure 31.

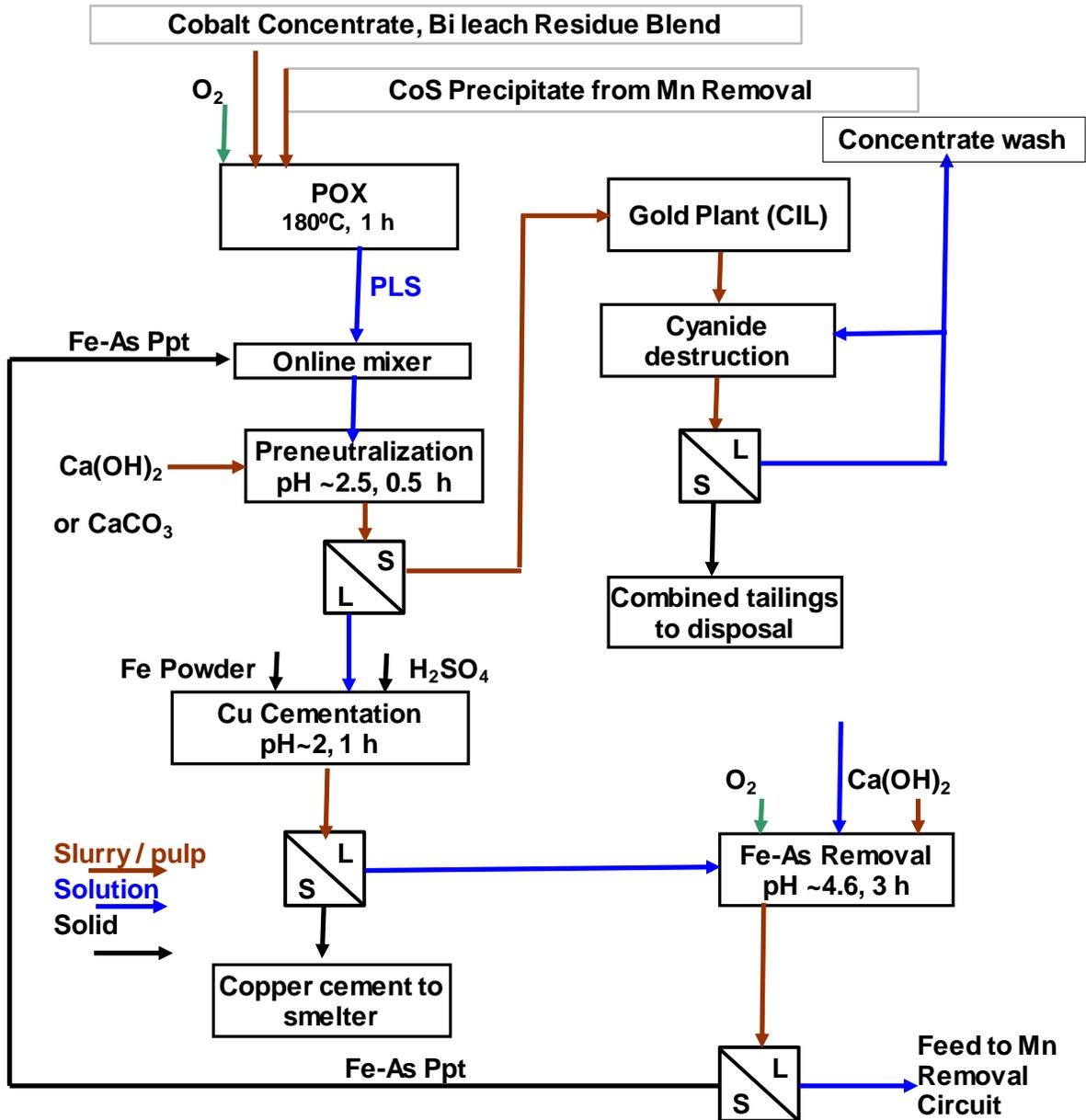
The pressure-oxidized discharge slurry containing the PLS (pregnant leach solution, pH) and leach residue is subjected to pre-neutralization at pH of about 2.5, using quicklime or limestone slurry at 25 % wt., 30 minutes residence time, 80-90°C, unheated, followed by liquid-solid separation with counter-current decantation washing (CCD).

The thickener underflow pulp consisting of POX residue, ferric arsenate and gypsum is directed to the gold recovery stage (CIL or CIP plant), followed by cyanide destruction, thickening and pressure filtration.

The combined thickener overflow and filtered solution is directed to the copper cementation with iron powder, at pH ~ 2.1 - 2.4 range maintained with sulfuric acid, 60 minutes residence time, 60°C feed temperature, unheated. The discharge slurry is filtered and washed on filter. The cement is dried and sold to the smelter.

The cementation barren solution is subjected to iron-arsenic removal at pH 4.6, oxygen and lime slurry at 25% wt. as neutralizing agent, 3 hours residence time, 50 to 60°C temperature, maintained. The discharge slurry is thickened and vacuum-filtered, no washing. The filter cake is recycled to pre-neutralization where it is blended with the POX discharge slurry. The overflow and filtrate are combined and directed to the subsequent manganese removal stage.

Figure 31 Proposed 2025 autoclave discharge flowsheet schematic



Manganese removal flowsheet schematic process description

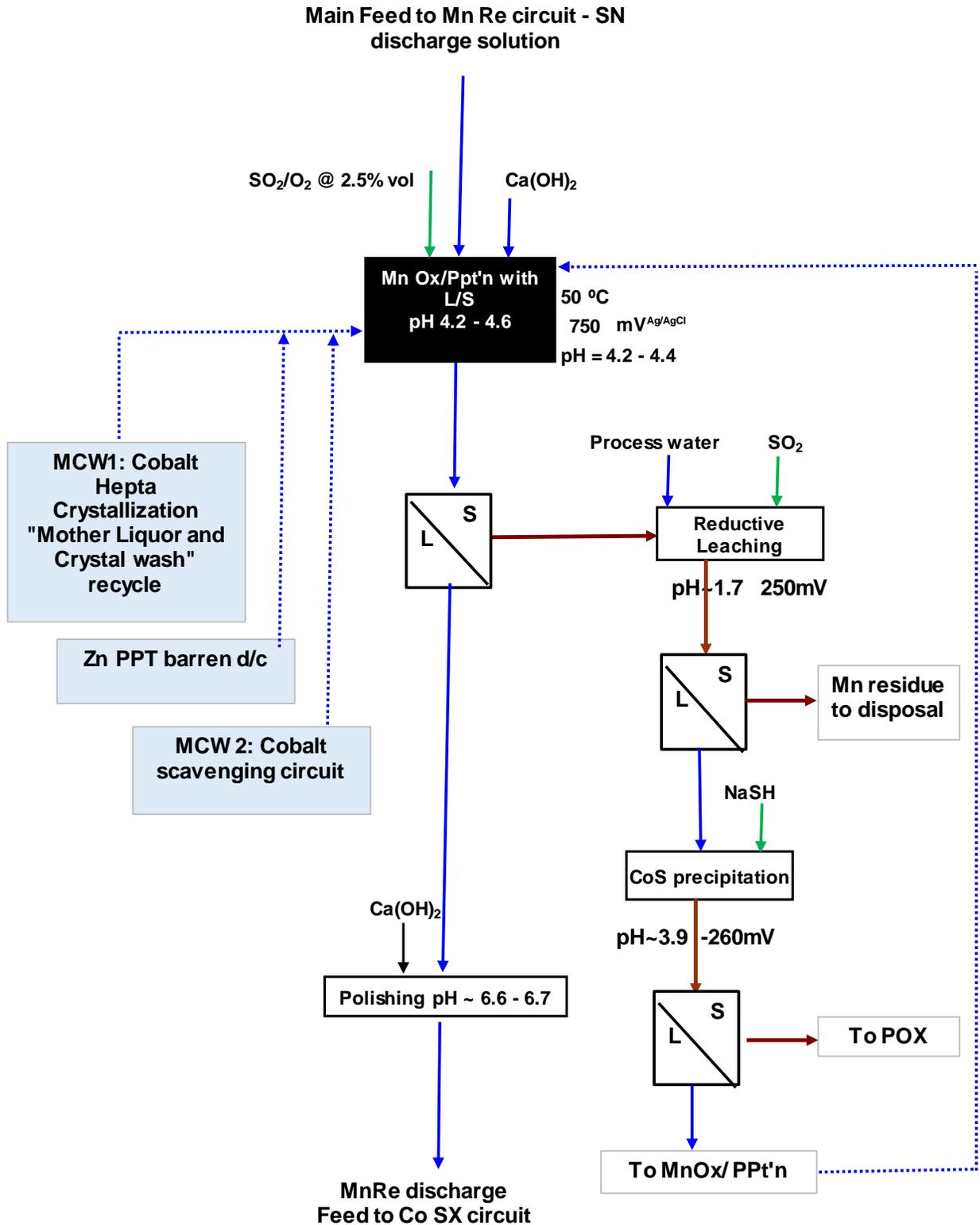
In addition to Stage II FeAsRe discharge solution, the feed to the MnRe circuit comprises the following recycle streams: Cobalt NaSH precipitation barren, Cobalt Hepta Crystallization Mother Liquor and Crystal wash ("MCW 1"), Zinc precipitate barren from the SX circuit and the Cobalt scavenging precipitation sub-stream of the aforementioned MCW stream ("MCW2"). The flowsheet is illustrated in Figure 32.

The manganese removal circuit operates using ~ 2.1% volume SO₂ in O₂ gas oxidizing mixture, pH range of 4.2 – 4.4 maintained using 12% wt. lime slurry, ORP ~ 700 - 750 mV, 70°C and ambient pressure, 2.5 hours residence time in the circuit.

The manganese removal circuit discharge is thickened and filtered. The filter cake is subjected to reductive leaching using SO₂ (50°C, 1 hour, pH ~ 1.7, 250 mV). The circuit discharge is thickened and filtered. The residue is directed to disposal. The solution is subjected to cobalt precipitation using NaSH (30°C, 1 hour, pH ~ 3.9, -260 mV). The circuit discharge is thickened and filtered. The residue is directed (recycled) to pressure oxidation. The barren is directed to the manganese removal circuit.

The manganese removal combined overflow and filtrate is directed to polishing aimed to remove any residual zinc and copper. The polishing discharge is thickened and filtered. The residue is directed to disposal. The solution is directed to the Cobalt SX circuit.

Figure 32 Manganese removal conceptual flowsheet



Solvent extraction, evaporation and crystallization flowsheet schematic process description

The manganese-free cobalt solution is directed to solvent extraction (SX) using 8% Cyanex® 272, a selective reagent for cobalt at 1.5 O/A ratio for 2.2 g/L Co in the PLS feed. The cobalt from the PLS is recovered with 99.9% efficiency throughout four stages of extraction at a target pH profile of 5.2, 5.0, 4.8, 4.6 from stages E1 to E4 respectively. The flowsheet is illustrated in Figure 33.

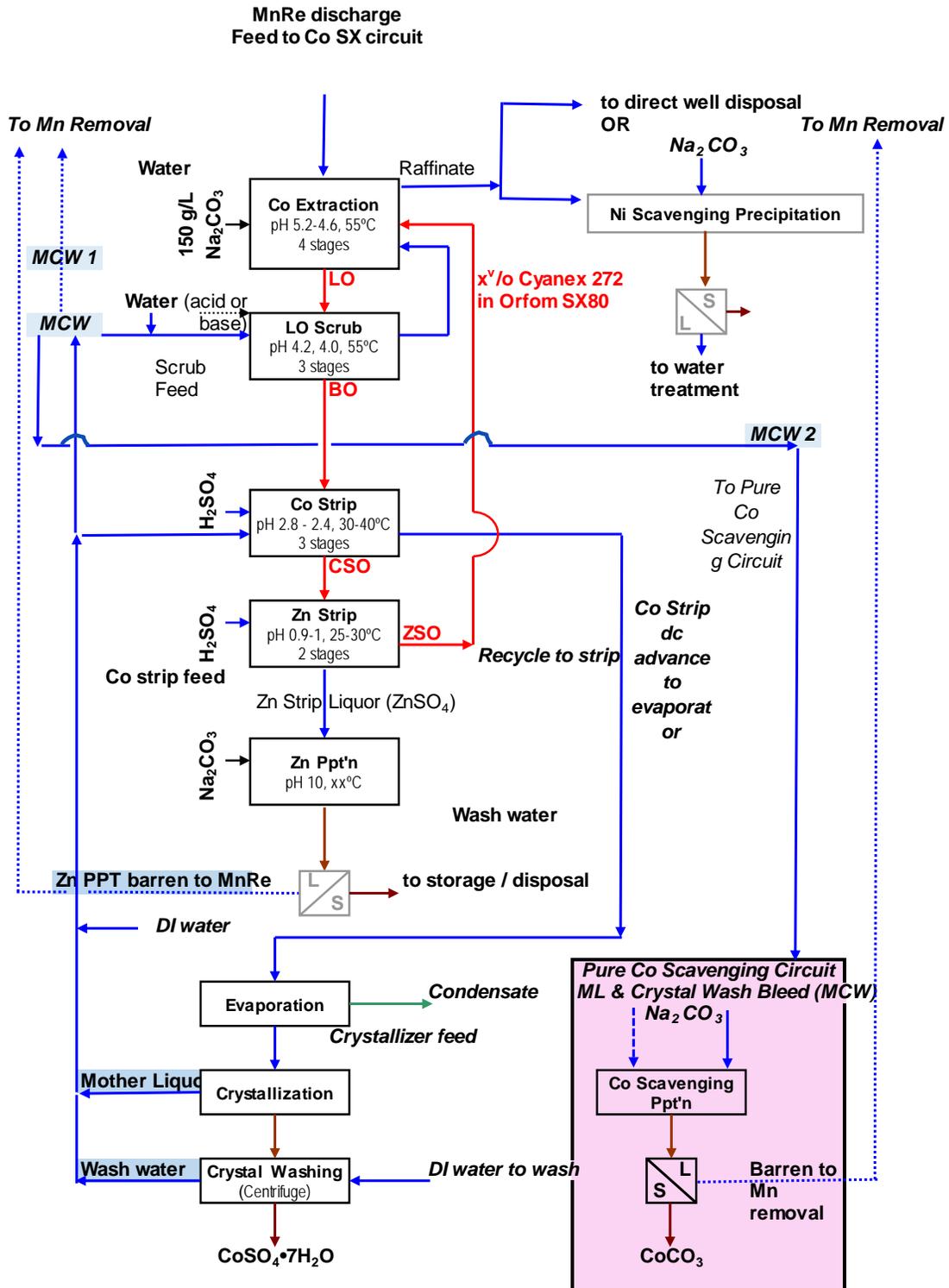
Calcium and magnesium are scrubbed-off quantitatively from the organic in three stages phase prior to stripping. Stripping is carried out using a blend of recycled mother liquor, crystal-wash solution and deionized water as needed at 150 g/L H₂SO₄ adjusted using sulphuric acid. The stripping is carried out in three stages with pH control for the target profile of 4.2, 4.1, 4.0 in stages B1, B2 and B3 respectively. The cobalt-stripped organic is subjected to two zinc strip stages to remove any traces of zinc at 200 g/L g/L H₂SO₄. The raffinate is preferably directed to the well disposal. Alternatively, it can be subjected nickel-scavenging precipitation in a dedicated circuit should the well disposal not be approved.

The cobalt strip solution is directed to evaporation at 101°C under low vacuum (~ 14”Hg), dual-stage, 2.3 hours residence time. The supersaturated evaporation discharge solution is directed to the three-stage crystallization circuit operating at 3 hours total residence time at a target temperature profile of 60/40/20 across the three stages as to achieve about 24 g/L target cobalt “bite” by cooling

Mother Liquor & Crystal Wash (MCW) Bleed is split between a direct recycle stream to the Manganese removal circuit (“MCW 1) and the cobalt carbonate scavenging precipitation circuit (“MCW2). The split ratio is determined by the purity of the mother liquor. The zinc precipitation baren is also directed to the Manganese removal circuit.

Solvent extraction was tested extensively but only before the manganese removal. The 2025 test work validated the current configuration with minor changes.

Figure 33 SX, evaporation and crystallization conceptual flowsheet



Bismuth hydrometallurgical recovery flowsheet schematic process description

The process flowsheet schematic is illustrated in Figure 34. The process is continuous, requiring surge capacity not shown in this schematic.

The bismuth concentrate is leached with regenerated ferric chloride lixiviant in a single stage configuration using a temperature controlled impeller-agitated mixing vessel. The hot leach slurry discharge is thickened. The hot underflow is directed to belt vacuum filtration with counter-current washing on filter. The filtrate is combined with the pregnant leach solution (“PLS”) overflow and directed to cementation. The well-washed filter cake is directed to the pressure oxidation circuit (“pox autoclave”).

Bismuth cementation is conducted in a single stage configuration using an impeller-agitated mixing vessel fitted with an arsine monitor. The gaseous phase discharge is vented through an arsine-retention scrubber.

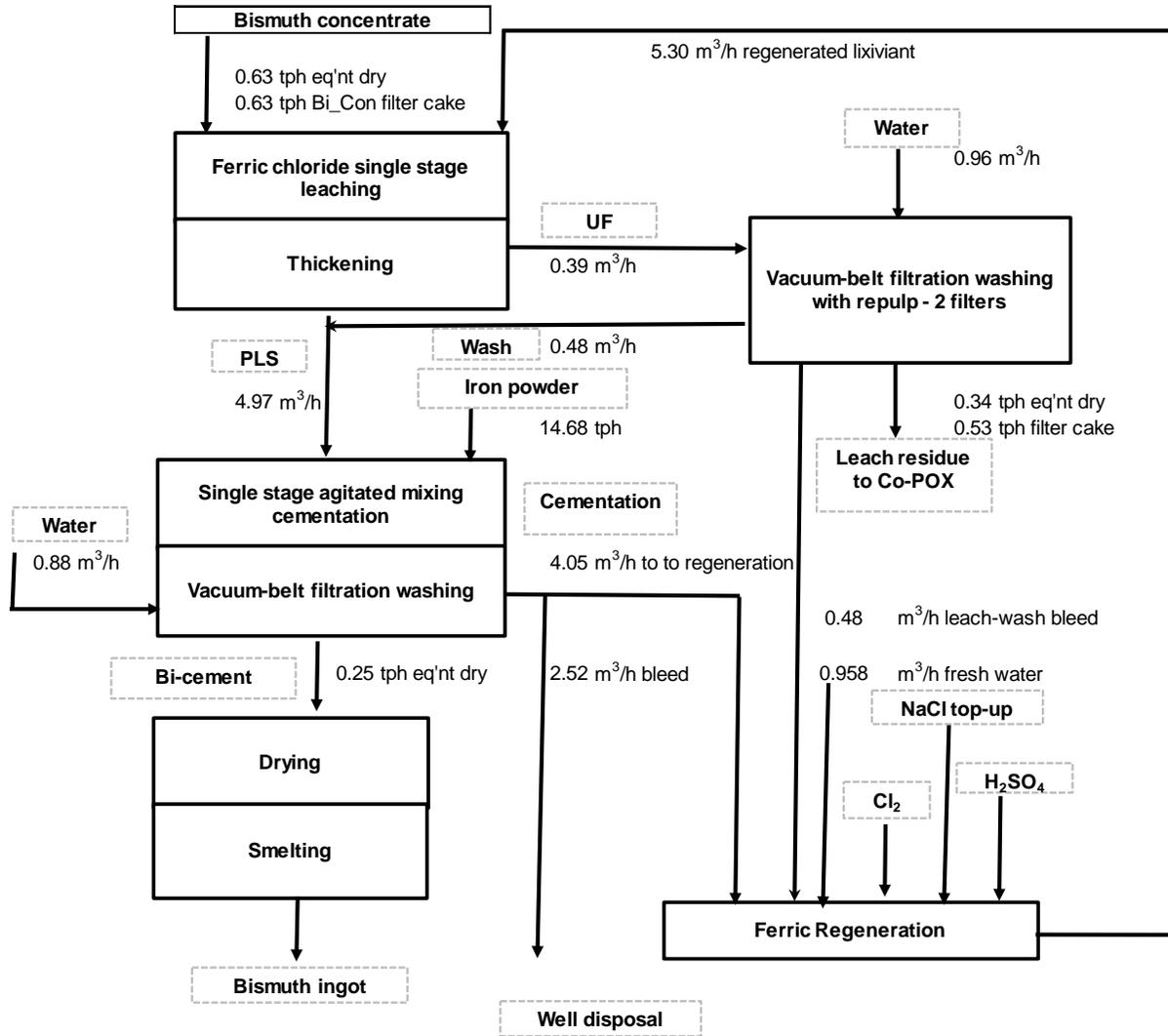
The cement slurry is filtered and washed on a belt vacuum filter. The well-washed purified cement filter cake is directed to drying and smelting.

The cementation barren solution is split into two streams, if necessary, based on the mass balance.

The main cementation barren solution stream is directed to ferric regeneration, which is conducted using chlorine. The leach-wash bleed is also directed to this unit operation. Sulfuric acid is added to the regenerated lixiviant to adjust its initial acidity according to the leaching conditions. Sodium chloride is added to top up the process losses. The regenerated lixiviant is directed to ferric chloride leaching.

Since permitting without iron removal is allowed, the excess of cementation barren solution stream is directed to underground well disposal.

Figure 34 Bismuth hydrometallurgical process flowsheet schematic



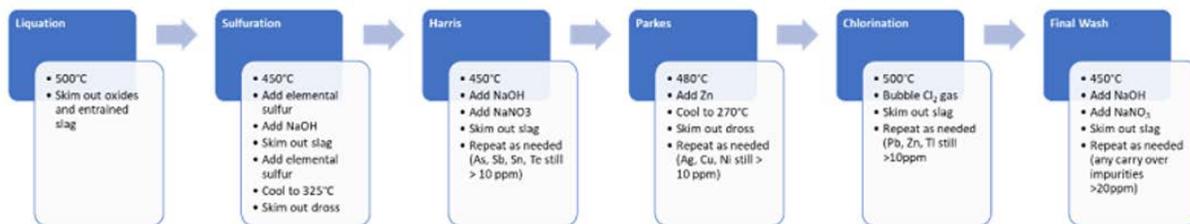
Bismuth pyrometallurgical recovery flowsheet schematic process description

The process flowsheet schematic is illustrated in Bismuth smelting and refining test work conditions and implied original conceptual produced by XPS flowsheet are illustrated in Figure 35.

Accordingly:

- Crude bismuth is produced by smelting at 950°C with flux (Na₂CO₃) and reductant (Charcoal) then subjected to the sequential refining stages outlines below.
- Liquation at 500°C – skim out oxides and entrained slag.
- Sulfuration a.k.a. Decopperization at 450°C – using NaOH, elemental sulphur, skim slag, cool to 325°C, skim dross.
- Harris: at 450°C – using NaOH, NaNO₃, skim slag.
- Parkes: at 480°C – using Zn, cool to 270°C, skim dross.
- Chlorination: at 500°C - using Cl₂, skim slag.
- Final wash: at 450°C - using NaOH, NaNO₃, skim slag, discharge to casting.

Figure 35 Bismuth pyrometallurgical process flowsheet schematic



Conclusions

Pressure oxidation related:

The 2008 pilot-validated pressure oxidation conditions were maintained for the blend-feed as follows: 1-hour retention time at 180°C-process temperature, and 6.8 atmospheres oxygen overpressure, 13-15% wt. solids (dictated by the sulphur content for autothermicity).

POX extractions were 95% Co, 79% Cu, 20% Fe, and 6% As (off-spec 1.4% Co concentrate with BiL residue) and 97% Co 74% Cu, and just 19% Fe and 5% As (on-spec 3.81% Co concentrate with BiL residue).

A key differentiated factor between the tests conducted with On-spec and Off spec samples feeds was the ferric to arsenic ratios of 23 vs. 14, respectively, hinting to the possibility of increased iron and arsenic removal even without the use of oxygen during the subsequent dedicated steps.

On-spec PLS tenors (before dilution with filtrate) assayed 5.6 g/L Co, 0.6 g/L Cu, 8.7 g/L Fe and 0.6 g/L As. For comparison, representative 2008 pilot PLS contained on average 5.5 g/L Co, 0.7 g/L Cu, 3.2. g/L Fe and 0.9 g/L As.

The residues were also comparable except for the sulphur as sulphide which assayed 0.6% S²⁻ in POX 4 residue (3.2% total sulphur), compared to 2.4 S²⁻ in the pilot residue (4.9% total sulphur), indicative of potentially more advanced oxidation in the current work, or/and, due to the nature of the current sample. Of note, the 2008 pilot sample consisted of cobalt concentrate, not blended with bismuth ferric chloride leach residue. Overall, the 2025 bench-scale pressure oxidation results exceeded the 2008 pilot results.

Selective gypsum precipitation test work investigating whether or not the gypsum produced according to the 2008 and 2017 current flowsheet contain sufficiently low arsenic levels as to allow for its safe disposal (as non-registered waste) or even potential commercial uses. The tests produced removals from 21 to 83% of the free acid while producing gypsum residues containing from 30 ppm As (0.02% removal) to 199 ppm As (1.25% removal), respectively, and within the pH ranging from about 1.1 to 1.4, respectively.

Recommended to determine whether certain industries accept arsenic in gypsum. Further testing for Class II landfill disposal criteria needs to be undertaken for the material for disposal option considerations; currently the low pH of the material makes it unsuitable for non-hazardous disposal without further treatment or amendments. No commercial production is recommended until the marketing study is completed.

Unseparated POX Discharge stream and gold related:

Primary non-oxidative neutralization test work investigated iron and arsenic removal on the entire unseparated POX slurry discharge while taking advantage of the oxidation potential of the ferric ion contained in the POX-PLS while comparing limestone vs. lime as neutralization reagents to allow for process design and cost flexibility.

Overall, the results of the 2025 primary neutralization test work provided that effective removal of 99% of the arsenic and from 80% to 92% of iron under optimized conditions was possible at pH ~ 2.5, without oxygenation and heating, using either limestone or lime and with a very short residence time of 0.5 hours, and without any cobalt losses by co-precipitation and entrapment.

The results compared are superior to those produced with PLS-sample during the 2017 test work but using oxygen and heated (2017). However, the arsenic removal in 2017 ranged from 56% to 91%, inferior compared with the current results produced without separating the residue from the PLS.

Base line carbon in leach cyanidation on the pressure oxidation discharge residue produced 98.5% gold recovery after 24 hours, with 98.1% adsorbed onto carbon at 4.425 kg/t Au. The specific consumptions were 0.64 kg/t NaCN and 11.6 kg/t CaO. Carbon in leach cyanidation on the PN 1 discharge residue (limestone) produced 97.3% gold recovery after 24 hours, with 96.9% adsorbed onto carbon at 2.605 kg/t

Au. The specific consumptions were 1.31 kg/t NaCN and 23.2 kg/t CaO. Carbon in leach cyanidation on the PN 4 discharge residue (lime) produced 97.5% gold recovery after 24 hours, with 97.1 % adsorbed onto carbon at 2.705 kg/t Au. The specific consumptions were 1.14 kg/t NaCN and 20.6 kg/t CaO.

Overall, the gold-recovery differential between the POX discharge and the limestone and lime neutralized residues were within about one percent. According to the 2024 mini-pilot data, about 6% of the overall gold reported to the clean scavenger tailings. Current results produced gold extractions insensitive to aging, averaging 83%. The recovery from the bulk rougher tailings was 65%.

Post cyanide destruction TCLP results on residues generated from the preneutralized unseparated autoclave discharge slurry cyanidations (lime and limestone) and clean scavenger tailings cyanidation (total of three samples) indicated that the samples were not leachate toxic. Modified acid base accounting tests (ABA) results in duplicate on the above three samples indicated that all displayed negative Net Neutralization Potentials (NNPs), implying that they are Potentially Acid Generating (PAG).

Copper cementation results produced residual copper levels of 0.3 mg/L Cu or less, precluding the need for polishing.

Secondary neutralization results indicated the possibility of completely removing the iron from the cementation discharge barren (<70 mg./L Fe with air, 10 mg/L with oxygen) and arsenic (<3 mg/L, both oxidants). However, these results did not meet the requirements of adequate iron removal for the manganese removal circuit. As a result, the design criteria were reverted to the 2017 conditions, listed in the dedicated section. The secondary neutralization discharge is the feed to the manganese removal circuit, which is followed by polishing before solvent extraction.

Cobalt Sulphate Heptahydrate specifications related

The cobalt sulphate heptahydrate produced from the polished stream exceeded 99.99% (20.959% Co), slightly exceeding the battery grade minimum of 99.99% (20.958% Co).

All heavy metal impurities assayed were either below or between the limits imposed by the battery grade specifications. Magnesium specifications will be met through the SX engineering optimization work as detailed herein.

In conclusion, downstream solution purification results confirmed the validity of the established secondary neutralization sequence by producing a cobalt sulphate heptahydrate feed which was subjected to manganese removal, solvent extraction followed by the evaporation - crystallization sequence. The design criteria remain valid but with the inclusion of the slight changes in the SX circuit,

detailed in the next section. The data are suitable to be refined by SysCad modeling for the purpose of the feasibility study.

Bismuth hydrometallurgy related

Bismuth leaching extraction after liquid-solid separation and washing of 98.71 % based on the leach-only (99.5%) and wash only (99.21%) recoveries and PLS and wash vs. feed recovery. The washing maximizes the chloride removal from the washed cake to about 38 mg/L final Cl- tenor and the bismuth tenor in the combined PLS and wash (45,345 mg/L Bi tenor).

Cementation updated conditions produced 99.99% cementation efficiency vs. solids from the NICO feed, leaving from 3 to 6 mg/L Bi in the barren solution. The NICO cements consistently assayed above 99% Bi by direct assays and above 98% by difference vs. impurities. Most notably, both cement products contained less than 0.6 % Fe. Unoptimized conditions applied to the NICO/Rio Tinto feed blend produced 99.3 leaching efficiency (before liquid-solid separation and washing) and 99.99% cementation efficiencies, producing a cement assaying 91.5% Bi with about 3.2% Fe content.

Lixiviant regeneration efficiency using chlorine was about 95%, over 3 hours addition time, requiring 0.84 ton chlorine per ton of iron oxidized dosed at 125% stoichiometry excess. This regeneration option was selected. Direct discharge of the bleed stream being permitted, regardless of its iron content, negating the need for the iron removal by oxidation. In this case, no oxygen is needed, and the lime would be used for pH adjustment only to ~4, bringing the specific consumption to 0.08 t/t lime as Ca(OH)₂ pure equivalent per ton of iron discarded.

Bismuth pyrometallurgy related

Successful conversion of two types of Bi cement provided by Fortune to Bi metal with >99.9% purity was achieved. A smelting recipe has been developed for the two cement samples, with some difference in the ratio of flux and reductant, depending on the composition of the feed. The metal produced from the smelting trials were successfully refined to produce ingots assaying from >99.98% to 99.99 % Bi metal.

Industrial implementation of the process is expected to yield overall recoveries generally exceeding 90% after recycling of the intermediate streams.

Recommendations

It is recommended that the process allow 10% oxygen overpressure excess to maximize the advantage of

the oxidation potential of the ferric ion contained in the POX-PLS.

Arsine generation was not detected during copper cementation, but the necessary safeguards should be maintained: monitoring and scrubbing.

There appears to be room for further improvement in keeping magnesium out of the cobalt product, as detailed herein. Firstly, the upper limits of cobalt loading must be found. Secondly, there may be some room for optimizing the pH profile in extraction by lowering the pH in E4. However, a more effective strategy may be to establish a pH profile in the scrub stages by adding pH modifier. In a longer campaign, this would have been done. Lower pH across the scrub stages may help displace more Mg back upstream with the scrub liquor. Addition of a third stage of Co scrubbing is another measure that may improve the Mg separation and help with some other trace impurities as well. A systematic program to adjust and optimize operating conditions in the production plant should enable these measures to be implemented as long as the plant has been designed to allow the recommended actions to be carried out, specifically pH control in scrubbing using acid and a third scrub stage.

The liquid solid separation criteria for most of the circuits should be enhanced by recycling the thickener underflow to the actual process (“seeding”) to allow for increased solids content in the filtration feed. Short-route recycling to the feedwell is also an option. Dewatering of the final moisture content should be amenable to extrapolation to a squeeze-pressure-filter target. Centrifugation could be considered as a last resort option due to its cost (capex primarily).

Updated criteria pending for liquid-solid separation, dewatering and handling design criteria streams subject to re-evaluation: PN discharge, Final plant discharge and Bulk concentrate. Placeholder values used in-interim for the PDC.

The Mother liquor & Crystal Wash (MCW) Bleed Split recycle to SX and Mn-removal circuit has not yet been tested. A combination of testing and modelling is recommended. Of note, cobalt carbonate sale vs. recycling ratio, determined by purity specifications. Process data to be generated by modelling.

Further optimization is required to ensure impurity specifications are met and Bi losses are minimized. It is recommended to develop a plan for treating the slags and drosses generated from the refining stages in order to recover the Bi units, as well as other elements that may be of interest to other industries (e.g. Te, Sb, Ag).

Phase 8 results

With successful completion of the project, Fortune is expected to receive a suite of final technical reports prepared by metallurgical consultants, including:

- A summary report on the 2024 Mini-Pilot Program
- Bench-scale Flotation test work report
- Jameson cell flotation test work report
- Additional concentrate production pilot report
- Design-grade cobalt flotation work report
- HPGR testing and product characterization report
- BWI variability report
- Bismuth metal production report

These deliverables included the finalized Process Design Criteria for incorporation into the Alberta Refinery engineering design, as well as the final reporting package required by Alberta Innovates.

Fortune also prepared an internal technical summary and presentation highlighting key findings from the program. This presentation was delivered at Extractions 2025, a Metallurgy and Materials Society (MetSoc) conference in Phoenix, Arizona. A second presentation will be delivered at MetSoc Conference, August 2026, Calgary, Alberta.

E. KEY LEARNINGS

- **Please provide a narrative that discusses the key learnings from the project.**
- **Describe the project learnings and importance of those learnings within the project scope. Use milestones as headings, if appropriate.**
- **Discuss the broader impacts of the learnings to the industry and beyond; this may include changes to regulations, policies, and approval and permitting processes**

RESPOND BELOW

Milestone 1

Phase 1 of the program successfully delivered an optimized, risk-mitigated flowsheet and a clear design basis for subsequent pilot testing and feasibility work. The phase began with a detailed review of prior test work, led by Fortune's Chief Metallurgist, Alex Mezei, which helped refine well-established metallurgical processes and define priorities for Milestone 2. Key improvements included:

Flowsheet and process enhancements

- Finalized process diagrams, test conditions, and detailed engineering data for bismuth circuit sizing.

- Improved management of feed variability and downstream product quality.
- Identified potential to produce a saleable Agri-gypsum by-product from neutralization of waste streams.

Flotation circuit optimization

- Validated 54-micron grind size and optimized reagents to maximize recovery of cobalt, bismuth, gold, and copper.
- Achieved ~4% mass pull, reducing ore volume from ~4,650 tpd to ~180 tpd, significantly lowering transport and processing costs.
- Secondary flotation after 14-micron regrinds successfully produced selective cobalt and bismuth concentrates.

These improvements collectively strengthen the NICO Project's process efficiency, environmental performance, and economic viability, while offering broader benefits to the critical minerals industry. The ability to significantly reduce ore mass for transport and processing improves project sustainability and cost-effectiveness, which is particularly valuable for remote northern operations. Enhanced flexibility to process third-party feedstocks and recover marketable by-products supports circular economy practices and supply chain resilience.

Milestone 2

Milestone 2 generated a cohesive set of technical and strategic learnings that collectively de-risk the NICO Project flowsheet and provide a useful reference for other critical-mineral developments in Alberta and beyond. Test work across the concentrator and bismuth circuits confirmed that a more compact, energy-efficient design can be implemented without sacrificing metal recoveries. Replacement of conventional cone crushing and ball milling with High Pressure Grinding Rolls (HPGR) and vertical stirred mills is expected to reduce concentrator capital costs by approximately C\$7 million and annual operating costs by about C\$1.5 million, largely through lower power demand and smaller equipment. Confirmatory HPGR and comminution testing at SGS and Metso verified that NICO ores are well suited to this configuration, providing defensible design parameters for equipment sizing and power requirements.

Flotation test work showed that modern cell technology can be used to recover more of the critical and precious metals already mined. The Jameson cell trade-off study demonstrated a 3–5% improvement in fine bismuth and gold recoveries from bulk rougher tailings while producing a cobalt concentrate grading approximately 4% Co for downstream hydrometallurgical testing.

Hydrometallurgical test work on the bismuth circuit confirmed that high metal recoveries can be achieved in a simplified, lower-cost flowsheet. Bulk ferric chloride leaching of the Milestone 1 bismuth concentrate, followed by cementation onto iron powder, achieved approximately 97% bismuth recovery and produced a bismuth cement averaging 95% purity. Large-scale testing demonstrated that the leach residence time can be reduced from three hours to one hour in a single-stage configuration without loss of recovery, supporting an estimated 67% reduction in leach circuit volume. Lixiviant regeneration tests using hydrogen peroxide and a lower-cost chlorine alternative further showed that the leach solution can be effectively regenerated and recycled, reducing net reagent consumption and improving water management.

Feed-flexibility test work demonstrated that the flowsheet can accommodate suitable third-party materials without compromising performance. Blending Rio Tinto's bismuth oxychloride intermediate from the Kennecott Smelter at a 1:10 ratio with Fortune's bismuth concentrate did not negatively impact bismuth recoveries, indicating the potential to augment annual bismuth production by 10–20% while diverting metal-bearing residues from long-term storage. This finding is strategically important for the project, as it supports the concept of the Alberta Refinery as a regional critical-minerals hub capable of treating both primary concentrates and secondary feedstocks.

Gold deportment studies completed in parallel showed that the refined base-metal flowsheet remains fully compatible with high gold recoveries. Monitoring during ferric chloride leaching confirmed that gold stays in the leach residue rather than reporting to solution, and subsequent carbon-in-leach (CIL) cyanidation of this residue achieved high gold extraction. Although the preferred base-case remains to blend bismuth leach residue with cobalt concentrate and recover gold after pressure oxidation of the combined residue, the confirmed viability of direct cyanidation provides an alternative recovery route that enhances operational resilience.

Taken together, the Milestone 2 learnings confirm that the NICO Project can be advanced using commercially proven, higher-efficiency technologies configured to deliver strong recoveries, reduced energy and reagent consumption, and flexibility to integrate third-party feeds. These outcomes materially strengthen the technical and environmental case for the Alberta Refinery, support advancement of the project's Technology Readiness Level, and provide transferable insights that can inform future engineering studies, investment decisions, and regulatory reviews for critical-mineral projects across Canada.

Milestone 3

Milestone 3 completed validation of the fully integrated NICO flowsheet from concentrate through cobalt, bismuth and gold circuits to final products and environmentally stable residues. Key learnings are summarized below:

Integrated cobalt circuit can reliably deliver battery-grade cobalt sulphate.

- POX on blended cobalt concentrate and bismuth leach residue consistently achieved up to ~97% cobalt and ~79% copper dissolution, confirming the selected autoclave operating window for design.
- Optimized primary neutralization at ~pH 2.5 using either lime or limestone (0.5 h residence, no heating or oxygenation) removed ~99% of arsenic and 80–92% of iron, while copper cementation reduced dissolved copper to <0.3 mg/L.
- Downstream manganese precipitation, solvent extraction and crystallization produced high-purity cobalt sulphate heptahydrate suitable for battery-grade applications.

Bismuth pyrometallurgical flowsheet is proven to commercial purity levels.

- Smelting of bismuth cement from the hydrometallurgical circuit produced a crude 99.8% Bi (“2N”) ingot, which was successfully refined via liquation, sulphuration and chlorination to 99.99% Bi (“4N”).
- The XPS program delivered finalized operating conditions and Process Design Criteria for inclusion in the Feasibility Study.

Gold recovery is robust, flexible, and compatible with the base-metal circuits.

- Gold deportment monitoring showed that gold remains in solid residues during ferric chloride leaching, enabling efficient downstream cyanidation without contaminating cobalt or bismuth solutions.
- CIL tests on POX residues (blended cobalt concentrate + bismuth leach residue) achieved up to 98.5% gold recovery; lime vs. limestone neutralization gave <1% difference, confirming reagent flexibility and eliminating the need to split POX discharge streams.
- Contingency testing demonstrated that intensive cyanidation of bismuth ferric leach residue can recover up to 95% gold within 48 hours during autoclave outages, while CIL of scavenger flotation tails adds further recovery potential.

Environmental performance of residues and cyanide destruction is demonstrably acceptable for Class 2 landfill disposal.

- TCLP testing confirmed that iron/arsenic precipitates from lime- and limestone-neutralized residues, POX residues, and CIL tailings meet criteria for disposal in a government-approved Class 2 landfill.
- Liquid–solid separation tests generated design data for thickening, filtration and long-term handling, while ABA testing confirmed acceptable long-term geochemical stability of key residue streams.
- Cyanide destruction optimization established operating conditions to reliably meet detoxification and discharge criteria for cyanide-bearing solutions and tailings.

Gypsum will be managed as an environmental residue first, with targeted—but constrained—by-product potential.

- Sulphate removal and gypsum formation from autoclave effluents were technically successful; however, arsenic incorporation into gypsum could not be reduced to levels suitable for broad commercial markets.
- The learning shifts gypsum from a primary co-product concept toward a managed residue, with future work focused on (i) niche applications that can safely accept gypsum with controlled arsenic, and/or (ii) ensuring gypsum can be treated to meet criteria for non-hazardous disposal.

HPGR-based comminution is fully supported by ore-hardness data and is ready for detailed engineering.

- Final LABWAL HPGR tests at SGS and Jar/Bond Work Index data from Metso, across four representative ore types, confirmed that HPGR can replace conventional crushing and milling in the concentrator.
- These data underpin final equipment selection, sizing and power requirements for the updated concentrator flowsheet.

The project now has a complete, documented design and knowledge base for commercialization. A suite of final technical reports covering the mini-pilot, flotation and Jameson cell performance, concentrate production, HPGR testing, and bismuth metal production has been or will be issued, along with finalized Process Design Criteria for all major circuits.

F. OUTCOMES AND IMPACTS

Please provide a narrative outlining the project's outcomes. Please use sub-headings as appropriate.

- **Project Outcomes and Impacts:** Describe how the outcomes of the project have impacted the technology or knowledge gap identified.
- **Clean Energy Metrics:** Describe how the project outcomes impact the Clean Energy Metrics as described in the *Work Plan, Budget and Metrics* workbook. Discuss any changes or updates to these metrics and the driving forces behind the change. Include any mitigation strategies that might be needed if the changes result in negative impacts.
- **Program Specific Metrics:** Describe how the project outcomes impact the Program Metrics as described in the *Work Plan, Budget and Metrics* workbook. Discuss any changes or updates to these metrics and the driving forces behind the change. Include any mitigation strategies that might be needed if the changes result in negative impacts.
- **Project Outputs:** List of all obtained patents, published books, journal articles, conference presentations, student theses, etc., based on work conducted during the project. As appropriate, include attachments.

Project Outcomes and Impacts

The outcomes of the project directly addressed and resolved several key technology gaps that had previously limited the economic and sustainable development of the NICO Deposit. One of the most critical challenges was the need to reduce the volume of raw ore for transportation while maintaining high recovery of valuable metals. Through pilot-scale validation of optimized crushing, grinding, and flotation processes, the project successfully demonstrated the ability to concentrate recoverable materials into a bulk flotation product with a low mass pull of approximately 4%, thereby significantly reducing transportation costs and associated emissions. The ability to generate high-quality intermediate concentrates not only improves downstream processing efficiency but also contributes to a smaller environmental footprint.

The project also filled a major gap in domestic metallurgical refining capabilities. North American smelters have historically lacked the capacity to process complex polymetallic ores like those of the NICO Deposit, resulting in dependence on overseas refineries. By developing and validating a process flowsheet tailored to this unique mineralogy, the project confirmed that high-purity critical mineral products can be produced domestically.

Further innovations, such as the regeneration of ferric chloride lixiviant, effective pre-neutralization of arsenic and iron without intensive conditions, and the potential to produce a saleable gypsum by-product, enhance process sustainability and waste minimization. These outcomes provide the foundation for a Canadian-based refining solution that can be extended to other domestic resources, smelter residues, and recycling streams.

Clean Energy Metrics:

First, the project directly supports investment in clean resource technologies by validating the recovery of critical minerals essential to the clean technology and battery supply chains. The production of high-purity bismuth, cobalt sulphate, and gold, alongside the potential recovery of a saleable gypsum by-product, reinforces the project's contribution to increased electrification and sustainable industrial practices such as soil stabilization and cement production.

The test work completed over the course of the project has played a critical role in advancing the Technology Readiness Level (TRL) of Fortune's integrated mineral refining processes. Milestone 1 successfully validated the optimized flowsheet for the proposed concentrator in the Northwest Territories, confirming its ability to produce high-quality intermediate concentrates. Building on this, Milestone 2 demonstrated the effectiveness of an optimized suite of hydrometallurgical techniques tailored for the Alberta refinery. With final test work on cobalt sulphate, copper cement, and iron/arsenic removal nearing completion, the project is on track to achieve TRL 6 and move toward commercial-scale engineering and eventual TRL 9 deployment at the proposed Alberta facility.

The mini-pilot conducted at SGS functioned as a small-scale field demonstration. By integrating commercially proven technologies into a unified pilot, the project simulated real-world conditions and validated process effectiveness across flotation, leaching, cementation, impurity removal, and refining stages. These outcomes de-risk future commercial investment, particularly in Fortune's proposed Alberta refinery, and set the stage for full-scale deployment.

Knowledge sharing activities conducted to date are included in Section I of this report.

The project also upholds its commitment to rural and Indigenous inclusion. The involvement of NWT Indigenous communities in the NICO mine project establishes a foundation for continued partnership, with cooperation and access agreements with the Tlicho Government. The NWT Indigenous community is already involved in the NICO mine project and Fortune has also signed a socio-economic agreement

with the government of the NWT, setting targets for northern and Indigenous employment and other benefits.

There may be further potential for Indigenous participation and investment in the Alberta Refinery. Fortune has held meetings with local nations regarding potential partnerships as well as the Capital Region Indigenous partnership. Fortune has also had discussions with the Alberta Indigenous Opportunities Fund and the Horizons West Fund.

Program Specific Metrics

The outcomes of Phases 1 through 3 establish a strong technical foundation for achieving the project's key performance metrics. By successfully producing high-quality intermediate concentrates and confirming the ability to process third-party materials; the project advances its objectives of reducing smelter waste and promoting the circular use of critical mineral resources. The substantial reduction in ore mass for transport, from 4,650 tonnes to 180 tonnes per day, significantly lowers the project's carbon footprint and reinforces the environmental benefits of a localized North American supply chain. The selective recovery of bismuth and cobalt concentrates ensures the availability of feedstock for downstream refining that meets battery-grade purity standards, aligning with market requirements.

Building on this foundation, the results from Phases 4 through 7 further demonstrate the project's ability to meet its performance targets. The successful production of battery-grade cobalt sulphate and 99.99 percent pure bismuth ingots confirms the project's capacity to deliver saleable critical minerals, supporting Performance Metrics 3 and 4. Streamlined leaching and cementation processes, along with effective impurity removal and waste treatment, contribute to Performance Metric 2 by reducing waste volumes and ensuring the safe disposal of residues. The ability to integrate third-party feedstock and recover metals from multiple process streams enhances operational flexibility and resilience. In parallel, reductions in reagent consumption and emissions strengthen the environmental benefits of domestic refining and support Performance Metric 1.

Additional Project Approval Conditions

Additional feedstocks

With the inclusion of Rio Tinto materials, cobalt and bismuth production at the proposed Alberta Refinery could expand by 10 to 20%. The integration of bismuth intermediates sourced from Rio Tinto's Kennecott smelter residues has been fully validated. These intermediates can be blended with NICO concentrates without adverse impact on recovery, refining efficiency, or final product quality. Their successful

incorporation enhances Total Orebody Effectiveness by reducing smelter waste, increasing metal recovery, and supporting the co-production of bismuth and cobalt at the Alberta refinery.

The Alberta Refinery is also expected to attract additional business from other projects, particularly in the NWT which has a collaboration agreement with Alberta. Preliminary discussions have been held with Glencore, Teck, Vale and 5N Plus to source additional bismuth feedstocks as well as with Fireweed regarding processing tungsten oxide concentrate from the Mac Pass Project to ammonium para tungstate, Vital Materials for REE processing, and Lift, for processing spodumene concentrates to extract the lithium.

Fortune has undertaken discussions with an Alberta company with access to metal shales from The Birch Mountain area of Alberta, and a company in the NWT has access to lithium and rare earth elements baring regarding providing feedstocks for the proposed refinery. Fortune continues to explore metal materials collected for recycling as a potential feedstock.

Post-Secondary Engagement within Alberta

To support continued research and meet initial project conditions, Fortune has actively engaged with post-secondary institutions and subject matter experts to explore collaborative opportunities. One such initiative includes potential in-kind and direct support for postgraduate research under the supervision of Dr. Pilar Lecumberri-Sanchez, Associate Professor of Geology at the University of Alberta, building on Fortune's existing work with the Geological Survey of Canada on the NICO Project and surrounding areas. Further discussions have taken place regarding data sharing and potential thesis sponsorships with researchers Wei Victor Liu, Qui Liu, and Alex Umnikov at the University of Alberta.

G. BENEFITS

The economic benefits of the NICO Project and Alberta Refinery include advancing Canada's and Alberta's reputation and capacity as a domestic refiner and supplier of Canadian sourced critical minerals and supporting Edmonton as a growing critical minerals processing hub for the emerging electric vehicle industry, new technologies and defense. As a vertically integrated Critical Minerals project with mid-stream processing the NICO Project will be a supplier of cobalt and bismuth products to reduce supply chain vulnerabilities from the traditional supply sources in foreign entities of concern. The products will also have supply chain transparency and security of supply of the contained critical minerals from ores through to the production of value-added products.

The Alberta Refinery will contribute to critical minerals processing and enable smaller producers to be developed by providing a domestic process solution for cobalt, bismuth, copper, and potentially other critical minerals. The Alberta Hydrometallurgical Facility will also be a process solution for waste chemical residues, scrap metals, and recycling of electronics and spent batteries in western Canada. Proof of this

concept is the process collaboration Fortune already has with Rio Tinto that will involve processing intermediates produced from Kennecott Smelter waste residues in Utah at the Alberta Refinery and augment commercial production of cobalt and bismuth. The facility will contribute to Canada's hydrometallurgical process knowledge and facilities.

The capital costs for the NICO Project were estimated at ~C\$770 million in 2020, including ~\$340 million for the Alberta Refinery. These costs are expected to be higher using 2025 escalated capital costs. The NICO Project will create ~350 direct jobs, including 120 at the Alberta Refinery. Using traditional industry multipliers, indirect employment is expected to be two times the direct employment. At today's metal prices, annual sales of metal products from the Alberta Refinery would be approximately US\$202 million from gold, US\$80 million from cobalt sulphate, US\$56 million from bismuth and US\$2.6 million for copper over the 20-year expected NICO Project life. As mentioned, with the inclusion of Rio Tinto materials, cobalt and bismuth production could expand by 10 to 20%. The Alberta Refinery is also expected to attract additional business from other projects, particularly in the Northwest Territories which has a collaboration agreement with Alberta.

The Alberta Refinery is expected to survive the depletion of NICO Deposit Mineral Reserves by processing other feed sources. For example, the Sherritt Refinery in Fort Saskatchewan has been producing nickel, cobalt and fertilizers for approximately 70 years, considerably longer than the initial mine that the facility was constructed to process concentrates. Fortune has secured a purchase option for its brownfield site in Lamont County in Alberta's Industrial Heartland Association.

Products from the NICO Project will contribute important supplies of cobalt sulphate and bismuth metal contributing to the supply of materials needed for the energy transition, new technologies, environmental products and defense. The cobalt sulphate is targeting the lithium-ion rechargeable battery industry and the manufacture of cathodes for batteries used in EV's, portable electronics and stationary energy storage. The bismuth is needed in numerous industrial applications, and an environmentally safe and non-toxic substitute for lead in brass, solders, free-machining steel and aluminum, radiation shielding ceramic glazes, ammunition and glass. Bismuth plugs will reduce greenhouse gas leakage and groundwater contamination from decommissioned oil and gas wells that have been improperly sealed or have concrete plugs have failed or leak. Shorter supply chains for these metals will also reduce emissions from domestic processing and transportation of the products to North American users as opposed to the current suppliers in Africa and Asia or both.

Additionally, development of this vertically integrated project will reduce supply chain risks for important critical minerals and diversify the supply chain from dependence on the current sources of supply in foreign entities of concern.

H. RECOMMENDATIONS AND NEXT STEPS

Please provide a narrative outlining the next steps and recommendations for further development of the technology developed or knowledge generated from this project. If appropriate, include a description of potential follow-up projects. Please consider the following in the narrative:

- Describe the long-term plan for commercialization of the technology developed or implementation of the knowledge generated.
- Based on the project learnings, describe the related actions to be undertaken over the next two years to continue advancing the innovation.
- Describe the potential partnerships being developed to advance the development and learnings from this project.

Fortune's long-term vision for the NICO Project centers on completing construction and advancing toward full commercialization as a vertically integrated producer of critical minerals in North America. The company aims to establish a reliable, transparent supply of cobalt, bismuth, gold, and copper, positioning the Company as a key contributor to sustainable resource development and the global transition to clean energy technologies. Central to this strategy is Fortune's commitment to responsible mining practices, full supply chain traceability, and alignment with Canadian government objectives to strengthen domestic critical mineral supply chains.

Over the next year, Fortune will focus on a series of actions to advance the NICO Project to a Financial Investment Decision (FID). These actions include finalizing an updated feasibility study, completing detailed engineering and permitting activities for commercial construction, and continuing to optimize processes validated through the test work completed in this project. The successful piloting of the concentrator and hydrometallurgical processes has de-risked the planned process and provided a strong foundation for scaling the operation. Efforts will also be directed at identifying and securing additional feed materials to support future expansion and ensure long-term operational viability.

To support the transition from development to production, Fortune is actively engaging with a broad network of strategic stakeholders. The company has entered into multiple Confidentiality Agreements with potential partners across the battery, automotive, aerospace, and private investment sectors. Discussions are focused on securing offtake-linked investments that will provide the financial backing necessary to advance the project. Notably, interest has been particularly strong among North American companies seeking ethical and secure alternatives to mineral supply chains currently dominated by the Democratic Republic of Congo and China. These partnerships are not only critical to financing but also to establishing Fortune as a cornerstone of Canada's emerging battery materials ecosystem.

I. KNOWLEDGE DISSEMINATION

Please provide a narrative outlining how the knowledge gained from the project was or will be disseminated and the impact it may have on the industry.

RESPOND BELOW

The knowledge gained from the project will be disseminated through technical reports and a formal presentation outlining Fortune's process technologies and key learnings. These materials will be shared directly with Alberta-based facilities, refineries, and post-secondary institutions to encourage broad market adoption and contribute to the growth of Canada's critical minerals sector. Additionally, project findings will be presented at industry events, including those hosted by the Alberta Industrial Heartland Association, to further support knowledge mobilization and foster collaboration across the sector.

A technical presentation and all findings are documented in this final public report which can be shared with Alberta refineries, researchers, and institutions such as the University of Alberta and University of Calgary.

Knowledge activities completed to date include:

- September 19, 2024; Presentation by Robin Goad on the NICO Critical Minerals Project Development at the Alberta Industrial Heartland Association meeting in Edmonton, Alberta, Canada.
- September 11, 2025; Panel Discussion Participation by Robin Goad for Alberta Critical Minerals Opportunities and the NICO Project at the Alberta Industrial Heartland Association meeting in Edmonton, Alberta, Canada.
- September 2025; Presentation by Cory Smith on the Fortune Minerals Rio Tinto Process Collaboration at a Metallurgical Conference in Salt Lake City, Utah, U.S.A.
- November 17, 2025; Presentation by Alex Mezei of Production of NICO Project Cobalt Sulphate Heptahydrate at Extractions 2025 in Phoenix, Arizona, U.S.A.
- November 27, 2025; Presentation by Robin Goad on the NICO Project Update at the Yellowknife Geoscience Forum in Yellowknife, Northwest Territories, Canada.
- 2026 (Pending) Technical Paper Approved for Publication from Extractions 2025 on Production of NICO Deposit Cobalt Sulphate Heptahydrate by Mezei, Alex, Johnson, Mike, Molnar, Ron and Goad, Robin, 15P.
- Presentation and paper is being prepared for the Met-Soc Conference of metallurgy August 17-20, 2026 in Calgary, Alberta for the Production of Bismuth from the NICO Project.

Sharing the project's knowledge has the potential to significantly impact Canada's critical minerals industry. Fortune aims to contribute to a critical minerals processing hub in the Edmonton area, leveraging the region's existing infrastructure, reagent supply, transportation networks, skilled workforce, and industrial services supported by the Alberta Industrial Heartland Association. By promoting broad

adoption of the project's outcomes, this initiative can strengthen Canada's critical minerals infrastructure and reduce reliance on high-risk foreign supply chains by offering an ethically sourced, domestic alternative to imports from countries such as the Congo and China.

J. CONCLUSIONS

Please provide a narrative outlining the project conclusions.

- Ensure this summarizes the project objective, key components, results, learnings, outcomes, benefits and next steps.

This project was initiated to generate technical data required to validate recent flowsheet optimizations and to support the updated feasibility study, detailed engineering, permitting, and financing efforts for the proposed Alberta Refinery. The program addressed critical challenges associated with the economic and sustainable processing of polymetallic ores and successfully filled key technology gaps that have historically limited domestic refining of complex critical mineral concentrates in North America.

The comprehensive seven-phase pilot-scale program at SGS Canada's Lakefield facility successfully validated core components of the process flowsheet, including optimized flotation techniques, bismuth leaching and cementation circuits, cobalt-bismuth integration strategies, and impurity removal methods. A low-mass pull concentrate was produced, significantly reducing transportation volumes, emissions, and downstream processing costs. The program also demonstrated the ability to recover high-purity bismuth cement and convert it into 99.99% ("4N") ingots, regenerate ferric chloride lixiviant, and pre-neutralize arsenic and iron under mild conditions. These findings enabled the development of a cleaner, more efficient hydrometallurgical process that can process blended feedstocks and produce battery-grade cobalt sulfate and a potentially marketable gypsum by-product.

The project generated critical learnings with broader relevance to the critical minerals industry, including strategies to increase process efficiency, reduce environmental impact, and improve feedstock flexibility. These innovations support circular economy practices, promote supply chain resilience, and lay the foundation for a Canadian-based refining solution that can be extended to other domestic ore bodies, smelter residues, and recycling streams.

The outcomes of this work not only strengthen the economic and technical case for the NICO Project but also contribute to Canada's broader clean energy and critical minerals strategies. By establishing a reliable, transparent, and vertically integrated supply of cobalt, bismuth, copper, and other metals essential to the

energy transition, the project supports national defense, battery manufacturing, advanced technologies, and long-term economic reconciliation with Indigenous communities.

Fortune's next steps include completing an updated Feasibility Study and Feed Study, advancing detailed engineering and permitting, and preparing for construction and commercialization. Continued efforts will also focus on securing additional feed materials and building strategic partnerships to ensure long-term scalability. Collectively, this pilot program represents a major milestone in the development of one of Canada's most strategically important critical mineral projects.