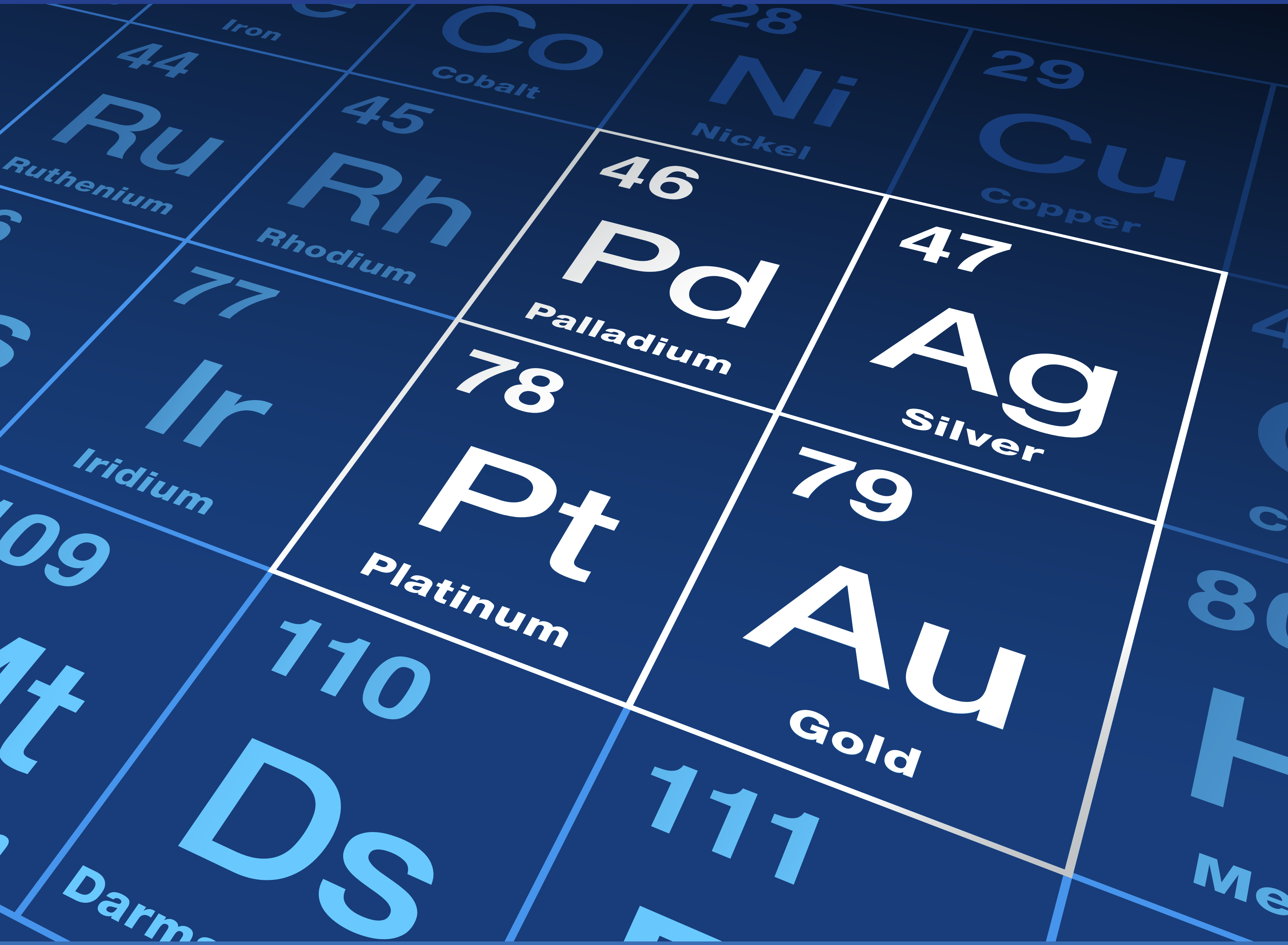
# THE IPMI JOURNAL



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# 2025
# IPMI JOURNAL

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## FOREWARD

It is with great pleasure that we announce the availability of this latest issue of the IPMI Journal, a publication of the International Precious Metals Educational and Scientific Foundation. This digital peer reviewed publication has been created as a vehicle for the sharing of technical information related to the science and technology of precious metals.

Since 1976, the International Precious Metals Institute has championed the exchange of information of critical importance to the precious metals scientific and commercial communities through its various communications vehicles. This has included instructional videos, annual conference symposia, special topic committee meetings, regional seminars, and professional and student achievement awards. Due to the breadth of important topics related to all aspects of precious metals, it has been the vision of the IPMI to provide a publication available to the general public that would become a key source of pertinent topics containing high quality publications by experts in their fields of study.

Historically, the IPMI has attracted world renowned experts in many fields (including Nobel Laureates Henry Taube, Ei-ishi Negishi, Robert Grubs, Professor Ben Feringa, and David MacMillan) in the dissemination of discoveries, inventions, and industry proven practices. Such information has a total value greater than the sum of its parts as seemingly unrelated innovations from distinct focus areas can be adapted to solve problems. Because of this important characteristic of information exchange, we have decided to create a digital journal, available to the general public, that will not only communicate state of the art discoveries and sound practices, but will also review historical communications from the IPMI's archives that have value and use even in today's environment.

On behalf of the members and leaders of the IPMI, we hope you will find value in this new publication.

Dr. Corby G. Anderson, co-editor
Dr. Robert M. Ianniello, co-editor
Mark Caffarey, co-editor
Lou Britton, co-editor

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connecting you to the world of precious metals

# THE INTERNATIONAL PRECIOUS METALS EDUCATIONAL AND SCIENTIFIC FOUNDATION

In 1976, the International Precious Metals Institute, Inc. ("IPMI") was founded to promote the development of precious metal science and technology. Over the past 49 years, the IPMI has provided its members with an extraordinary body of technical and educational work and an exceptional series of technical videos. The IPMI has also recognized leaders in the industry and academia and students through its long standing Awards Program. This recognition and financial support for continued dedication and research in the field of precious metals is a cornerstone of the IPMI and owes its support to generous sponsorships and endowments.

In 2019, the IPMI made a fundamental structural change by reorganizing IPMI into a trade association to focus on its membership and the needs of the precious metal industry. At the same time, IPMI preserved its scientific and educational roots by renaming its original organization the International Precious Metals Educational and Scientific Foundation (the Foundation).

The Foundation will continue IPMI's charitable activities by focusing on its long-standing Student and Industry Awards Program and by continuing to promote the science and technology of precious metals as its primary mission. This Journal of the International Precious Metals Institute is a cornerstone project of the Foundation.

Another of the Foundation's primary goals will be to expand its fundraising activities to ensure the long term sustainability of its educational and scientific work, including new and expanded initiatives, such as the student internship program, designed to attract a new generation of trained professionals to the precious metals industry.

On behalf of both the IPMI and the Foundation, I want to personally thank our Board of Directors, Awards Committee, corporate sponsors, donors, and the benefactors of our endowments for their continued hard work, support, and dedication to our mission.

I would also like to thank Dr. Corby Andersen, Dr. Bob Ianniello, Lou Britton, and Mark Caffarey, who are the co-editors of our Journal. Their research through our historical treasure trove of technical papers has produced an extremely important and relevant body of work. Special thanks to all the authors of the papers in this sixth issue. It is through work like yours that the industry has continued to grow and improve technologically. And lastly, thank you to the sponsors of this publication. Because of your generosity, the net proceeds of the Journal will go directly to the Foundation and help provide continued support for our programs.

Larry Drummond
Executive Director

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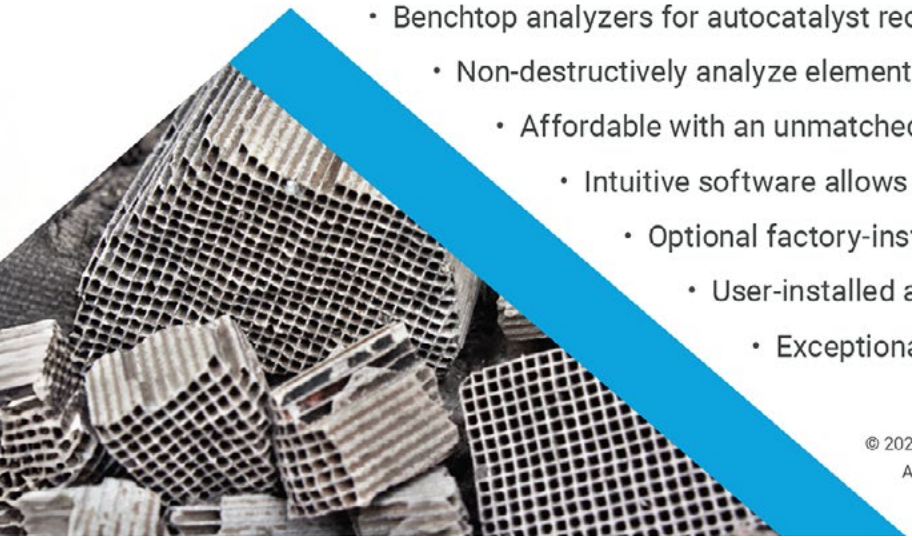

# IPMI® UPCOMING EVENTS

## IPMI Legislative and Regulatory Affairs Annual Meeting

January 27-28, 2026
The Army & Navy Club, Washington D.C.

## IPMI Winter Meeting

February 17-19, 2026
Hyatt Grand Cypress Resort, Orlando, Florida

## IPMI 50th Annual Conference

June 6-9, 2026
Hyatt Grand Cypress Resort, Orlando, Florida

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# PRECIOUS METALS

## VITAL TO THE SPACE SHUTTLE OPERATION

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# PRECIOUS METALS - VITAL TO THE SPACE SHUTTLE OPERATION

by Myron E. Browning
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ROCKETDYNE, Div. of
ROCKWELL INTERNATIONAL

Precious Metals play a vital role in the operation of the U.S. Space Shuttle. Gold is invaluable in preventing possible catastrophic hydrogen embrittlement of the high temperature, high strength nickel base superalloys used in the turbines and other areas. Silver and its excellent properties of oxidation resistance, thermal conductivity and spark resistance is the obvious choice plating of seals and similar support systems. Gold braze alloys bond the cooling tubes used to effect efficient operation of the nozzle. Rhodium and platinum lend superior high temperature properties to coating sensors, support devices and related components. Palladium and iridium are used in braze alloys and thermocouple/sensor elements.

In the operating scheme of the U.S. orbiter vehicle, the main propulsion system employs 3 reusable, high performance liquid propellant rocket engines. Each space shuttle main engine (SSME) operates on a mixture of liquid oxygen/liquid hydrogen at a chamber pressure of 3000 psia, producing an individual unit thrust of 375,000 pounds.

In examining the turbopumps which contribute to this amazing engine, one finds that low pressure and high pressure turbo-pumps are necessary for efficient operation. The high pressure fuel turbopump (HPFTP) is a three-stage, centrifugal flow-type pump, driven by a two-stage hot-gas turbine and supplies hydrogen to the engine fuel circuits. The high pressure oxidizer turbopump consists of two centrifugal-type pumps on a common shaft driven by a two stage hot-gas turbine. The main pump supplies the oxidizer (oxygen) to the main chamber injector, the heat exchanger, low pressure turbo-pumps and pre-burner pump. This permits an increase in oxidizer pressure as it is supplied to the fuel and oxidizer pre-burners.

The vital role of the precious metal gold in the scenario just related is to prevent potential catastrophic hydrogen embrittlement to sensitive structural alloys used to fabricate critical turbine rotors. The 1st and 2nd stage rotors of the HPOTP and HPFTP are gold plated with up to 5 mils of 24 carat gold over a nickel plated diffusion barrier to prevent base metal hydrogen embrittlement damage. All gold plated turbopump components are blister tested at 1000°F for plating, bonding, porosity and processing discrepancies. Figure 1 shows the front and back of a 1st stage HPOTP turbine revealing the critically plated fir-tree areas (at outside edge of rotor) which holds the turbine blades.

Balancing is critical as the turbo-pumps operate near 30,000 rpm. Any areas that are damaged during assembly or test must be replated to assure an ample amount of gold protects the base metal. Often, small cells are constructed to permit gold plating in a small selected area without stripping and re-plating the total component.

In the top view one notices the curvic coupling rings near the center of the disc to facilitate coupling of stages 1 and 2. The lower view shows the bolt holes which require uniform gold plating through-out accomplished by using internal anodes. Figure 2 reveals a schematic diagram of the HPFTP turbine with the gold plating components (rotor assemblies) marked. Figure 3 shows the gold plated rotor components of the HPOTP turbine. The rotor shown in Figure 1 appears at the left of the schematic. A view of the high pressure oxidizer turbo-pump assembly is shown in Figure 3. The two rotor components are shown in relation to the other working parts of the turbine.

Figure 4 shows assembly photographs of the HPOTP and actual pictoral placement of the rotors, the blades and major shaft. The upper section reveals the bolt segments and the fir-trees which are gold plated and receive the precision fit turbine blades after a special broaching operation. The low view shows the reverse view and notes the gold plated areas. The operating HPOTP produces more than 28,000 hp while the functioning HPFTP produces more than 76,000 hp.

Assembly of the SSME 77:1 expansion ratio nozzle, shown in Figure 5 involves 1080 (A-286 alloy) tubes bonded to an Inco alloy 718 jacket. In addition the 1080 tubes must be joined to each other. There are 2160 tube ends to alloy 718 manifold joints. All in-all there is about 4 miles of braze joints. The life expectancy is 240 cycles and 7.5 hours total service with inlet temperatures of -360°F, 6700 psi inlet pressure and 55 pounds/sec liquid/gaseous hydrogen in the coolant circuit. To select the proper braze alloys the following factors were considered:

1. Perform all brazing between 1800 and 1925°F
    - Heat Treat compatibility with A-286 and Alloy 718
    - Minimum of two cycles, but as high as four cycles
2. Thin wall tubes (Nickel Plated)
    - Min. Wall 0.0008 inch
    - Multiple 30-Hour braze cycles
    - Alloy requires minimum erosion
3. Minimize liquation in slow heating
4. Optimize strength and ductility at -430 to +1200°F
5. High remelt of first cycle alloy
6. Corrosion Resistance required
7. High Pressure $H_2$ Embrittlement - 7000 PSI $H_2$
8. Density - Cumulative Mission Payloads at $100,000/LB
9. Cost Limitations
10. Alloy Producibility - Foil, Wire, and Powder

## RESULTS OF ALLOY DEVELOPMENT PROGRAM

A. Required Braze Temperature - Achieved

B. Erosion Characteristics - Excellent

- Approx. 300,000 feet of tubing brazed with no wall thinning
- Data Base includes conditions where alloy actually puddles

C. Liquation Characteristics - Very Good

- Heating rates above solidus slower than 1°F/MIN still produce uniform joints

D. Strength - Excellent

E. Ductility - 25% Elongation

F. Remelt of 1st Cycle Alloy - Adequate

- Minimum of 1000 PSI shear strength at 1825°F
- No Tooling required second cycle

## FINAL SELECTIONS

First Cycle Alloy (R.I. 46)

- CHEM: 35Au, 14 Ni, 10 Pd, 9.5 Mn, BAL Cu (31.5)
- MELTING RANGE: 1800-1840F
- BRAZE RANGE: 1900-1925F
- SECOND CYCLE ALLOY (R.I. 49) - Also third and fourth if required
- CHEM: 31 Au, 10 Ni, 10 Pd, 16 Mn, BAL. Cu (33)
- MELTING RANGE, 1700-1740F
- BRAZE RANGE, 1800-1825F

Figure 6 shows a table of room temperature strength of gold braze alloy wire annealed at 1300-1350°F. It is estimated that some 250-300 troy oz. of wire and foil (gold alloy) is used in the first cycle brazing of the nozzle. The second cycle brazing utilizes some 250-300 troy. oz. of gold alloy paste per nozzle. This combines to total approximately some 550-600 troy oz. for the standard 2 cycle braze of the SSME nozzle. Alloy RI-46 (35 Au-14 Ni-10 Pd-9.5 Mn-31.5 Cu) for the 1st cycle and alloy RI-49 (31 Au-10 Ni-10 Pd-16Mn-33Cu) for the second cycle is presently used.

Figure 7 presents two tensile strength vs. temperature graphs of braze alloy RI-46 with A-286 (lower graph) and with alloy A-286/Alloy 718 (upper graph). The high tensile strength from cryogenic levels to 800°F for the braze alloy over the 0.0015 in. nickel plated base metals is apparent for both the A-286/A-286 and the A-286/718 points.

Figure 8 shows the placement of the gold alloy braze material in the SSME flight nozzle. Foil sheets (0.001 & 0.004") are spot welded for holding.

In Figure 9, we see the A-286 tubes being stacked in the nozzle prior to furnace brazing. Tubes are plated with a special Watt's nickel prior to brazing..

In Figure 10, the monstrous brazing furnace is seen in the open mode (clam shell type) with a nozzle being prepared for brazing. Furnace conditions require heating to 1900-1925°F which requires about 25 million BTU and between 2-6,000 CFH argon for the 35 hour treatment.

Figure 11 reveals a fully brazed flight nozzle being inspected before final assembly.

In addition to the RI-46 and RI-49 gold brazing alloys described previously, there are some seven other gold braze alloys used in SSME applications. These are:

| | | | |
|---|---|---|---|
| NiORO | 82% Au | 18% Ni | |
| Silcoro 75 | 75% Au | 20% Cu | 5% Ag |
| Silcoro 60 | 60% Au | 20% Cu | 20% Ag |
| Cusil | 72% Ag | 28% Cu | |
| Nicoro | 35% Au | 62% Cu | 3% Ni |
| Palniro 7 | 70% Au | 8% Pd | 22% Ni |
| Palniro 1 | 50% Au | 25% Pd | 25% Ni |

Materials are used for brazing valve, pump, pre-burner, seals and flange joints. It is estimated that approximately 100 troy oz. of gold (and silver) are used in miscellaneous non-nozzle component joining.

Silver is used primarily because of its high corrosion resistance and low yield strength (conforms to mating surface topography) for pressure seals and sealing surfaces on some eight functional systems. Sterling silver seal rings and couplings are also employed. Rocketdyne has 2 special silver plating specifications and 5 gold plating specifications for finishing seals, pre-braze components and other sections and devices in the SSME. Platinum, rhodium plus iridium and palladium are used for various special braze alloys, thermocouples, sensors and high temperature elements of the SSME and supporting flight hardware.

Figure 12 shows 2 views of the SSME with power-head attached over nozzle body (lower photo) and being test fired (upper photo).

Figure 13 shows the final space shuttle during its initial launch December 12, 1981 with all precious metals performing as anticipated.

Precious metals are vital materials in the operation of all today's U.S. space shuttles and will be long into the future.

## ACKNOWLEDGEMENT

The author is grateful to the following Rocketdyne Engineers for collecting much of the data presented in this paper:

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Mr. F. M. Kuck

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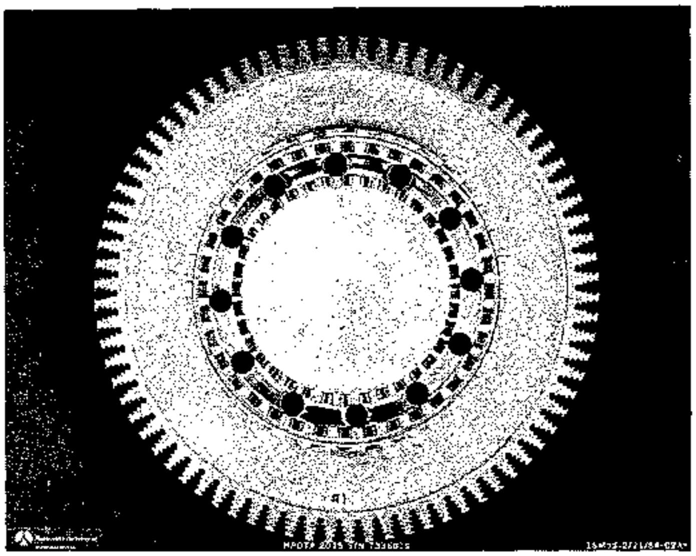
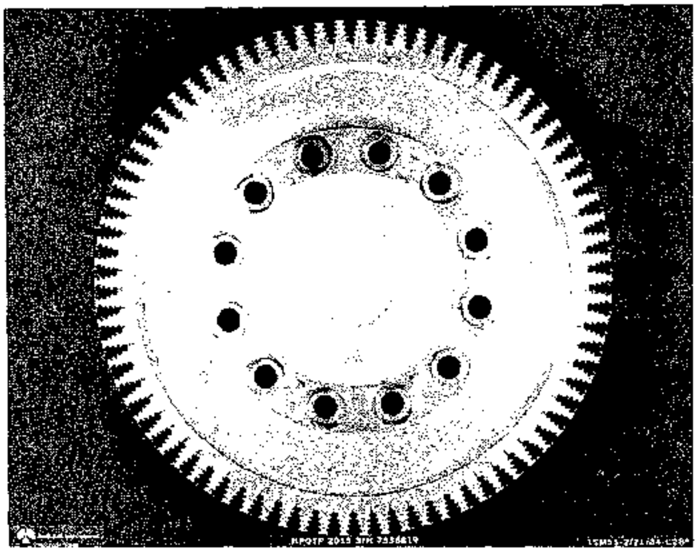


NOTE : Curvic Coupling Rings Near Center of Disc to Mate Stages 1 & 2



NOTE : Bolt Holes Requiring Uniform Gold Plating Throughout

FIGURE 1 Gold Plated High Pressure Oxidizer Turbopump Rotors

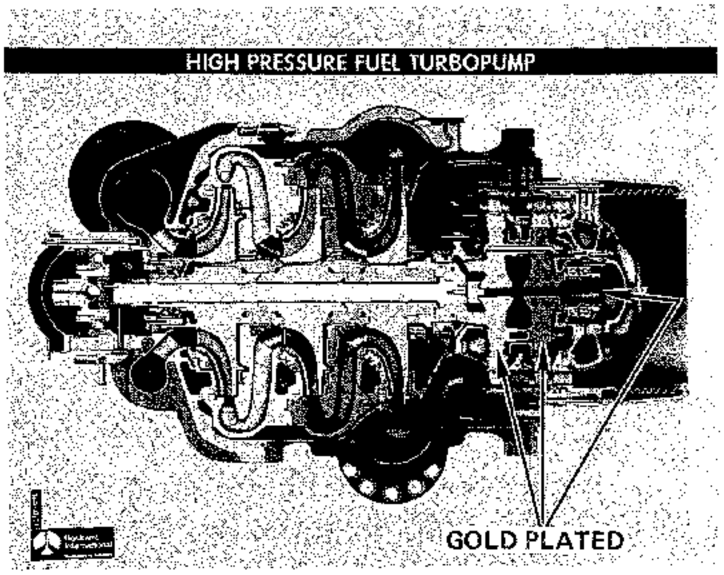


FIGURE 2 High Pressure Fuel Turbopump
Denoting Gold Plated Rotor Sections

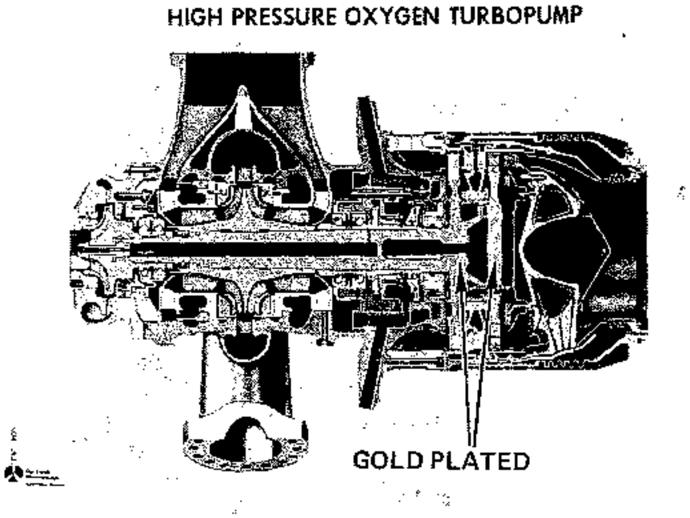
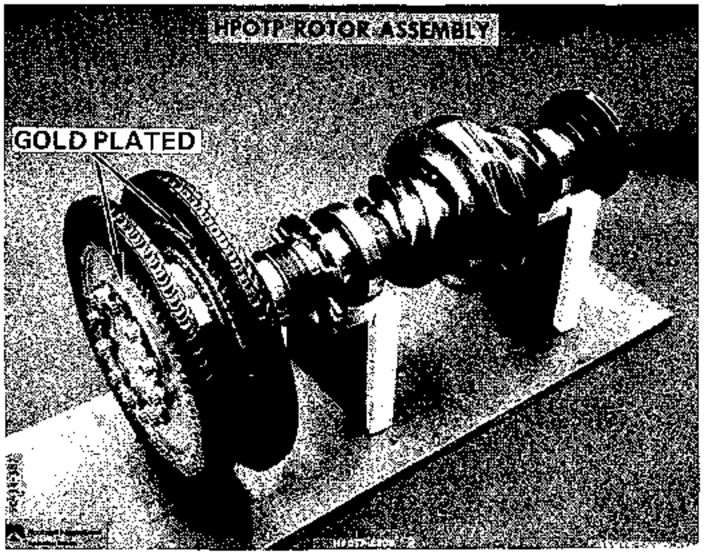
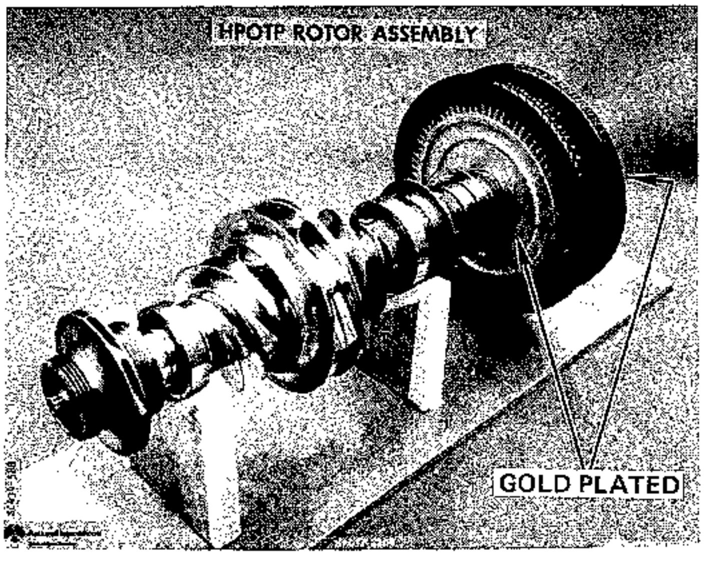


FIGURE 3 Gold Plated Rotor Components
From Turbopump Schematic



NOTE: Gold Plated Fir-trees holding Turbine Blades



NOTE: Gold Plated Flats & Mating Surfaces

FIGURE 4 High Pressure Oxidizer Turbopump Rotor Assembly

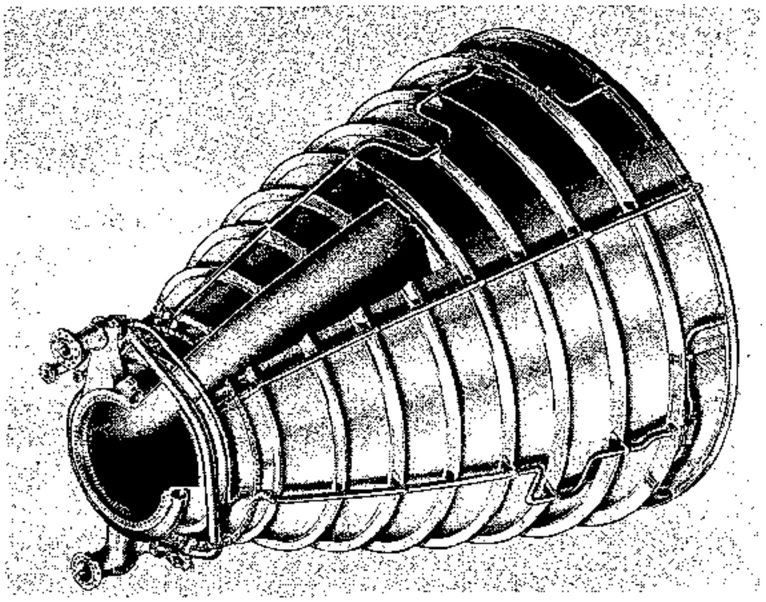


FIGURE 5 Cut-away View of 77.5 : 1
Expansion Ratio Nozzle

# ROOM TEMPERATURE STRENGTH OF GOLD BRAZE ALLOY WIRE ANNEALED 1300°F-1350°F

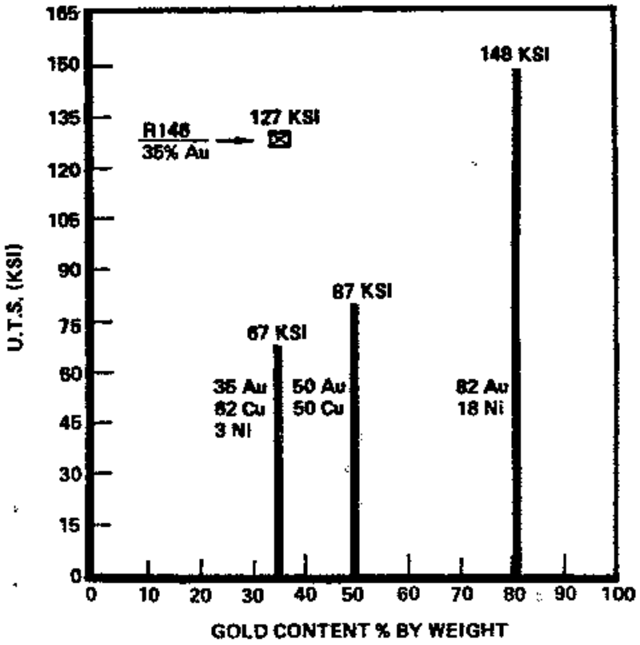
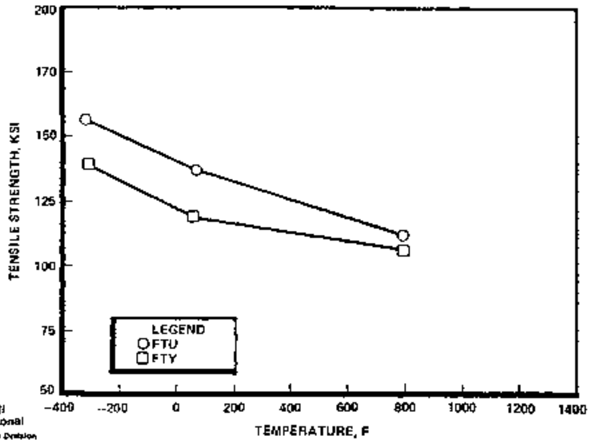


FIGURE 6 Room Temperature Strength of Gold Braze Alloy Wire

## TENSILE STRENGTH A286/ALLOY 718

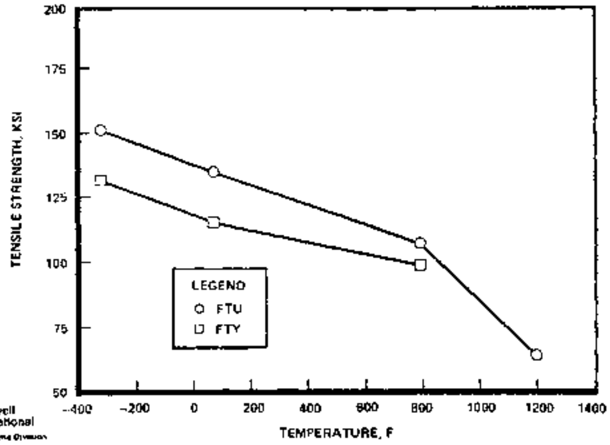
BRAZE ALLOY RI46 RB0170-170
A286/718 – .0015 IN. NICKEL PLATE



Rockwell International
Rocketdyne Division

## TENSILE STRENGTH WITH A-286

BRAZE ALLOY RI46 RB0170-170
A-286/A-286 ~ .0015 IN. NICKEL PLATE



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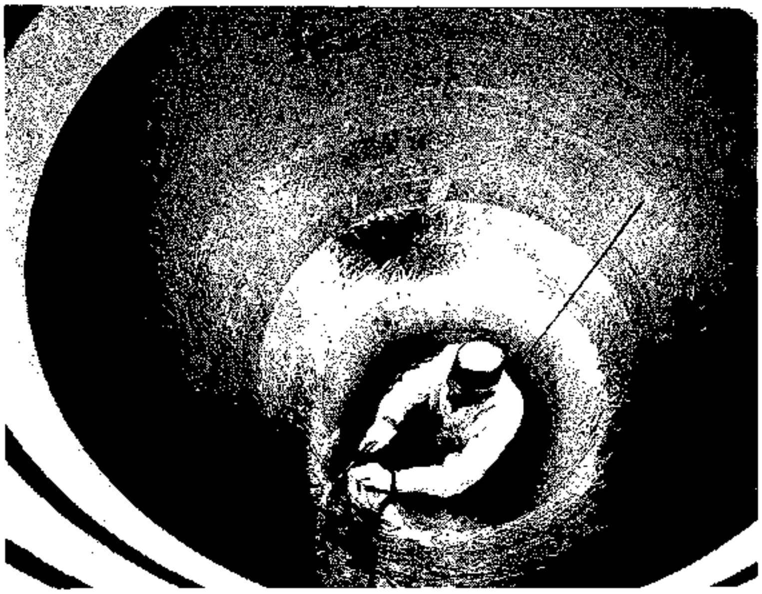


FIGURE 8 Placement of Braze Alloy in Flight Nozzle

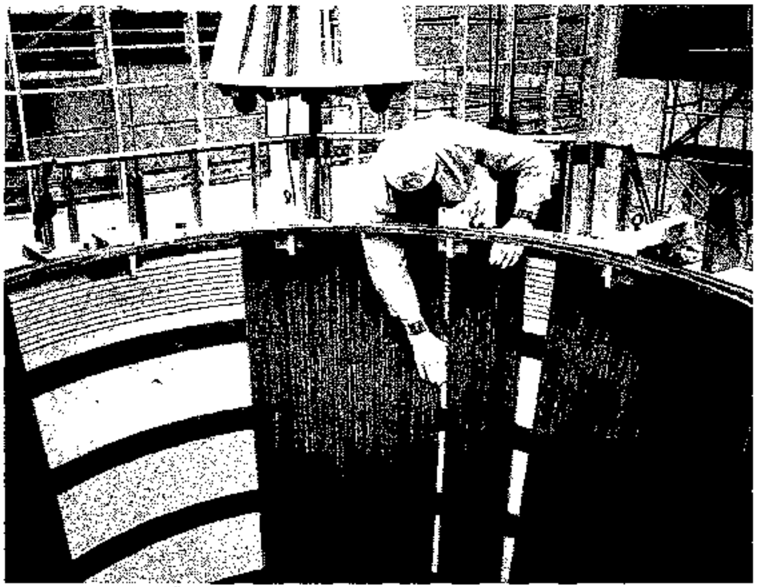


FIGURE 9 Stacking of A-286 Tubes in Flight Nozzle

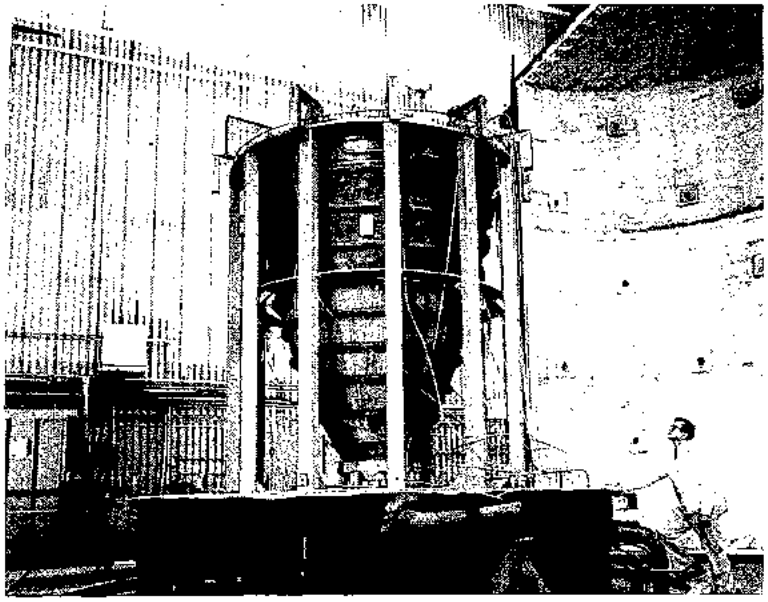
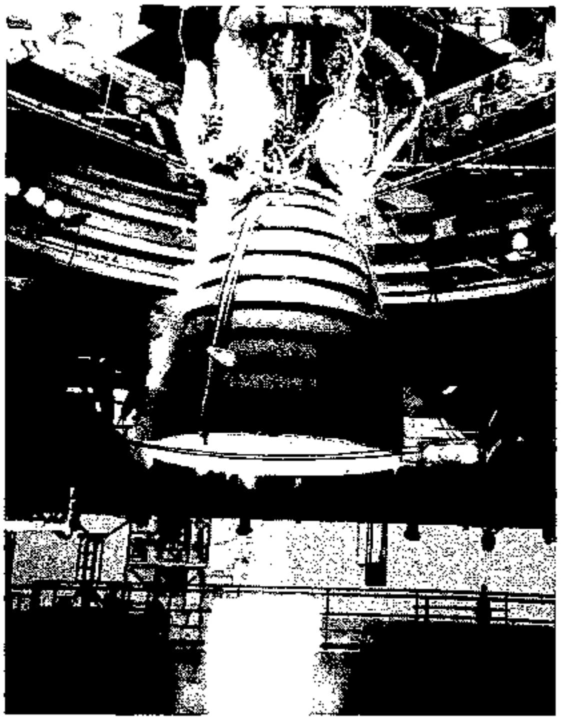


FIGURE 1 0 High Temperature Braze Hearth with Nozzle in Place

FIGURE 11 Fully Brazed Nozzle Being Inspected



Upper View:
During actual test firing

Lower View:
Assembled with powerhead
over nozzle

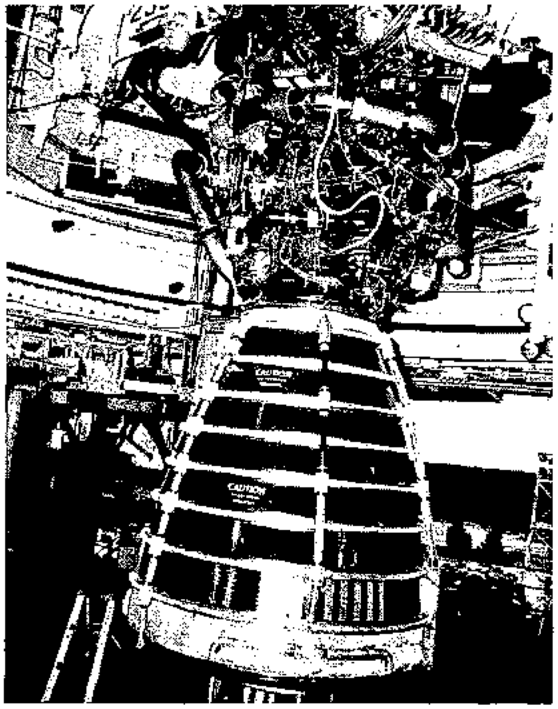


FIGURE 12:
Space Shuttle Main Engine
fully assembled

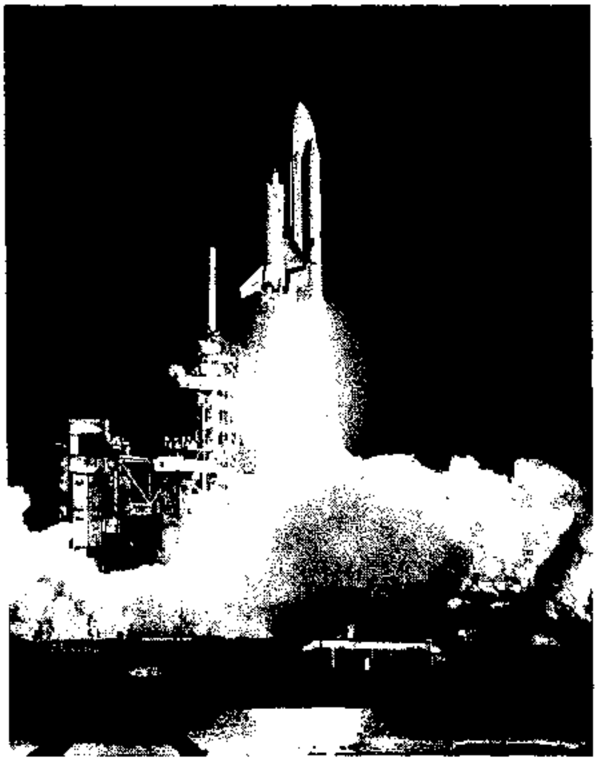


FIGURE 13 Fully Assembled Space Shuttle during maiden flight, December 12, 1981

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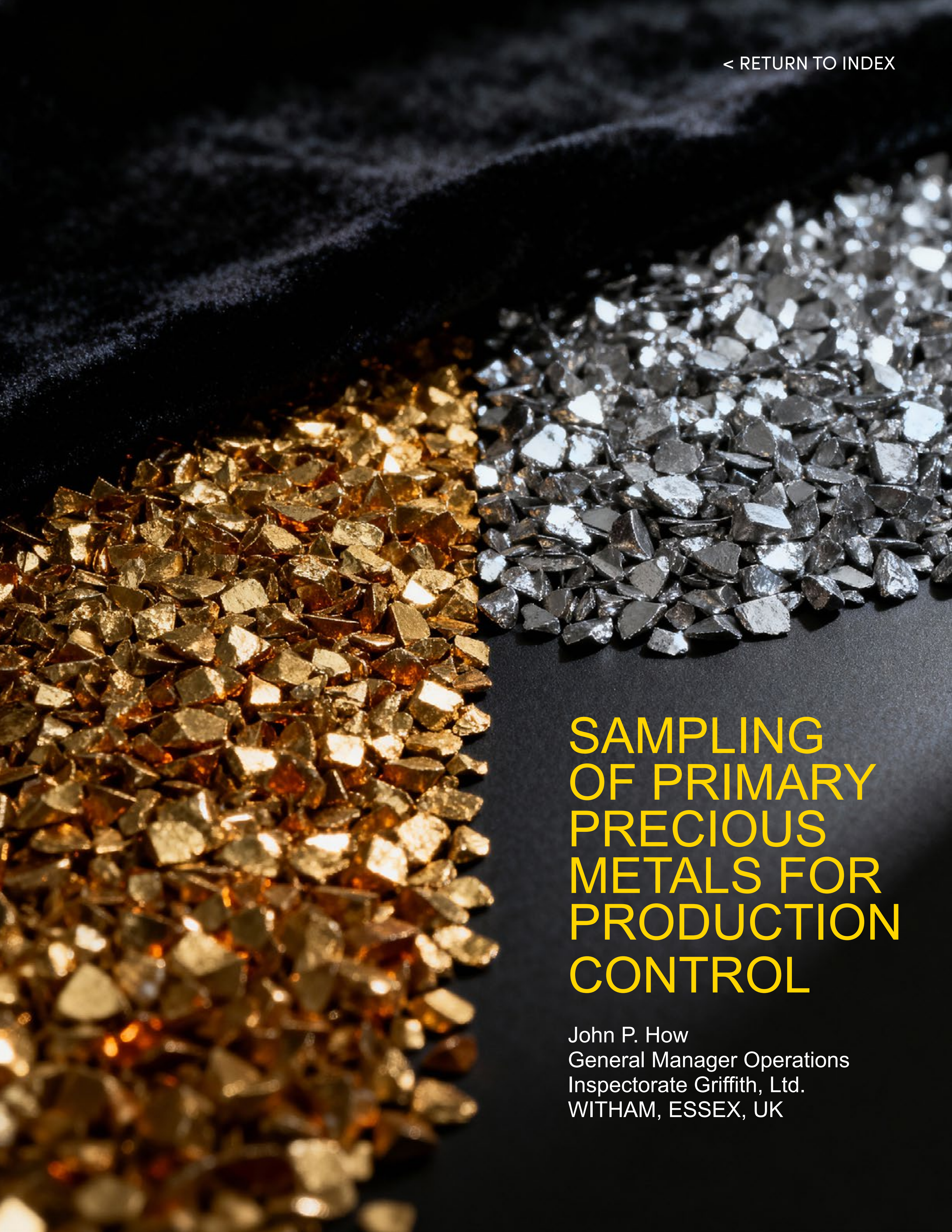

# SAMPLING OF PRIMARY PRECIOUS METALS FOR PRODUCTION CONTROL

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# SAMPLING OF PRIMARY PRECIOUS METALS FOR PRODUCTION CONTROL

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## Abstract

Primary precious metals production is a complex multi operational process.

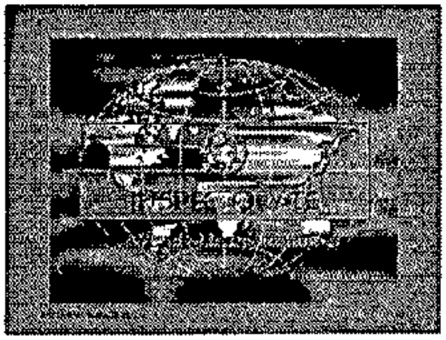
Advances in the technology available now make extraction of lower and lower grades more economic but accurate control of these various operations is becoming more vital to maintain the economics of the operation.

Some details of the current common sampling methods adopted at these various stages are discussed along with some practical improvements that can be adopted to increase the certainty levels of the samples drawn.

Sample preparation for analysis is considered with a check list to maintain integrity of these processes.

## Key Words

Inspectorate, precious metals, sampling equipment, sample integrity, mining, refining



In the following paper I hope to be able to give you a brief view of a few sampling problems we have been involved in at various mine and primary production sites throughout the world, as well as the problems, I hope to be able to show you a number of solutions to some of these problems which we have tried. The results of these solutions, I am happy to say, have been reasonably successful. For the purposes of publication I have only included a few of the photographs and drawings from the presentation which I hope will assist the reader in gaining a greater appreciation of some of the problems.

*Inspectorate* do not claim to be *experts* in the solution of primary material sampling problems but we do have a worldwide coverage and through our experience at different sites in vastly different parts of the world we have been able to gather together a wealth of experience which through practical application we have been able to use to the benefit of a number of our clients.

The problems I will be describing and some of the possible solutions have been gathered together for this paper from work in Europe, Russia, South Africa, India as well as both North and South America.

Not all of our proposals have been adopted by our clients in their entirety, and one or two of our initial proposals have not resulted in startling immediate improvements to the accuracy of the samples obtained, those that were less than totally successful, on initial implementation, will not be mentioned in this paper, as some are currently under investigation and various improvements have yet to be statistically analyzed.

I mention the fact that we have had these odd poor results merely to emphasize the extremely complicated nature of some of the sampling problems and that we have always tried to propose practical solutions which would provide a substantial improvement at "as importantly" economic rates. We believe there are very few sampling problems that cannot be over come given enough time and money. The trick is to adopt a system that is as near fool proof as possible without costing more to operate than the entire product in question is worth.

Most primary production starts with some sort of hole in the ground be it an open cast operation or a deep mine.

The geologists will have done their work in plotting the mine and outlined the areas for production of ore.

Production control at this stage is handed on to the mine staff who need grade information relating to the ore as it is being mined or removed from the pit.

## Blast Hole Sampling

Blast hole drill sampling is usually the first sampling exercise to be submitted to the laboratory for analysis.

The accuracy of this sample is critical to mine development planning and all that can be done to gain an accurate sample at this stage will pay dividends at a later stage.

The biggest problem encountered at this stage is one of contamination of the sample, which can take the form of both enrichment or down grading, both must be avoided if meaningful results are to be obtained.

Working underground can exaggerate these problems especially where the face is narrow and head room is limited.

Catching the chips from the drill head is the normal method of obtaining a sample for preparation for analysis. The use of water or compressed air to flush the hole during drilling provides its own problems of actually catching a representative sample of these chips.

The use of a cowl around the drill stem will deflect the majority of drillings into the receiver which can be lidded and sealed for removal for preparation. If the cowl is designed to be free standing it will not inhibit the sight lines of the driller or interfere with the drills operation, and will certainly provide a better sample than the more traditional container placed below the drill stem catching a supposedly random sample of the chips as they are ejected from the hole. The sample collected will usually be larger than that normally obtained which usually leads to a more accurate sample but requires careful preparation.

Care should be taken whilst removing water from this sample at the face not to loose the finer particles. A small siphon or air driven pump gives better results removing the water after a short settling time, rather than attempting to pour the water off the top, which usually leads to losses of these finer particles.

If core samples are being taken these should be carefully handled and the use of appropriate core sample boxes will reduce the chance of loss and contamination by mixing.

The traditional method of sampling at open pit drill sites is to use a segment container to catch a proportion of the sample as it is ejected from the drill hole.

This segment sample box can suffer from a number of problems. How tall should the container be? Too tall and you loose some particles which are not ejected high enough to reach the lip, too short and there is a danger of loosing particles as they bounce in and then out again.

How large a segment should you take? 1/4th, 1/8th, 1/16th. Again small gives an easily handle-able sample but there is a danger of only catching the small particles which can bias the sample.

How close to the stem should the container be placed? Too near and there is a danger particles will hit the side and fall back into the hole, too far away and again there is a danger that particles will not be ejected far enough to enter the sample container.

If a segment container is used it obviously must be placed at the correct angle to be truly representative. Painting the outer face helps with alignment but this is still operator dependent. What ever method is used the primary sample is usually reduced on site prior to dispatch to the laboratory for preparation and analysis.

The static riffle is usually the most common reduction method and with it comes the usual problems of its correct use and appropriate sizing, I have rarely seen an appropriate size riffle in use on site as the riffle has to be easily handleable and this usually leads to a smaller one than is strictly necessary.

A number of these points could be considered minor and indeed individually they probably are, but their combined effect can lead to erroneous figures being provided which can have enormous on cost's.

A small air driven mobile sampling unit can overcome the majority of these problems and provide a better prepared and more accurate sample for analysis.

If a low sided split circular tray is placed around the entire drill stem this will catch almost the entire core ejected from the drill stem and material can be removed either manually or with a vacuum lance to a hopper feeding a small crusher, sized to provide a product for a static riffle mounted below. The first split from the riffle can be refed to the crusher and riffle to reduce it as necessary to an appropriate sample for submission to the lab.

The vacuum lance is the easiest system to operate for the drill gang as the sample can be continuously removed from the collection tray both safely and without losses. Manual removal of the sample is a little more labor intensive but the use of correctly shaped high sided shovels will ensure no losses occur during transfer to the crusher.

## Crusher Feed Samples

Having removed the ore from the mine or pit some form of primary crushing is usually necessary. Sampling this crusher feed is usually some form of stock pile sampling be this entirely manual or with the use of mechanical equipment.

A reasonable sample can be regularly drawn with a front end loader, but this will always need considerable preparation prior to reduction and analytical sampling, it is not unsurprising that the majority of mines do little if any sampling at this pre crushing stage and more weight is given to the samples obtained from the drill cores.

## Mill Feed Samples

Post crusher sampling is a different matter and the easiest place to perform this is on the feed belt to the primary mills.

Variations of cross stream, cross belt and stopped belt sampling are in use the more mechanical the system the better the sample tends to be. The triggering of the sample drawing is however of importance and the weight based samples using a belt weighing system tend to be more accurate than a pure time based system. Correct sizing of sample devices is vital as is the ability of the sampling device to collect a complete sample including any very fine particles, and most importantly any moisture which can be free water at this stage. The efficiency of most cross belt samplers can be greatly improved by using a belt profiler at the sampling point, either permanently in position, or to reduce belt wear, raised just prior to operating the sample sweep.

The treatment of this sample should take into account this moisture content so some form of sealed container is usually necessary to hold the bulk sample prior to preparation on a shift or daily basis.

The mechanical samplers should include some sort of belt loading trigger which will prevent a sample being drawn, either time or weight based, whilst the belt is not fully loaded, as this can bias the bulk sample.

## Concentrator Feed

Output from the primary mills is usually fed to some form of concentration in most cases these are solution based so the input and output is in the form of a slurry.

Most mines draw a number of samples throughout the concentrator process at the various stages. The three most important however are the feed the output and the tails. Without a reliable result at these three points it is impossible to gauge the efficiency and the expected outturn of metal from any further processing.

Concentrator feed is usually a thick slurry ex the mills. Some operations use a two or even three stage milling process to provide a fine enough feed to the concentrator. Concentrator feed rates can vary from a few hundred gallons an hour to many thousands of gallons per hour. Figure 1 shows a twin stream input from a bank of 4 mills running at approximately 1000 gallons an hour each.

*Figure 1.*

The most common sampling device in use is the cross stream pulp sampler. These adopt various similar designs on the basic principle of a traveling funnel with the sample being led off to some form of further reduction. Figures 2 and 3. Shows 2 slightly different types of pulp sampler.

The early models have twin stainless steel adjustable knife edges and open topped straight backed with an outlet spout leading to an open launder below. The use of strobe photography and video can show some of the problems associated with these cross stream cutters.

*Figure 2*

*Figure 3*

The stream can become turbulent immediately in front of the cutter face, as can be clearly seen in Figure 3, so it may be necessary to lengthen and clam the inlet chute to ensure an even flow is delivered to the sample head.

Experimentation with both wooden and hanging chain baffles has been found to have the desired effect in especially turbulent flow streams.

With an even flow, the rate of travel, of the sampling device across the stream can be adjusted to give the least disturbance to the flow and opening the jaws to well in excess of the theoretical three times top size will have beneficial effects.

This does not however solve the main problem with this type of cutter, in fact it usually exaggerates the problem in that the cutter is rarely able to cope with the flow of solution and a proportion floods over the top and escapes through the adjustment slots. This splash in a number of cases has been found to end up in the open launder and spread liberally around the area. Figure 4 shows the sampler causing a bow wave to splash over the open top of the sampler whilst in Figure 5 the sampler is totally lost in the turbulence caused by it's travel.

The samplers can be single pass or twin pass, if twin pass, there should be a sufficient delay to ensure the stream has settled prior to any second pass.

Most of the early systems have some sort of static water spray at the stationary end which is designed to wash down the sample chute after the sample is taken if these are incorrectly aligned they can either be in effective or can wash in some of the splash from the surrounding area. The launder usually delivers directly to a vezin type sample cutter single twin or quadruple dependent on the volume of sample required, the sample is then collected in a suitable container.

The samples are usually combined with a number of lots and pressure filtered every 2-4-or 8 hours to form a damp cake which is sent for preparation.

The newer type of pulp sampler has been developed from the early models to eliminate a number of problems, the internal knife edge adjustment bolts are

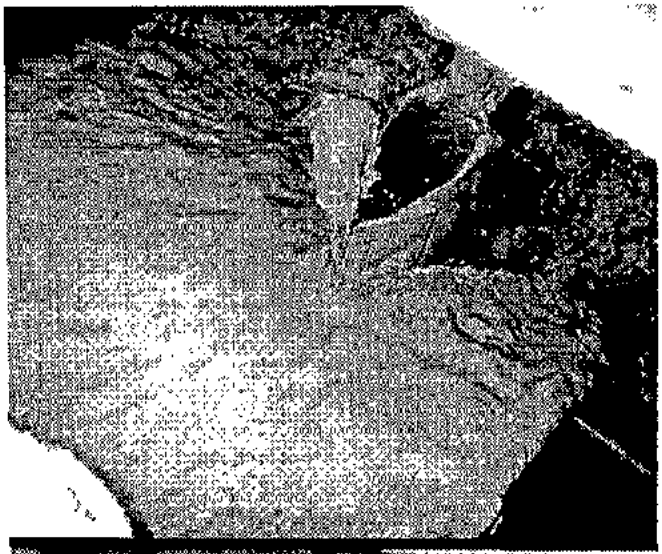


*Figure 4*

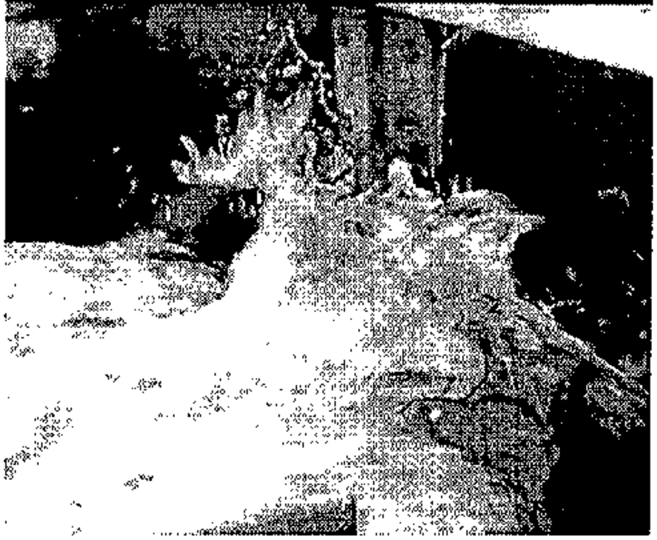


*Figure 5*

replaced with set screws giving a smooth internal surface to the pulp cutter to prevent disturbance of the flow. The adjustment slots are now only in the actual knife edges which has stopped the sample flowing out of the earlier design of slots. Small plastic inserts cover the adjustment bolts presenting a smooth face to the pulp stream reducing any disturbance to the flow. The top of the sampler has been enclosed and there is a wash down spray head installed, connected to a flexible pipe. The cutter is generally larger than the early models and the base is bulbous to accommodate the full flow of sample without forcing some sample out of the bottom of the cutter due to the high flow rate. Figure 6.

The open connection to the launder has been replaced by a flexible tube directly connecting to a flow control hopper. This flow control hopper has twin spray heads installed in the top to provide wash down. At the base of this flow control hopper, is a small ring collar with a

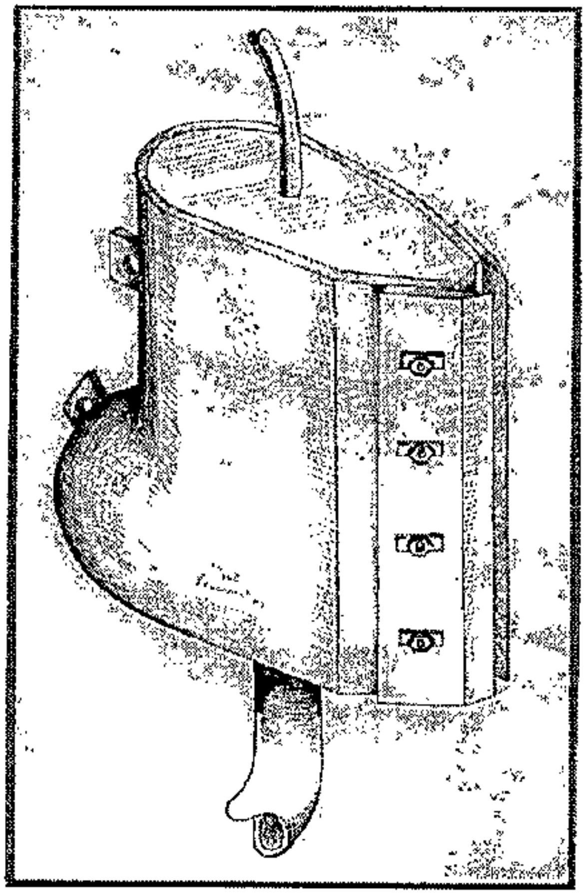


*Figure 6.*

series of air inlet holes. Compressed air is fed to this ring which has the effect of providing a mixing action to the slurry in the flow control hopper keeping the heavier particles in suspension and enabling a well mixed product to be fed under control to the vezin type cutter. Figure 7.

The vezin cutter has small spray heads installed in its top to provide final wash down. The entire sampler assembly is now controlled by a small micro processor which ensures all equipment sprays and compressed air are operating prior to drawing the sample and is closed down following a satisfactory wash down cycle.

The triggering of the main pulp cutter is determined by a combination of flow and density of the feed material rather than the more normal time based systems. The flow meter also ensures the sample is only drawn at normal flow rates rather than at low flow times. A header tank arrangement can ensure flow rates can be evened out over short periods of low or excessive flow.

The concentrate then is passed to some form of thickener and in a number of cases settling tanks.

## Concentrator Output

The settled concentrate is then filtered to provide a wet concentrate of varying moisture content.

Filter technology has advanced considerably and it is now possible to produce a damp cake containing only a few percent moisture although some high volume producers are limited to the larger continuos vacuum drum filters which produce a fairly wet cake, which on settling in collection bins, invariably produces free water on top, which is pumped off.

Sampling of the concentrate presents many problems largely related to the condition of the concentrate as presented. In essence the higher the moisture content the greater the problems. All mines would like to have a dry fine con-

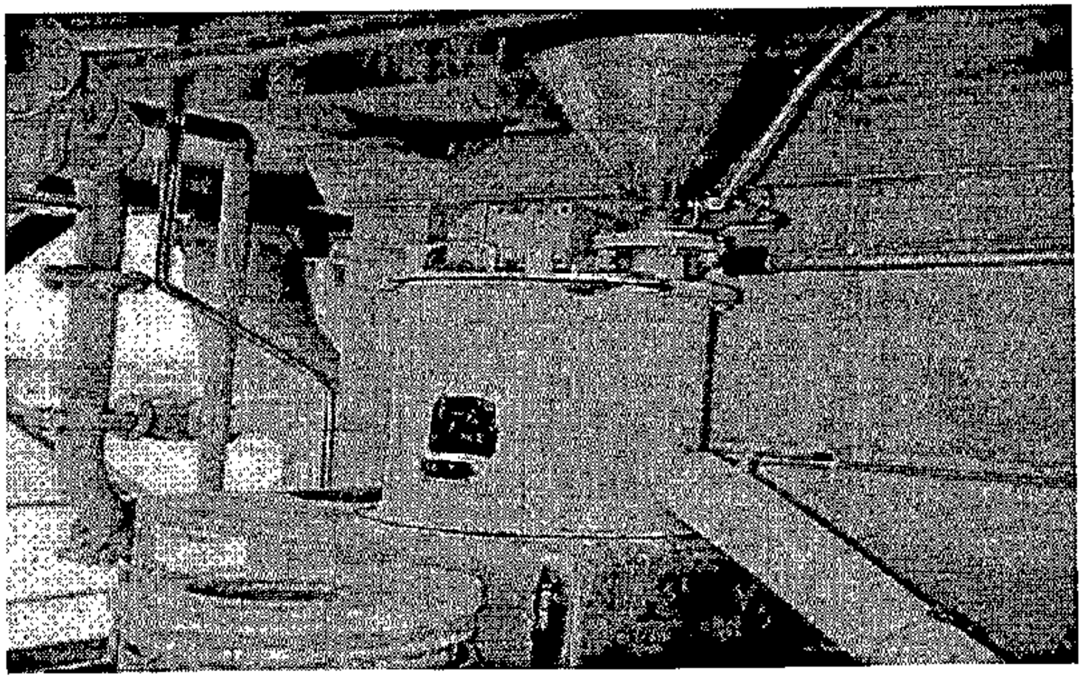


*Figure 27.7.*

centrate product and the smelting or chemical refinery manager who receives the concentrate would be greatly relieved also.

Sampling of such a product would be a great deal easier and accuracy could be assured with some basic equipment. However cost dictates that in all but a very few operations totally drying this concentrate is not an option.

Our friends with operations near to the equator take full advantage of any sun to assist in these operations and those in the more northern or southern climbs have developed some ingenious uses for waste heat from any source, in some cases purely to prevent freezing. In most cases, however, a less than perfect concentrate product is delivered to sampling and further processing.

The smaller bulks tend to be the dryer types of concentrate and these are sampled by a number of varying methods. In one instance a giant dough mixer is used prior to drawing a sample using a small slotted pipe auger. In another case a continuous sample is drawn from a drum vacuum filter by a traveling scraper/scoop. The sample being washed into a container using a controlled and metered amount of water.

The most common method still tends to be some form of auger with mechanically driven types being in the majority. These augers can be mounted on gantries to sample directly from lorries or placed above storage bins for sampling prior to dumping into process. Most have an outer core and a mechanically driven set of flights. The sample either being retained in the flights within the core or discharged into a collecting container from the top of the flights. Experimentation with these augers has found the best method for each particular situation dependent on both the type and condition of the material being sampled. The variations include lowering both core and flights together into the concentrate whilst the flights are turning, to lowering just the core then powering the flights down into the material within the core.

Each method needs to be tried preferably using a known concentrate which has been adapted to resemble the condition of the main product. A number of sets of samples should be drawn and as much statistical data assembled before a final procedure is adopted. Obviously the more sample that can be drawn from as many places as possible the better, I have seen a 6 core and flight assembly in use on a fixed gantry which can sample a lorry and trailer assembly in only a few minutes following loading from a belt system which gave a very even spread through both lorry and trailer.

The concentrate now moves on for further processing be this either hydrometallurgical or pyrometallurgical, the various products and byproducts now become both more concentrated and more homogeneous which relieves a number of problems in the sampling field.

## Refinery Sampling

I do not propose to elaborate on the methods adopted, as these are well known and an enormous amount of work has been done and continues to be done to refine the accuracy of the equipment and methods being adopted.

I would however sound one word of warning, the waste streams from the processes are as important as far as sampling is concerned as the product streams and it is sometimes the case that the sampling of slag from a blast furnace or the final polishing filter product is not given the priority it sometimes warrants.

Recycling slag is both costly and embarrassing for the smelting manager, but it is far better to discover a problem early and recycle the odd load of slag rather than contemplate recycling a small mountain due to in accurate sampling prior to dumping.

I have, so far, said little about methodology but it is of little use having the best most appropriate equipment unless that equipment is used correctly.

The quality of the sample can be severely compromised by the person drawing that sample if he has not been adequately trained and fully understands the importance of the actions he is asked to perform.

When we are asked for assistance our recommendations are based on best practice, but also we try to design systems that are easier to operate correctly than incorrectly. The inventiveness of some of the work force demands admiration but unfortunately this inventiveness sometimes leads to short cuts which can severely compromise the end result.

The more mechanical the sampling system generally the better, I avoid the word *automatic* as I have yet to see a bulk sampling system that does not require a considerable amount of expert input to maintain its efficiency and accuracy. If you can design a system to be accurate and easier to operate correctly than incorrectly the chances of error are greatly reduced.

I can recall a site where there were never any failures of reject slag to meet the PM levels for dumping on Friday nights or all day Saturday and Sunday. Everybody was happy, the sampler who was short handed at the weekends had devised an easier method of sampling. The blast furnace foreman did not need anybody to drive the shovel to recycle the slag so he could use the extra man to cover where he was short handed and the chemist could perform the analysis very quickly as PM levels were always so low. The tiny bead left on the cupel following fire assay hardly needed weighing let alone parting to determine separate gold and silver levels.

The installation of a simple mechanical sampling device revealed that slag's at the weekend had a reject rate of approximately 10% which compared almost exactly to the reject rate of the Monday to Friday slag's. These method modifications are usually not deliberately designed to give inaccurate results, it is merely a lack of understanding of the consequences of a minor action.

In one plant we issued every man with a $5 trigger operated hose nozzle and put quick release fittings on the hose points. This resulted in the pulp samples being filtered and washed without losses experienced in the past when the operators used their thumbs on a 2 inch fire hose to wash the samples. In another area we supplemented the written working instructions with stick man drawings that were laminated and attached to the sampling points.

These combined with through training for the operators improved the quality of the sample derived.

A minor modification to a sampler prevented unauthorized speeding up of that sampler with resultant results improvement. All these unauthorized modifications were discovered usually after a couple of days work on site and invariably at 3am in the morning. The modifications were discussed with the operators prior to their implementation so that certain benefits for them could be incorporated. Increasing the size of a platform or installing a shelf to rest a sample container on cost only a few $ but the shelf stopped half of the sample being tipped on the floor because the container was too heavy to lift.

## Bulk Sample Preparation

Having concentrated on improving the quality of the production control samples at the point of drawing them equal attention must be paid to the preparation of these samples for the analytical results to be meaningful.

The bulk sample should be sufficient to obtain an accurate sample but not so large that its preparation becomes a problem.

If the volume of your pressure filters is only 25ltrs there is little point in drawing a 50ltr sample as despite instructions to the contrary 25ltrs of the excess solution will probably be poured off prior to filtration.

If 50ltrs is considered the minimum sample size either use a 50ltr filter or modify the filter with a 25ltr funnel hopper so that the operator can empty the sample container in one go and top up the filter with a two way valve as half of the material is filtered.

If it is really necessary to draw off excess water prior to filtration provide a small siphon, air or water driven, which will remove water after settling without loosing the fine particles which usually happens if attempts are made to merely pour off excess water.

A couple of sheets of scrap computer paper placed below the filter paper on the pressure filter will prevent the filter cake and wet filter paper from falling apart during transfer to the container. The computer paper does not inhibit the filtering action and has a higher wet strength than filter paper alone. Once dried any residue can easily be brushed off or if incinerated it will contribute a negligible amount of ash to the sample and can save any loss of sample during preparation and transfer from the filter.

Sample containers should be clearly identified, the lids also which should if possible be snap fit and secure. We have all dropped a sample in our time but very few of us were brave enough to admit it. With a secure container the chances of loss or contamination are limited.

Accurate recording and observation is vital, if you use a paper system, use pre printed forms tag labels etc. Ensure all the boxes are relevant then insist all boxes are filled in. If a PC based recording system is used make it impossible to miss out any details that are considered vital.

Ensure there is adequate maintenance of sampling equipment and calibration on a regular basis of any measuring equipment, scales, ovens etc. There will always be pressure on maintenance to keep production going but it should be remembered that production without correct monitoring and control can be a costly omission. Discourage the temporary "fix" as "temporary" has a multitude of meanings. I have witnessed a temporary fix that last 4 years and was introduced to all similar samplers as a "modification". The sample was biased but the equipment never needed attention. Re engineer out minor problems rather than "temporary fix" them.

## Laboratory Sample Preparation

Preparation and sample reduction within the lab can be more easily controlled. Attention to cross contamination problems must be up most in the lab managers mind.

Extracted booths to work in but control the extraction rate carefully. Vacuum lines for clean up of working areas and equipment after use will give better results than brushes. A quick wipe down with a methylated spirits soaked paper towel will quickly provide a clean dry working area. The paper towel disposed of rather than using cotton wipes which merely spread the dust collected the next time they are used.

Enough equipment so that screens can be regularly washed preferably in ultrasonic tanks and then oven dried, plus all the general basic lab hygiene and safety rules which must be taught and regularly monitored and re-taught. 15 minute refresher courses on 1 specific topic can be fitted in every month and can reinforce the quality of the preparation area output.

Having obtained a dry fine sample in the laboratory it's accurate reduction to a suitable size sample for the chemist becomes vital. Mixing and reducing by the traditional coning and quartering method with rolling on rubber sheets is adequate if well performed. The installation of a spinning riffle can however eliminate some of the human error failings of this method whilst giving a faster through put.

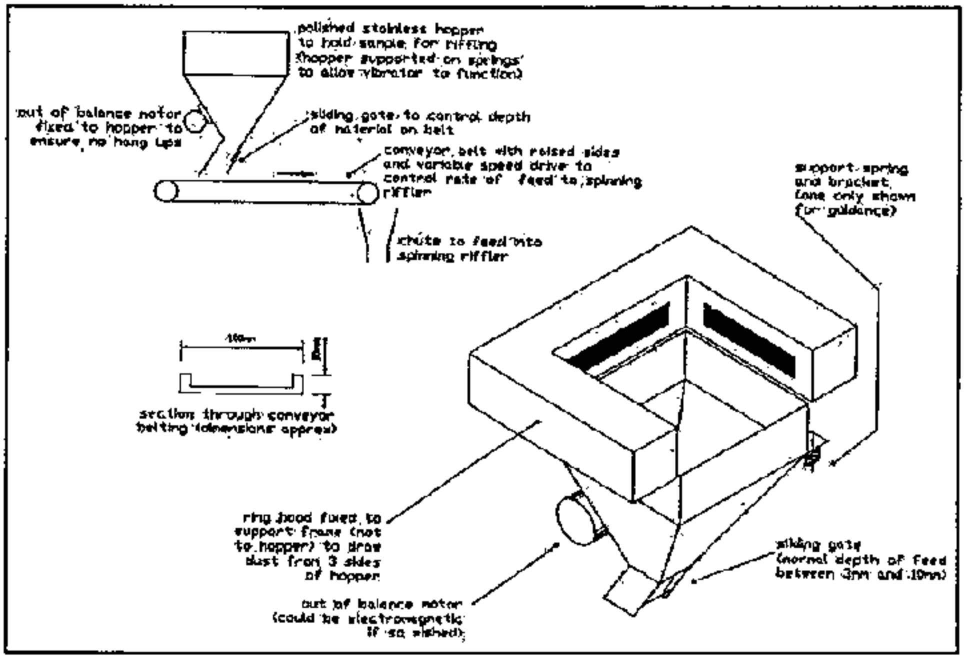


*Figure 27.7.*

A belt drive feeder with associated vibrating feed hopper can greatly speed this operation whilst eliminating some of the segregation problems associated with vibrating chutes. Large prepared samples can be quickly reduced accurately to analytical size samples and the open design of the equipment makes cleaning both fast and efficient leading to greater overall output. Figure 8 Shows an engineering sketch that was used to locally manufacturer a suitable hopper and belt drive unit feeding an existing spinning riffle thus avoiding the problems associated with the original vibrating feeder.

I have not traveled all the way from London just to inform you that stick men drawings will solve all your problems. They won't. Nothing I have said today is new and I saw a few nodding heads during the presentation. I would encourage you to take a fresh look at your operations. Question everything you do. Get someone else to look at it for you. Yes, I would love to come in and help but you can do this by using your own staff from other sites. Be honest and open with each other, share your successes and also admit your failures. None of us are perfect. We can all learn from each other. Our industry is vital to the world economics and the more efficiently we run it the better for us all.

I hope this presentation has sown some seeds in your minds which you can use to benefit your own operations. It is based upon primary production problems but those of us who have worked in the secondary recovery fields as well will, I am sure, recognize some of the problems and hopefully will find some of the ideas of use.

Finally I would like to acknowledge the help and assistance given to me in compiling this paper. This list is too exhaustive to be mentioned here and a number of our clients naturally are not listed as the work we have performed on their behalf is confidential. Their and others help was invaluable in compiling this paper.

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# INDUSTRIAL APPLICATIONS OF PLATINUM GROUP METALS: IMPROVING THE QUALITY OF LIFE

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# INDUSTRIAL APPLICATIONS OF PLATINUM GROUP METALS: IMPROVING THE QUALITY OF LIFE

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## ABSTRACT

Use of the platinum group metals in four industries worldwide--electrical/electronics, chemical, glass, and petroleum--is outlined in specific applications. Despite the widespread use of PGM's, few consumers are aware of the role that PGM's play in their everyday lives. This role is demonstrated through a review of PGM's contribution to four areas which typify today's quality of life: the modern automobile, the contemporary home, our current lifestyle, and the high-technology world in which we live. Emphasis is placed on what the absence of PGM's would mean to the alternatives consumers have to improve the quality of their lives.

## KEYWORDS

Platinum group metals, industrial applications, electrical/electronics, chemical, glass, petroleum, unique properties, quality of life.

As you have been told, I am here to speak to you this afternoon about the six platinum group metals and their use in industrial applications.

Since a review of the use of PGM's in auto catalysts, jewelry, and health applications has been given by others who have or will be speaking to you shortly, I am going to discuss the use of platinum, palladium, rhodium, ruthenium, iridium, and osmium--and their alloys--in the electrical/electronic, chemical, glass, and petroleum industries worldwide--essentially, the "all other" or "everything else" category. These four industries account for somewhere between 18% and 22% of the total usage of PGM's worldwide.

Let me first summarize how PGM's are used in each of these industries.

The electrical and electronics industries consume about 6 to 7% of worldwide usage of PGM's. The most commonly used metals are platinum, palladium, ruthenium, and rhodium, and they are used in a variety of applications which include: low-voltage, low-energy contacts; thin- and thick-film pastes, circuits, and components; multi-layer capacitors; semiconductors; thermocouples for a wide range of industries and industrial applications; a diversity of sensors, electrodes, voltage regulators, thermostats, switches, and meters; and fuel cells--which, as you know, are still only in the experimental stage.

The chemical industry consumes about 5 to 6% of worldwide usage of PGM's, primarily platinum, palladium, and rhodium. The largest use is in platinum-rhodium gauzes employed in the production of nitric acid. Other major uses in the chemical industry are as process catalysts functioning generally as chemical intermediates. Among the chemical processes which use PGM's as catalysts for organic synthesis are hydrogenation, dehydrogenation, and isomerization. PGM's are also employed to produce laboratory apparatus, such as crucibles, dishes, beakers, boats, and other implements for environmentally specific or critical reactions and processes. Platinum is also utilized directly in anti-cancer drugs. In the chlorine, electronic, automotive, construction, and waste treatment industries, PGM coated titanium anodes are used as a replacement for graphite.

The petroleum industry consumes about 3 to 4% of PGM's worldwide, primarily platinum, palladium, and rhodium. The predominant processes employing PGM's are: reforming and isomerization to produce high-octane gasoline feedstocks and aromatic chemicals; hydrocracking to yield transportation distillates, heating and fuel oils, and light gases; combustion promotion in catalytic fluid cracking units; and others in the manufacture of specialty petrochemicals. PGM's are also used by the petroleum industry to reduce pollution from fossil-fueled power plants. The power generation, petrochemical and chemical industries are increasingly reliant on PGM based catalysis to eliminate or reduce stationary sources of pollution in compliance with the ever tightening tolerances of environmental legislation.

The glass industry consumes about 4 to 5% of the world's use of PGM's, most of this as platinum and platinum alloys. The bulk of this consumption is in bushings and spinnerets used to make glass fibers. PGM's are also required for the high temperature atmospheres of glass-melting tanks and as stirrers and crucibles to make high-quality optical and specialty glasses.

Other applications include structures for conveying molten glass, devices to form fiber optics, laser-glass melters, and hardware in the extrusion of glasses with high melting points. PGM's are essential to the production of products from roofing materials, insulation, fiberglass curtains, and printed circuit boards to automobile and boat bodies.

As previously noted, these four industries, with the partial list of applications I have just outlined, account for somewhere between 18% and 22% of the total usage of PGM's worldwide, or about 700 thousand of the approximately 3.5 million troy ounces used last year. So you can see the "all other" category makes a surprisingly significant contribution to international platinum demand, giving these applications a much more prominent stature than often initially assumed.

The role that PGM's play in industrial production is obviously an important one; and increased use of PGM's in this decade makes it clear that their role in industrial production is also growing in importance. Simply listing the ways in which these four industries use platinum group metals, however, is far from the whole story. It is their significance to the consumer, in terms of quality of life and modern lifestyles, that is a key issue in today's changing world.

It has been estimated that 1 out of every 5 products we use today is either made out of platinum, or that platinum was used at some point in its production. If you include platinum's sister metals, the number of products might even be 1 out of 4.

Right in this room, that percentage is probably even higher. Everything from the public address system to the air conditioning system, from the windows to my eyeglasses, from the furniture to the flooring material, from the wall coverings to the clothing covering each of us--all were made employing PGM's at some stage of their production process.

But despite this universality, platinum group metals have a low profile in today's world. Compared to other raw materials--such as wood, copper, tin, and aluminum--PGM's are not as obvious when it comes to the role they play in our everyday lives.

Except for their use in jewelry, they are virtually invisible to the consumer. We in IPMI, because we deal with precious metals every day, may not be as conscious of this low profile as perhaps we should be.

Being in Las Vegas, and being a betting man in any case, I will make this bet with you:

If you went down to the Strip right now, or to any main street in America, and you stopped the first 100 people walking by, other than the other members of the IPMI who chose not to be here this afternoon, and you asked them what role the platinum group metals play in their daily lives, 99 probably would not have the slightest idea.

The 100th might make a lucky guess--but he is probably the same guy who always beats the slot machines.

And, frankly, all 100 would probably say it doesn't matter--because they couldn't afford it anyway.

I would like to suggest that we take a step back and a brief look at the role which PGM's play in our everyday lives. Let us take some specific, clear-cut examples in just four areas: our cars, our homes, our lifestyle, and the world around us.

Our automobile has become one of the most basic and essential elements in our lives. It takes us to work, shopping, to our church or synagogue, to the movies, on our vacations. It is both the core and the symbol of something we routinely take for granted: our mobility.

What would happen if we removed from that car everything made of PGM's, or that was made from a process employing PGM's at some stage?

Well, let's see.

Automobiles have five basic systems: fuel, lubricating, cooling, electrical, and emission control.

Obviously, the catalytic converter whose platinum reduces the carbon monoxide and hydrocarbons resulting from the combustion process would be one of the first components to go.

Next is the engine. It is made of steel, which is primarily made of iron ore. That ore was mined with explosives, which were more than likely made from nitric acid. Nitric acid is made from a process involving the combustion of ammonia and water over platinum gauze--the key component that makes the production of nitric acid cost effective. The steel itself was produced in a process controlled using thermocouples composed of platinum wire and accessories.

Even if the engine were still there, it would not run--at least not efficiently--on the gasoline that would be in the gas tank. Refining and reforming catalysts made with the help of PGM's produce the high-octane gasoline that powers today's automobile engines.

As a matter of fact, you can also toss out the entire lubricating system. Platinum-based catalysts work their unique magic in the refining process that produces your motor oil.

And while you are at it, pull out the entire cooling system. The coolant was produced with the help of PGM catalysts, as well. In addition many of the circuits that control your air conditioning and heating systems are composed of PGM alloy components that keep them working despite severe operating conditions.

Just about all of your car's electrical system has to go too. Some modern cars have up to 1,000 capacitors; these, and most of the micro-processor controls in your electrical system, use PGM alloys because they are more reliable than other metals. The electronics in your AM/FM stereo radio, your cassette player and also your electronic ignitions include PGM's either printed or plated or fastened on, so you will have to scrap them too.

Now you can start looking for all the rubber in your car. Not just the tires, but the gaskets around the windows, the engine mounts, and the hundreds of other places where rubber is used. Chemical intermediates containing PGM catalysts went into making that rubber, so they have to go too.

Say good-bye to the windshield and all the window glass. Plus the headlights, all the light bulbs in the dashboard--do not forget that one in the glove compartment--and anything else in the car made of glass. As we have already discussed, glass makers use a variety of equipment and temperature sensing devices made of PGM's because of their ability to function properly despite the high temperatures and corrosive environment of the glass manufacturing process.

Next to go is all the trim: the grill, door handles, wheel covers, decorative strips. Modern electroplating processes employ PGM coatings on titanium anodes to produce chrome trim with greater corrosion resistance and longer life--all at lower cost, and with no contamination to the process or the environment.

The fabric in your car? It has to go. If it is man-made, PGM's were used to make it. Even if it is not man-made, it was undoubtedly treated with chemicals that, somewhere along the line, were exposed to PGM catalysis.

You can say good-bye to the paint as well. Paint manufacturers use PGM catalysts as chemical intermediates. In fact, without the unique catalytic capabilities of PGM's, today's automobile paint would not be as easy to apply, look as good, or last as long.

Well, at least you still have the body of your car, right? Wrong! Everything in your car made of steel has to go: the chassis, frame, doors, hood, trunk lid, and fenders. As previously described, steel-making is dependent on PGM's.

Aha, you say. Your car has a plastic body. Sorry, but it goes too. It is a fiberglass-reinforced plastic body, made with the help of platinum bushings that do not warp or deform despite the high temperatures involved in the manufacturing process for fiberglass.

In fact, you can remove everything else in your car made of plastics, even if it is not fiberglass reinforced. Either the catalytic effect of PGM's, or their ability to function in hostile chemical or high-temperature environments, contributed to their manufacture.

If you have not guessed by now, your car--at least the car you drive today--could not exist without platinum group metals. There might be something else in your driveway--a Stanley Steamer--but unless you are an antique car aficionado, you probably would not enjoy driving it, and it probably would have still cost you $20,000.

Now, let's take a look at the second example of the role that PGM's play in your life: your home. What would not be there--at least in its present form--if PGM's did not exist?

If I came to your home to visit you, the first thing I would probably notice is that your lawn is not as green as usual--the nitrogen fertilizer you normally use to maintain a lush, green lawn would most likely be too expensive if it had to be made without the benefit of platinum gauze.

The next thing I would probably notice is the absence of aluminum siding. Of course, the same nitric acid that ended up as fertilizer is also used to manufacture the explosives that get aluminum ore out of the ground.

When you show me in, I cannot help but notice that your wall-to-wall carpeting is gone. Nitric acid was also used to manufacture its fibers.

A barren wall is where your home entertainment center used to be. The TV set is gone because its glass picture tube was manufactured using process hardware and temperature sensors made of PGM's. Its capacitors and printed circuits, like those we threw out of your car, were also made with PGM's. Your stereo and compact disc players are gone for the same reason.

Yes, thank you, I would like a drink. Oh, a paper cup will be fine; sorry you do not have those handsomely decorated martini glasses any more.

I will not ask where your microwave oven is; I know that without the technology made possible by the distinctive properties of PGM's, the microwave probably would not have even been invented. In fact, all of the modern appliances I would normally find in your kitchen--the food processor, coffee maker, toaster oven--they all exist based on technology reliant upon PGM's. Yes, I guess you still <u>can</u> cook without them, but do you still remember how, . . . or like me, never learned?

Now let us look at the third example of the role that PGM's play in our lives: lifestyle. The food, clothing, and leisure-time activities we enjoy so much.

Clearly, you would not go hungry. But fertilizers produced employing PGM catalysis make up about 40% of all the fertilizers used to increase crop yields. Without the platinum gauze that makes the production of nitric acid cost effective, crop yields would be lower, and the price of food higher.

But the cost of food is only one of the factors influenced by the use of PGM's. Many of the preservatives which keep food fresh, and prolong shelf life, are made with intermediates which rely on PGM's and their catalytic capabilities.

Convenience is another lifestyle component. The food processing industry relies heavily on PGM based temperature sensors and electronic systems in dehydration, blending, quick-freezing, packaging, even shipping food products to the marketplace and storing them prior to sale.

What about our clothing? Man-made fibers are made by a series of processes which depend, in part, on the special properties of PGM's. The bleaching, dying, and weaving of the fabric is made easier and faster because of PGM's.

Yes, of course clothing could be made without PGM's; we did it for centuries. But the cost of the clothing would be higher, the wearability would not be as good, and the variety of fabrics and colors and styles would be limited.

Platinum group metals, you see, give us a wider choice--and thus enhance our lifestyles.

Leisure-time activities are also influenced by PGM's. Platinum bushings play a major role in the methods used to produce our plastic sailboats and power boats, our surfboards, and fishing rods and reels. Camera lenses are made with the help of PGM's, so is the film. Video cameras, and videotape itself, are end products of manufacturing processes that employ PGM's at some stage.

Even the sports on which we spend so much leisure time--whether competing or watching--are dependent on PGM's.

From today's synthetic turf to the scoreboards, from the electronic machines that print the tickets to the weather resistant uniforms of the ticket takers, from the saddles on the race horses to the jockeys' flashy colors, from the bowling balls to the pinsetting machines . . . All owe at least part of their origin to the platinum group metals.

The fourth and final example of the role that PGM's play in our lives involves the world around us: the earth and sea beneath us, the air we breathe, and the sky above us.

Motion sensors, made with platinum, detect earthquakes and the motion of bridges. Chemicals made with performance catalysts of PGM's reduce the pollutants from fossil-fueled power plants and other stationary generators of electricity. Electronic systems made with PGM's help our ships to navigate the seven seas, and guide our jet planes to their destinations.

Our defense systems are heavily reliant upon PGM's. Communication satellites cannot operate without them. Without PGM based thick-film composites, capacitors, and semiconductors, our astronauts and cosmonauts could not function in space. Space stations orbiting the earth, and expeditions to other planets, will need the magic of the six platinum group metals to reach their goals.

Finally, one of our most vital concerns--our health, and that of our loved ones--is greatly dependent upon PGM's. Electronic circuitry made with the help of PGM's monitor the heartbeats of our newborn children. The probes on diagnostic devices that help doctors find out what ails us would be far less reliable without PGM's. Pacemakers and hearing aid implants would not exist. Neither would many of the drugs which stop cancer cells from multiplying.

My talk this afternoon has been about the industrial applications of PGM's--but my message is about the role they play in enhancing the quality of our lives.

I have given you many examples of how the special properties of the platinum group metals--their resistance to corrosion, their ability to function at high temperatures and under high-stress conditions, and their catalytic capabilities--allow them to serve us where other metals cannot--or not as well.

That raises the issue of substitution. It is inevitable that industry will continue to search for alternatives to PGM's. But substitutes require a trade-off, a compromise which may result in reduced performance or lower quality, less reliability or higher cost. And in most cases, that compromise has been--and will continue to be--unacceptable--to industry, to government, and to consumers.

As the world draws closer together--drawn by a telecommunications system the pony express rider could not have dreamed of--drawn by a rate of technology development which continues to accelerate--platinum group metals will play an even greater role in enhancing the quality of life.

PGM's offer us the clearly better alternative between today's automobile and the Stanley Steamer; a modern home and a log cabin; our cellular phones and the pony express; between video games and checkers. Furthermore, by enabling us to enjoy the technology and capability of today's computers, PGM's give us the opportunity for the technological extension of our minds in contrast to the idle dreams of the science fiction writer. As our horizons expand, PGM's extend the limits of our world from the boundaries of the earth to the expanse of the universe.

No matter how good your imagination, no matter how creative your mind, it is impossible to imagine a world without platinum group metals. And even if we <u>could</u> imagine it, no one would be willing to go back to it.

That is what platinum group metals have given us: A legacy to a better quality of life.

Thank you.

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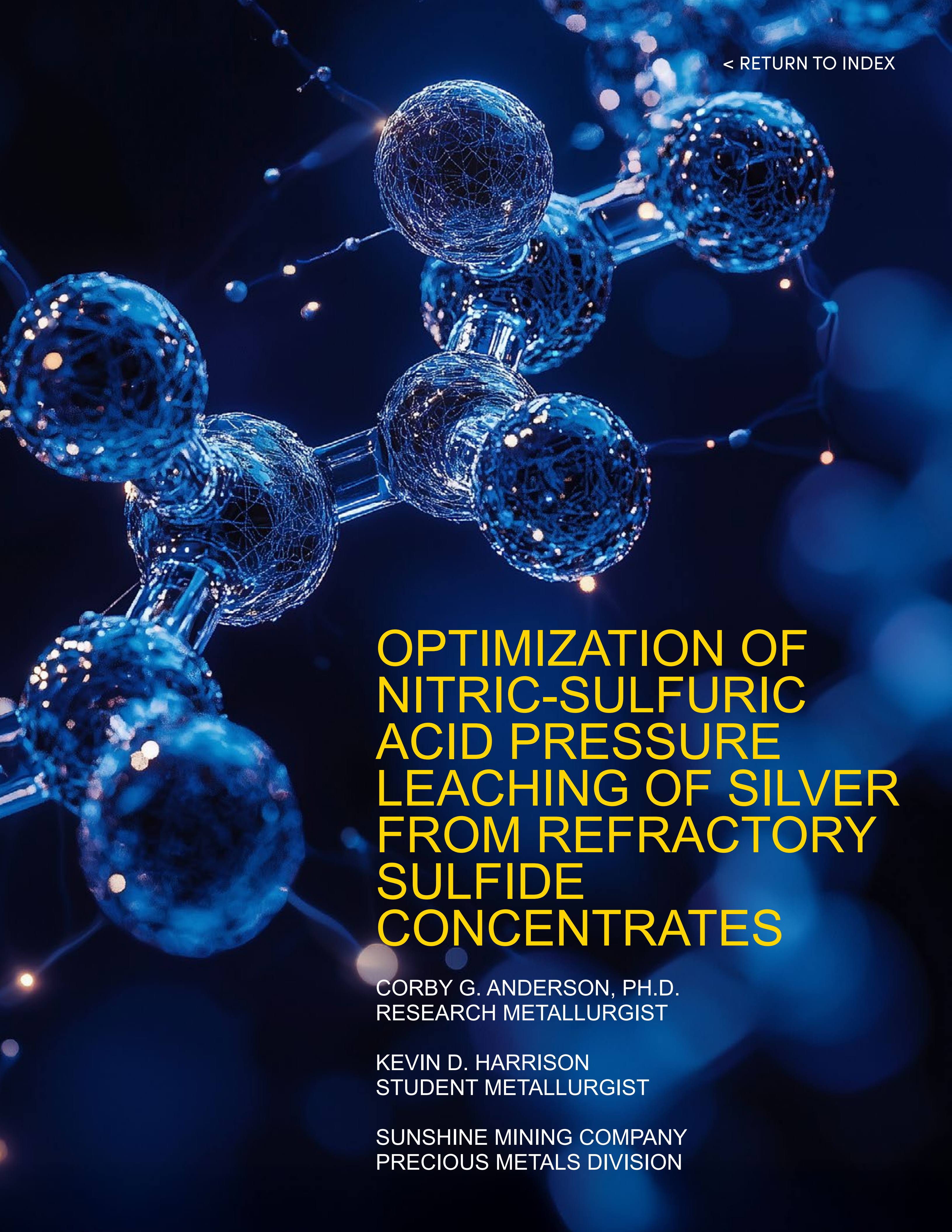

# OPTIMIZATION OF NITRIC-SULFURIC ACID PRESSURE LEACHING OF SILVER FROM REFRACTORY SULFIDE CONCENTRATES

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# Optimization of Nitric-Sulfuric Acid Pressure Leaching of Silver From Refractory Sulfide Concentrates

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## Abstract

Since 1985, the Sunshine Mining Company has used a patented nitric-sulfuric acid pressure leaching system for production of silver and copper from a refractory sulfide ore body. To better understand this novel process, a program of lab scale testing was undertaken to quantify the effect of system variables on silver recovery. Some of the factors studied include temperature, agitation, acidity, vessel pressure, particle size, percent solids and oxidizer amount. In turn, this knowledge will be applied in actual plant practice to upgrade reaction time, increase concentrate throughput, raise silver recovery and lower reagent costs. A discussion of plant process changes resulting from lab scale testing is included.

## Keywords

Refractory Ores; Silver; Copper; Pressure Leaching; Precious Metals; Nitric-Sulfuric Acid Leaching; Hydrometallurgy; Silver Leaching; Sulfide Leaching; Nitric Acid;

## Introduction

As one of the premier silver producing mines in the world, Sunshine Mining Company has relied on nitric-sulfuric acid pressure leach technology since 1985 for production of copper and fine silver from refractory ore. Developed and patented (1) by Sunshine in the 1970's, this process is unique among other nitric-sulfuric systems in its employment of low temperatures, pressures, and nitric acid concentrations. In that it is a full scale operating process, it also stands apart.

On the other hand, this uniqueness has carried with it some well documented (2) startup challenges. As can be expected, when scaling up entirely new technology to a plant scale, mere operation as expected can be a major achievement. As most of these initial difficulties have been rectified, the next step was to optimize the process. To do this a laboratory campaign was undertaken to successfully simulate, reproduce and define effects of process variables on silver recovery. From this work it was hoped that the system could be understood well enough to upgrade its performance.

## Overview

The concept of using nitric acid as a leaching agent in hydrometallurgy is not new. As its oxidizing properties are well known, it has been a natural choice for experimental work in oxidation of sulfides. Bjorling (3) used nitric acid in conjunction with sulfuric acid for dissolution of pyrite, chalcopyrite, sphalerite and several other materials. Brennecke (4) used this technology to study the leaching of copper sulfide concentrates at 105° C. The application of nitric-sulfuric leaching to copper concentrates has also been studied by Prater (5). Nickel concentrates have been treated in this manner in a study by Ouellet (6). Kunda (7) developed a process for production of silver for Terra Mines sulfide ore utilizing this technology For pretreatment of refractory ores, the Nitrox (8) and the Arseno (9) processes depend on nitric acid to oxidize gold bearing sulfides.

As related before, while all of these processes are similar in their general leach chemistry, the Sunshine process is distinct in that it is actually used in production. This is true most likely for two reasons. First, due to the rapid, exothermic reaction rates generally experienced with nitric-sulfuric leaching, it can be quite difficult to control on a large scale. Thus, many companies have avoided scaling up to production levels because of the uncertain nature of the process. From operating experience, the Sunshine refinery has overcome most of these control difficulties. Next, for production, the relatively high cost of nitric acid precludes its use in abundance unless regeneration is viable. The Sunshine process conveniently avoids this regeneration problem by requiring a minute amount of nitric acid (2g/L) for efficient silver leaching.

As mentioned before, these related processes share in common the use of nitric acid as an oxidizer. The key economic step is the regeneration of nitric acid from the reaction product gases. In the oxidation of a metal sulfide the following reactions can occur:

$$3MeS + 6H^{+} + 2HNO_3 = 3Me^{++} + 3S^{o} + 2NO + 4H_2O \qquad (1)$$

As can be seen, the nitric acid reacts with the sulfide to solubilize the metal value and produce nitric oxide gas. As this gas has a limited solubility in aqueous solutions, it tends to transfer out of solution. In the Sunshine pressure leach system, a closed vessel with an oxygen overpressure is used. The nitric oxide gas emanating from the leach slurry accumulates in the head space of the reactor where it reacts with the supplied oxygen to form nitrogen dioxide gas. This can be viewed as:

$$2NO + O_2 = 2NO_2 \tag{2}$$

In turn, the nitrogen dioxide is much more soluble than the nitric oxide and it is readily absorbed back into the slurry. This allows regeneration of the nitric acid in-situ.

$$3NO_2 + H_2O = 2HNO_3 + NO \tag{3}$$

As is well documented, all of these reactions are exothermic. Throughout the leach cycle, this process continues, resulting in oxidation of the sulfide material. Of course, this is a simplistic model of the leach mechanism. For illustrative purposes, it will suffice.

## Sunshine Plant Pressure Leach System

In order to more fully appreciate the nature of the research work presented here, it is important to understand how the plant system actually operates on a day to day production basis. The flowsheet in Figure 1 shows the general configuration of the facilities at Sunshine. Of note is the inclusion of a fully operational mint. Though not detailed in this paper, the mint gives Sunshine the unusual capability of complete product integration from mine to market.

From the mining operation, 1000 tons per day of approximately 20 troy ounce per ton silver are processed through a standard milling and flotation procedure. The native orebody, and resulting concentrate, is predominantly argentiferous tetrahedrite. However, significant amounts of sphalerite, pyrite, galena and tennanite are present, all containing some silver value. For antimony removal, the mill tetrahedrite concentrate is processed through another patented Sunshine leach process(10). In operation for nearly five decades, it produces metallic antimony metal through a combined ambient pressure alkaline sulfide leach and diaphragm cell electrowinning process. Simplistically, the solubilization of antimony can be described as:

$$3Cu_2S{:}Sb_2S_3\,_{(s)} + 3Na_2S = 3Cu_2S\,_{(s)} + 2Na_3SbS_3 \tag{4}$$

Here the tetrahedrite mineral is preferentially leached of its antimony content leaving behind a chalcocite residue. Before the advent of the current pressure leach system for silver, this system proved to be an effective method of antimony removal from the high grade silver/copper values shipped to smelters. This alleviated any penalties administered by refiners for high antimony content. Now, it serves the dual function of producing more readily leached solids for the pressure leach system and of lowering any penalties encountered for antimony content in the leach by-products. As such, it functions as the largest domestic producer of antimony metal. For convenience, the final solids from this process, as well as the typical assays of the mine ore and the mill concentrate, are presented in Table 1.

In Figure 2, the actual layout of the pressure leach

Figure 1

Sunshine Mining Company
Process Flowsheet

### Mine

1000 Ton/Day Hoisted

20 oz/Ton Ag Ore Grade

### Mill

Crushing, Grinding and Flotation

25 Ton/Day Concentrate Production

### Antimony Plant

25 Ton/Day $Na_2S$ ($S^o$):NaOH Ambient Pressure Leach

Cathode Sb Electrowinning

### Silver Refinery

20 Ton/Day Nitric-Sulfuric Acid Pressure Leach

Ag Precipitation, Conversion and Refining

Cu SX/EW

Solid Leach By-product Handling

Custom Au & Ag Dore Refining

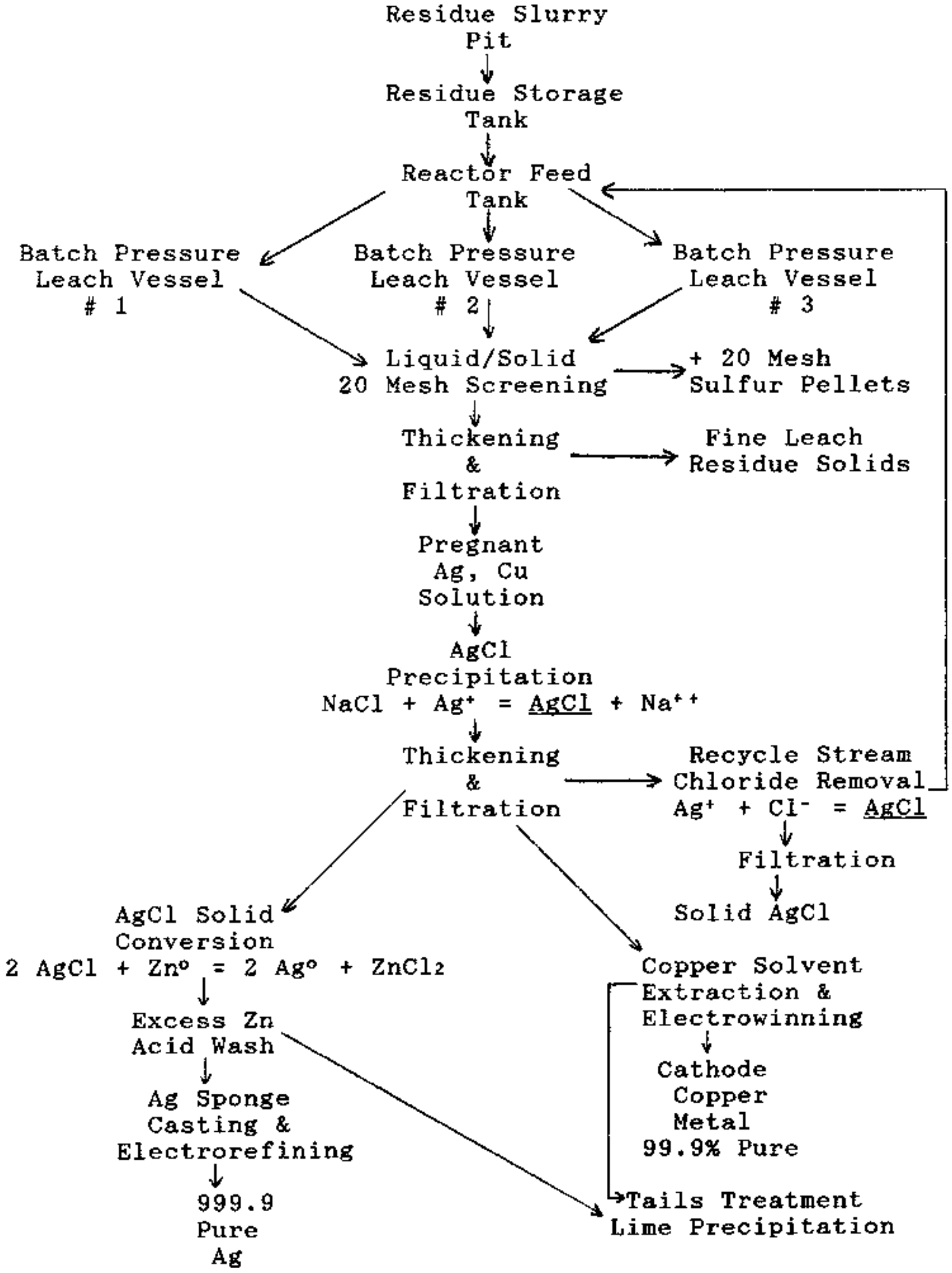
### Sunshine Mint

Au & Ag Bullion and Coin Production

Au & Ag Bullion, Jewelry and Coin Sales

Figure 2

Sunshine Silver Refinery
<u>Flowsheet</u>



area in the silver refinery is shown. Also illustrated are the downstream processes for handling and refining the leach products. As a first step, these downstream processes will be briefly covered. In essence, after the silver, copper and lesser metals have been solubilized in the leach, the hot slurry is transferred from the reactor to screening, thickening and filtration of solids. The pregnant solution is stripped efficiently and selectively of its silver using NaCl solution to precipitate AgCl.

Table 1
Typical Composition of Sunshine Mine
Mill and Antimony Plant Products

| Product Type | Ag, oz/Ton | Cu, % | Sb, % |
|---|---|---|---|
| Run of Mine Ore | 20.0 | 0.4 | 0.4 |
| Mill Concentrate | 1200.0 | 22.0 | 19.0 |
| Refinery Feed | 1300.0 | 24.0 | 0.5 |

$$Ag^{+} + NaCl = AgCl_{(s)} + Na^{+} \qquad (5)$$

The solid AgCl precipitate is filtered from the pregnant solution, washed thoroughly and converted to elemental silver with zinc powder.

$$2AgCl + Zn^{o} = 2Ag^{o} + ZnCl_2 \qquad (6)$$

Using electrorefining, the silver is further refined to a purity of 99.99%.

What remains of the pregnant solution contains considerable quantities of copper and free sulfuric acid. To save acid, one third of this stream is recycled back to the leach area for make up solution. As it contains a significant amount of chloride ion due to silver stripping,it is cleaned up by precipitation methods. This entails using some of the silver containing pregnant stream to react with the chloride ion in the recycle stream. Filtering of the contained solid AgCl leaves a liquid with a slight excess of silver in solution. This process ensures that no chloride ion is transferred to the pressure leach vessels, comprised of 316L stainless steel. This is important for two reasons.

First, as is well documented, the potential for corrosion using this type of pressure vessel in chloride environments immense. Secondly, it ensures that silver solubilization in the leach will not be inhibited by AgCl precipitation.

The other two thirds of the solution contains significant quantities of acid and copper. After slaked lime neutralization of much of this acidity, the copper is extracted using conventional solvent extraction and electrowinning technology. A high grade copper metal cathode is produced.

In a nutshell, that is how the solubilized leach products are processed. As the thrust of this work involves the leach area, a close look at the daily operation of the nitric-sulfuric acid leach system is necessary. Table 2 illustrates the operating conditions required for the pressure leach. In practice, copper residue from the antimony plant is sent to the residue slurry pit. Here it is mixed with water and agitated until it is pumped into the plant. Upon entry into the plant, it is put in the residue slurry storage tank. This slurry density is monitored by a nuclear density gauge.

Table 2
Sunshine Pressure Leach
Operating Conditions

Initial Weight Percent Solids = 8% to 17%

Initial Free Sulfuric Acidity = 170 to 200 g/L

Initial Preleach Solution Copper Content = 2 to 10 g/L

Preleach Temperature = 85° to 100° Celsius

Preleach Time = 30 to 90 Minutes

Initial Pressure Leach Temperature = 80° to 90° Celsius

Vessel Pressure = 90 PSIG

Reaction Time = 1 to 2 Hours

Maximum Reaction Temperature = 145° to 155° Celsius

Final Free Sulfuric Acid Content = 105 to 115 g/L

Next, the slurry is pumped into the reactor feed tank. It is here that several important plant operations come together. First, and foremost, is the makeup of the feed for the batch pressure leach. Of primary importance are the percent solids in each batch and the initial sulfuric acid concentration. The slurry content is made up in the range of 8 to 17 percent by weight solids, the actual number depending on the overall reactivity of the feed (ie. sulfide content). Again, a nuclear density meter is used for approximating slurry density.

For a typical leach batch, the sulfuric acid content is in the range of 170 to 200 g/L. As will be further illustrated, this number is conditional upon obtaining an adequate final free acidity for avoidance of silver robbing jarosite formation. The initial batch acidity is controlled by metered additions of sulfuric acid, followed by free acid sample titration. As more than the amount of slurry needed in one pressure reactor is made up at a time, this method is satisfactory.

Along with batch makeup, the reactor feed tank serves as a point of recycle for the aforementioned downstream silver free solutions. As a significant process parameter in terms of cost recovery is sulfuric acid, it is essential that it be recycled instead of consumed by lime neutralization. Thus, each batch is partially made up with the 100 g/l sulfuric acid recycle solution. Also, the actual quantities of recycle are determined by the amount of copper needed in the preleach step. This is because of the nature of the feed from the antimony process. As it contains small amounts of entrained sodium sulfide solution, contact with acid at elevated temperatures causes immediate evolution of $H_2S$. This poses a corrosion as well as a health hazard. Fortunately, the copper content of the recycle solution allows this problem to be abated. By adjusting the batch makeup to have an excess of copper in solution, any $H_2S$ gas generated in the preleach will be contained by formation of copper sulfide.

Another important aspect of the reactor feed tank is the elimination of siderite from the pressure reactor feed. In plant practice it has been found that the presence of these iron carbonates in the pressure leach can inhibit the leach reaction. As carbon dioxide is one of the reaction products when siderite decomposes, it could conceivably fill some of the head space in the pressure vessel. As nitric-sulfuric acid leaching is dependent upon regeneration of oxidants in the head space, any inert gases present could inhibit reaction rates. Thus, the heat, acidity and residence time available in the reactor feed tank serve to relieve the feed of any carbon dioxide gas before entry into the pressure vessel.

Lastly, the reactor feed tank serves as a heat sink for flashdown of the pressure leach system. At the end of a pressure leach the heat from the reaction ,mostly in the form of steam, is vented directly into the preleach reactor. This is an efficient method of direct heat exchange between reactor vessels.

Once the proper batch requirements are accomplished in the reactor feed tank, the slurry is pumped sequentially to any one of three pressure leach vessels. Each vessel is charged half full with approximately 4000 gallons of slurry (ie. equal to 2 tons of dry feed). As each pressure vessel is fitted with dual agitation props, (one at the bottom of the agitation shaft and one half way up the shaft) this places the tank level just above the level of the upper prop. It is perceived that this prop arrangement allows splashing of the slurry into the headspace thereby enhancing mass transfer of the regenerated nitrogen dioxide species back into the slurry.

For removal of air in the headspace, a stream of oxygen is allowed to flow through the reactor for one to two minutes. Next, the vessel is sealed and about 10 gallons of nitric acid is pumped in. Oxygen is then allowed to flow into the vessel to provide the overpressure required for the regeneration reaction. Although the usual operating condition calls for 90 psig total pressure in the reactor, during the initial reaction start about 70 psig is maintained. This is because of the ever present danger of pressure spikes. Caused by the buildup of gases in the headspace (ie. NO) coupled with the extreme heat generation (ie. $PV=nRT$) during periods of rapid sulfide oxidation, these spikes pose the problem of overpressuring the vessel, causing the safety rupture disks to burst. However, through operating experience, careful control of nitric addition and the aforementioned low initial oxygen overpressure, these spikes are all but avoided.

Initially, the temperature is about 80 - 90° C. As time goes on oxygen begins to be consumed and the temperature rises to about 115 - 120° C. In this range, elemental sulfur deposited as a product layer on the reacting particles begins to liquify.

Because of reduced diffusion resistance to reactants, there is a sudden rise in temperature and oxygen consumption. (This phenomenon will be discussed in greater detail later in this paper.) The reaction continues to a temperature of about 145-155° C . Careful batch makeup by the operators insures that the reaction ends in this temperature range. Accordingly, 87%+ of the silver and 97%+ of the copper are leached.

When the process is completed, the pressure vessels, as noted before, are flashed of their heat content directly into the reactor feed tank. To avoid plugging problems caused by sulfur chunks, the vessels are agitated while flashing steam to a temperature below 108° C. When this temperature is reached, the vessel is discharged and liquid/solid separation commences.

From the leach, three major products are produced. These are the pregnant solution, sulfur pellets, and the fine leach residue. The sulfur pellets are first removed from the slurry by screening on a 20 mesh industrial screen. X-ray diffraction has revealed them to be composed of elemental sulfur and unreacted pyrite. They also contain small amounts of copper and silver sulfide.

The fine leach residue consists of the minus 20 mesh unsolubilized solids. X-ray diffraction revealed them to be composed of a mixture of lead sulfate, pyrite and sulfur, as well as some unreacted silver and copper sulfides. This material is allowed to settle in a thickener and is subsequently captured in a filter press.

The pregnant solution consists of the solubilized silver, copper, iron and zinc in a sulfuric acid matrix. As previously outlined, this solution is treated for separation of valuable constituents downstream.

Along these lines, Table 3 illustrates the relative percent constituencies of the leach products in terms of where they normally report. Table 4 reveals typical assays of the leach products.

Overall, the system works quite well for solubilization of silver. However, from plant experience, other laboratory studies and x-ray diffraction work, it is known that several processes can work to rob the silver from solution, if conditions are correct.

Foremost of these problems is silver containing jarosite formation. Though factors of temperature and iron ion concentration do play a role in its formation, in the Sunshine system, it is more a function of free acidity. In practice, anytime the free acidity in the leach falls below approximately 100 g/L, silver will climatically precipitate as an argentojarosite. This phase, along with plumbojarosite, has been conclusively identified by x-ray diffraction of solids emanating from leach solutions with low free acids. Thus, for high recovery, this situation is to be avoided at all costs.

Another bad actor in silver losses is the presence of unreacted pyrite. As it has a high affinity for collection of surface coatings of sulfur, it tends to be inhibited from being oxidized in the leach. These sulfur coatings remain intact after the leach is completed, thus causing the majority of the pyrite to report to the +20 mesh sulfur pellet product. The unfortunate side effect of pyrite is that it can react electrochemically to cement silver from solution. As revealed in a study by Hiskey(11), the reaction of note is:

$$Ag^{+} + FeS_2 = Ag^{o} + Fe^{2+} + 2S^{o} \qquad (7)$$

In plant practice, as well as in lab testwork, it is imperative to separate the pyrite containing sulfur pellets from the silver containing pregnant solution immediately after the reaction is completed. If the pyrite is allowed to mix for any period of time after the oxidizing conditions provided by the nitric-sulfuric acid leach are removed, silver losses will begin to occur. Luckily, in the plant, the first step after leach discharge is 20 mesh screening to remove most of the sulfur pellets. Additionally, if a leach has to be held until there is room downstream for discharge, it is kept under oxygen pressure without removal of the nitrogen species. For the lab work, the pyrite containing sulfur pellets were screened away from the pregnant solution immediately upon discharge from the reactor vessel. This minimized silver cementation by pyrite and ensured reproducible results.

Table 3
Final Relative Elemental Distribution
of Plant Leach Products

| | Ag | Cu | Fe | Pb | Sb | Zn |
|---|---|---|---|---|---|---|
| Pregnant Solution | 87.6% | 97.2% | 80.0% | 0.0% | 0.0% | 99.0% |
| Fine Leach Residue | 9.4% | 0.9% | 6.0% | 99.5% | 99.9% | 0.5% |
| Sulfur Pellets | 3.0% | 1.9% | 14.0% | 0.5% | 0.1% | 0.5% |

Table 4
Final Elemental Analysis
of Plant Leach Products

| | Ag | Cu | Fe | Pb | Sb | Zn |
|---|---|---|---|---|---|---|
| Pregnant Solution | 5 g/L | 40 g/l | 25 g/L | 1 ppm | 1 ppm | 2 g/l |
| Fine Leach Residue | 800 oz/T | 1% | 3% | 35% | 4% | 0.1% |
| Sulfur Pellets | 100 oz/T | 0.5% | 10% | 0.1% | 0.1% | 0.1% |

## Laboratory Methodology

On a plant scale, the goals of optimization were to maximize silver recovery, minimize reaction time, increase throughput rate by increase of percent solids in each batch, and minimize reagent costs. For effective delineation of the parameters influencing achievement these goals, a reliable, reproducible method of simulating the process on the lab scale had to be developed. Due to the nature of nitric-sulfuric pressure leaching itself, this had not before been. a readily achievable task. In fact, the operating plant facility parameters were largely worked out during the startup phase rather than in laboratory or pilot plant settings. Thus, as an overall research philosophy, it was decided from the start to define the system from a process standpoint. Any thermodynamic, pure kinetic or mathematical modelling studies could then be approached as an extension of this work. Thus, for this testwork, a successful, accurate method of leaching, liquid/solid separation, sampling and assaying was devised.

For the study, bulk samples of silver refinery feed were gathered at the antimony plant. The material was thoroughly mixed and split into smaller portions. Because of acute sensitivity to pyrophoricity upon exposure to air (ie. ignition can occur in large masses!) and to ensure uniform moisture, the samples were sealed in plastic and stored in a refrigerator. Table 5 indicates the elemental content of these representative head samples. For actual testwork, a two gallon Parr autoclave vessel ,constructed from 316L stainless steel, was used. A volume of 5 liters of slurry was used in each test. This was agitated with a set of dual axial propellers. One was placed on the bottom of the agitator shaft and the other further up to coincide with the interface of the slurry and the reactor headspace. This arrangement is similiar to the design utilized in the plant reactor vessels. The guiding philosophy behind this dual prop arrangement is twofold. First, it provides sufficient agitation and pumping action to move any solids from the reactor bottom and into solution. Secondly, it can provide, if the tank level is correct, enhanced mass transfer to and from the slurry by splashing at the interface.

Table 5
Elemental Content of
Bulk Research Head Samples

| Ag | Cu | Fe | Zn | Pb | Sb |
|---|---|---|---|---|---|
| 1230 oz/T | 17.8% | 19.8% | 1.96% | 2.57% | 0.75% |

In practice, the lab autoclave was charged with the required mix of preleach reagents and copper residue. A head sample of copper residue was split out for moisture and elemental analysis. The vessel was sealed, except for an air vent to maintain ambient pressure, and the agitators were turned on at the proper speed. As external heating was required in the lab vessel, the heater was turned on and the proper temperature was set. After attaining 90° C, a timer was started and the slurry was preleached for the proper time.

After preleaching, the vessel was purged with oxygen to a pressure of 90 psig. This successfully ridded the headspace of any inert gases present from the preleach step. Next, the proper agitation was set for mixing. The external heater set point was raised to 120° C. The required amount of nitric acid was added using a valved tube attached in-line with the oxygen flow tubing. In practice, this nitric acid containing tube was first attached to a port on top of the pressure vessel. Then, the oxygen gas line was attached to it, forcing the nitric acid into the reaction chamber. This method assured that all the nitric acid reported to the vessel safely and reproducibly as the oxygen flowed in to pressurize the vessel to the required amount.

To monitor reaction progress, a plotter was attached to the autoclave thermocouple. This gave data regarding relative temperature rise with time. Indirectly, this gave a measure of relative reaction rate data. Also, oxygen flow rates were monitored and recorded manually with conventional lab flowmeters.

Normally, there was a rapid temperature rise up to about 110° C followed by a marked slow down. As will be discussed more thoroughly, this is conceivably due to buildup of a sulfur reaction product layer on the sulfide mineral. At about 120 - 125° C, the reaction took off again resulting in a rapid temperature rise. Since the initial test work found the system to be extremely sensitive to temperature, reactor cooling was used to control the maximum temperature value.

When the required reaction time was reached, the vessel contents were cooled to 85° C. The vessel pressure was vented with tubing to a vented lab hood. Then, the autoclave was opened and discharged. Sulfur pellets were screened away and washed on a 20 mesh screen. The remaining slurry was vacuum filtered for fine residue removal. All solids were thoroughly washed and then dried overnight in a lab oven. Liquid volumes were measured at room temperature. Elemental analysis was performed on all solids and liquids by atomic absorption. From this data, mass balances for each test were performed.

## Selection of Baseline Conditions

In order to successfully delineate the relative importance of system variables, a comparative and reproducible baseline had to be established. An accurate reflection of the plant process could only be achieved through careful selection of experimental conditions. After some initial testwork was done, the parameters shown in Table 6 were chosen as baseline conditions. Resulting recoveries of silver and copper for several baseline tests are shown in Table 7. As can be seen, reproducibility and precision were excellent. Also, overall mass balances performed on each leach indicated that metallurgical accounting accuracy was excellent. This prevailed throughout the research. Thus, any subsequent testing was performed by only varying the isolated parameter being studied. The other conditions were left at the established baseline values.

## Agitation

The first series of tests undertaken was to minimize the effect of agitation on the silver recovery rate. Though this study was not of a fundamental kinetic nature, in order to clearly see other variable effects, it is important to minimize the mass transfer reaction barrier. As seen in Figure 3, this is effectively done at a stirring speed of 1000 RPM. Above this speed, there is no dependence of silver recovery on agitation.

Table 6
Experimental Baseline Conditions

Initial Free Sulfuric Acid = 200.0 g/L

Initial Ferric Ion Concentration = 2.3 g/L

Initial Cupric Ion Concentration = 18.5 g/L

Preleach Temperature Range = 90 - 105° C

Preleach Time = 60 minutes

Slurry Solids Content = 110 g/L

Solids Particle Size = 80% -25 micron

Agitation Rate = 1000 RPM

Nitric Acid Concentration = 2.0 g/L

Total Leaching Pressure = 90 psig

Time of Pressure Leach Reaction = 90 minutes

Maximum Reaction Temperature = 140° C

Table 7
Recoveries In Pressure Leach
at Baseline Conditions

| Test | Ag % | Cu % |
|---|---|---|
| 1 | 81.22 | 96.03 |
| 2 | 78.65 | 98.20 |
| 3 | 80.14 | 97.76 |
| 4 | 81.17 | 97.76 |
| 5 | 78.76 | 98.19 |
| 6 | 79.48 | 97.59 |
| Avg | 79.90 | 97.59 |
| % RSD | 1.30 | 0.82 |

Figure 3
Effect of Agitation on Ag

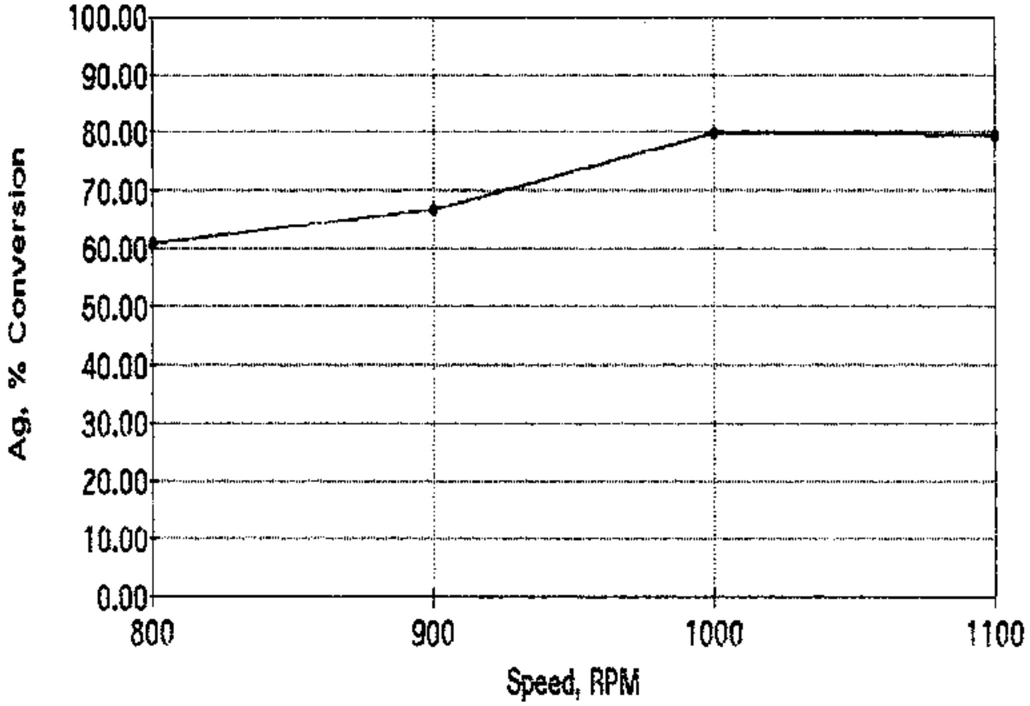
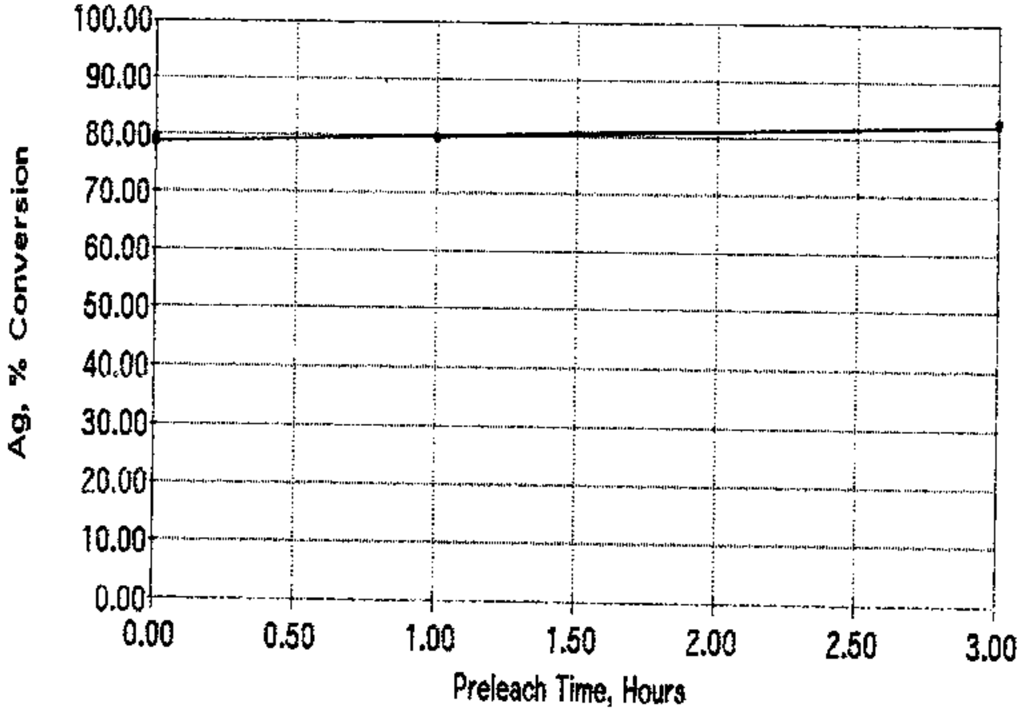


Figure 4
Effect of Preleach Time on Ag



Thus for all subsequent leach experiments, 1000 RPM was chosen as the standard stirring speed.

## Preleach Time

As in the actual plant situation, preleach time in the reactor feed tank can be an important factor. In relation to the, pressure leach recovery, it is important for removal of $CO_2$ from the iron carbonates present in the ore. As alluded to earlier, this gas can partially fill the headspace of the pressure vessel, cutting down the volume left for oxidizing species. As seen in Figure 4, with the copper residue samples used in this work there is not much effect on conversion no matter the amount of preleaching. However , the trend is toward slightly higher recoveries with more time. A complete mass balance of the major components leached in a 1 hour preleach is given in Table 8.

## Pressure Leach Reaction Time

In plant practice, the normal end of a leach is indicated when no more oxygen is consumed. As nitric-sulfuric sulfide oxidation rates are quite high, there is no problem seeing this end point. However, in the lab, it was observed that in most cases, there was always some consumption of oxygen after the full allotted reaction time had elapsed. To find out the degree of continued reaction, differing time periods were used with baseline conditions. As is shown in Figure 5, there is continued recovery of silver with extended time periods for the leach. However, at 45 minutes of reaction, the point where oxygen consumption and temperature rise were usually seen to be diminished, the majority of the silver had been reacted.

Moreover, this trend of increased silver conversion with time of reaction is important from an operation standpoint. It is quite obvious that the deleterious effects of pyrite cementation of silver are overcome by keeping the leach under pressure with nitric acid present. Thus, in the plant, any leaches which must be held for extended periods of time should be kept under these conditions.

## Initial Iron Ion Concentration

To understand the effect of iron in solution, ferric and ferrous sulfate were added to the preleach in varying quantities. The results of these tests are shown in Figures 6 and 7. As can be seen, increased ferric ion concentration results in increased silver conversion. Conversely, increased ferrous ion concentration inhibits silver conversion. This is due to the relative redox properties of each species. It is well known that ferric ion is a relatively strong oxidizer. Also, ferrous is well known as a reducing agent. Because of these properties, these ions either contribute to or debilitate the initial preleach Eh. This fact is illustrated clearly in Figure 8. Using slurries preleached at baseline conditions ,with the same amounts of ferric or ferrous ion as the data of Figures 6 and 7, initial Eh (Ag/AgCl reference) readings were taken. The results clearly indicate the strong dependence of subsequent pressure leach silver recovery on initial Eh. As silver and its compounds are

### Table 8
### Final Preleach
### Elemental Distribution

| | Ag % | Cu % | Fe % | Pb % | Sb % | Zn % |
|---|---|---|---|---|---|---|
| Preleach Solution | 0.0 | 0.0 | 19.6 | 0.0 | 30.3 | 70.0 |
| Preleach Solids | 100.0 | 100.0 | 80.4 | 100.0 | 69.7 | 30.0 |

Figure 5
Effect of Reaction Time on Ag

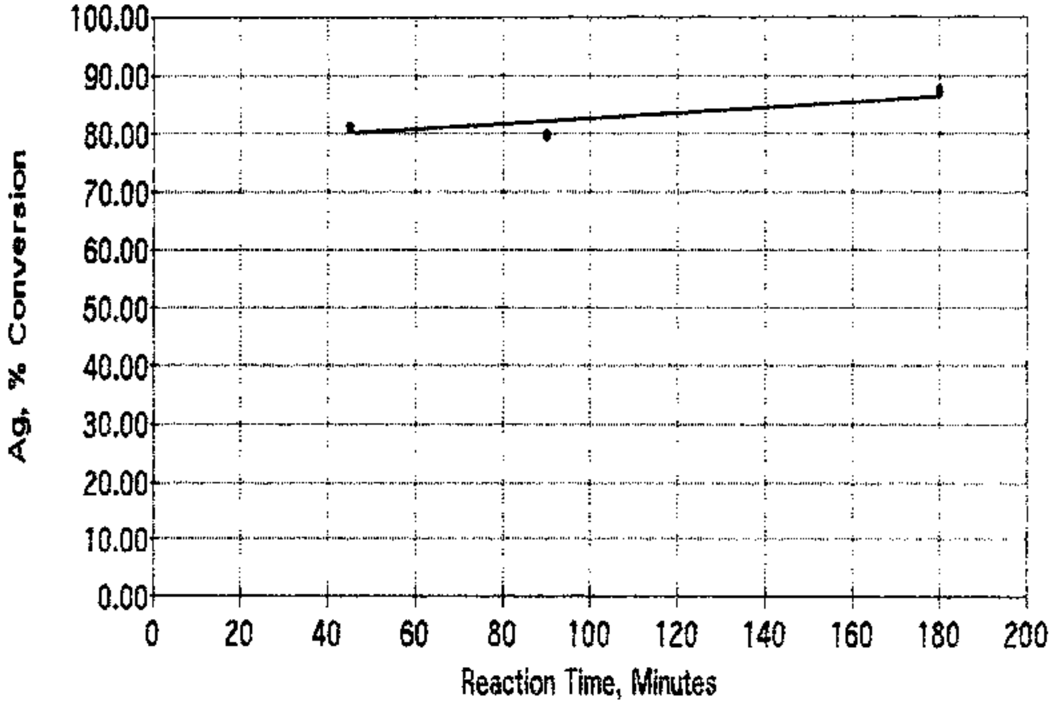


Figure 6

Effect of Initial Ferric Conc. on Ag

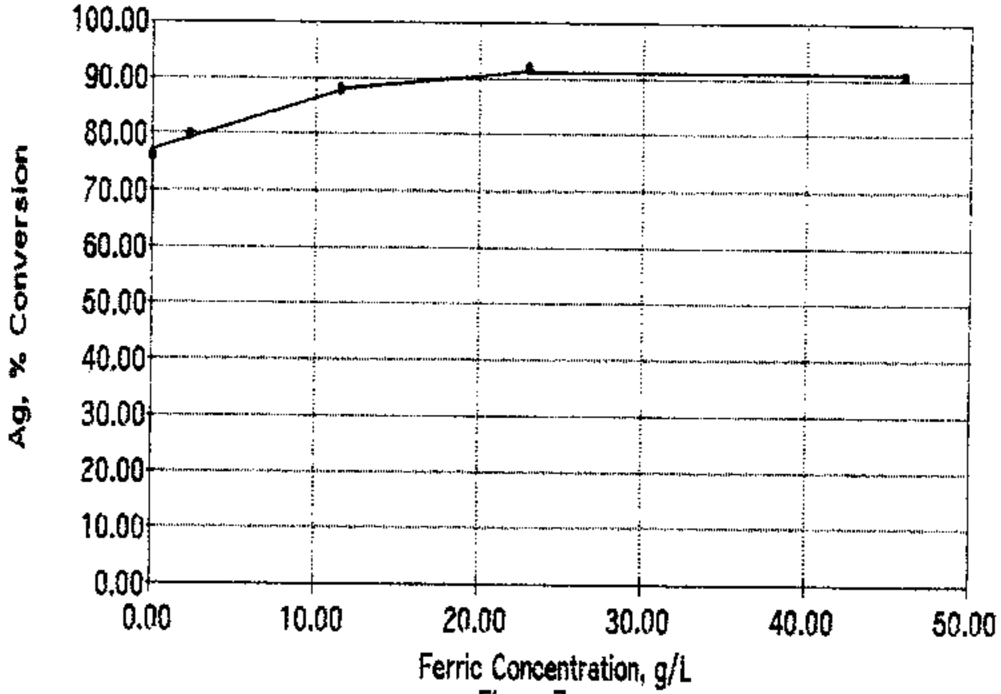
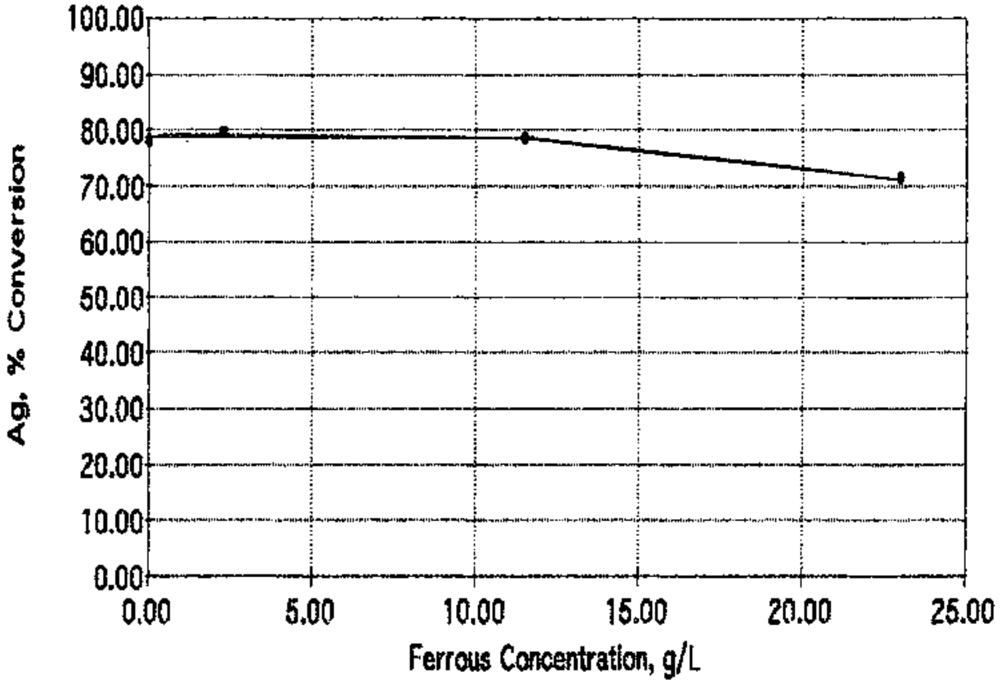


Figure 7

Effect of Initial Ferrous Conc. on Ag



quite noble, this is not surprising. Highly oxidizing conditions are needed to solubilize silver compounds in acidic conditions. In nitric-sulfuric pressure leaching, this phenomenon has been witnessed before. As shown by Fair (12), the Nitrox process requires high Eh values for reasonable silver recoveries.

Along these lines, the magnitude of the effect of ferrous suppression of silver recovery is a bit surprising. Ordinarily, it would reasonable to expect that the ferrous ion would be quickly oxidized to ferric, with little effect on recovery. Yet, ferrous ion in solution has been used by Brennecke (4) for nitrate ion removal. As the ferrous was oxidized to ferric, the nitrate was reduced evolving NO gas. In the Sunshine system, initial ferrous oxidation by nitric acid could be causing a rapid evolution of NO gas to the headspace. This would result in initial slurry depletion of the nitrogen species vital for silver dissolution. Moreover, this depletion would be happening at a crucial time when the silver mineral is most amenable to oxidation (ie. little or no product layer diffusional barrier, sulfur or otherwise). Thus, the effect would be magnified. Whatever the case, more research would be necessary to define this phenomena.

## Maximum Temperature of Reaction

In the preliminary work of establishing baseline conditions, it was found that the silver recovery was very sensitive to the temperature reached in the reaction. Very early on in the work, it was decided to limit the highest temperature achieved by cooling the vessel. While this aided in achieving reproducible results, it also led to an investigation of the effect of temperature. The results of these tests are shown in Figure 9.

As can be seen, there appears an optimum recovery value at about 125° C. Above, and below, this maximum reaction temperature, recoveries are depleted. This temperature is best due to a number of factors at work during the leach.

First of all, in the plant and in the lab, there is a rapid gain in reaction rate around this temperature. This is a result of kinetic phenomena. As x-ray diffraction work has indicated, one of the major minerals present in the refinery feed is chalcocite ($Cu_2S$). As illustrated by Bhappu (13), this mineral is oxidized sequentially to CuS and then $Cu^{++}$. Below 155 - 160 ° C, the metastable phase of elemental sulfur is the reaction product rather than sulfate. Below the melting point of sulfur, a non-conductive sulfur layer builds around the reacting sulfide. As this layer builds, it impedes the reaction to the degree that diffusion through the product layer is a major reaction barrier. However, above the melting point of sulfur, this product layer will liquefy, thereby enhancing the reaction rate. For most sulfide minerals, this phenomenon is common and has been recognized by other researchers (14,15,16). In his work, Braithwaite (17), found a definite shift in activation energy above the melting point of sulfur ( approximately 120° C) indicative of a change from diffusion to chemical control. In the case of this research, this is most certainly the situation being witnessed. At 125° C, any sulfur present as a product layer will have melted, allowing exposure of other unreacted sulfides.

Figure 8
Effect of initial Eh on Ag

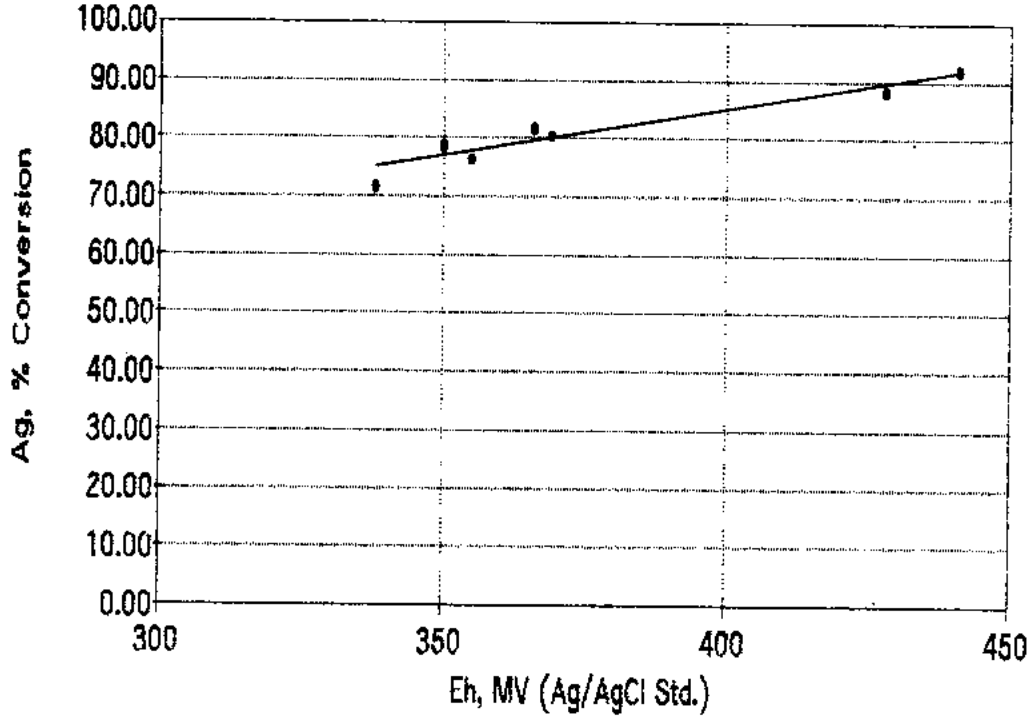
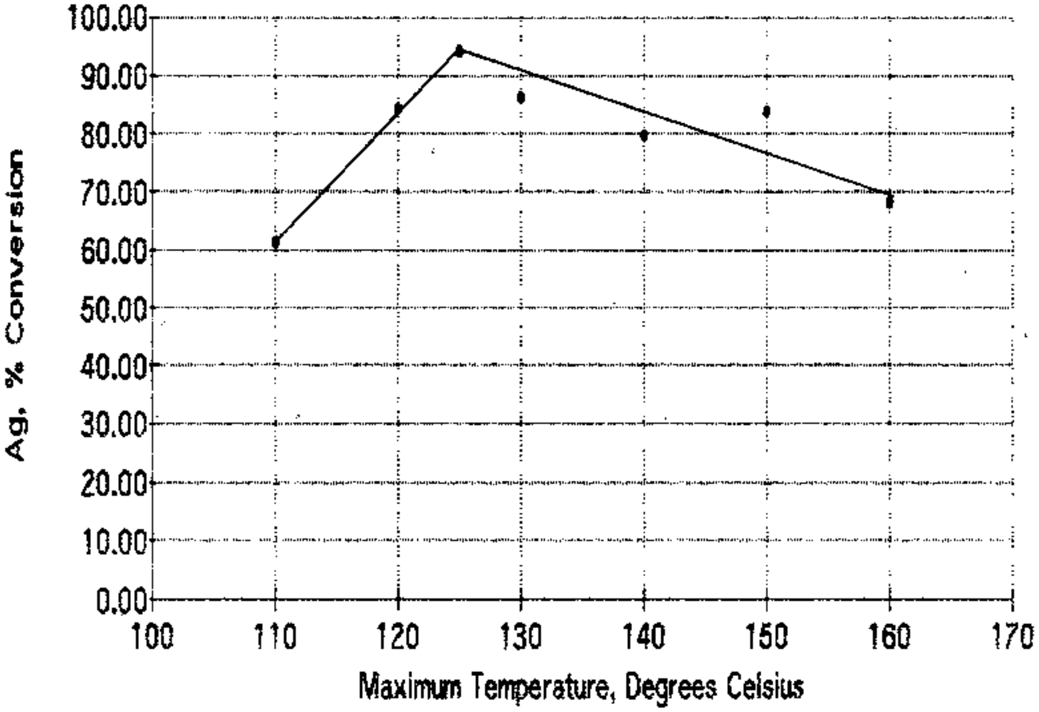


Figure 9
Effect of Max. Temperature on Ag



Thus, an enhanced reaction rate is noticed in terms of temperature rise, oxygen consumption and ultimately, in silver solubilization. Conversely, any tests run below this optimum temperature were prone to lower conversions from the continued and growing presence of a metastable sulfur product layer.

Above, 125° C, there was a decline in overall conversion most likely due to several factors. First, is the nature of the reacting system itself. All of the reactions in the vessel, particularly the nitric acid regeneration chemistry, are highly exothermic. Ultimate thermodynamic recovery potentials are lessened for exothermic reactions with an increase in temperature. In the chemistry of nitric acid manufacture, which is similar to the regeneration step in the leach process, the concept of cooling is practiced regularly for increased conversion. Surprisingly, this is also done for kinetic reasons. Classic work done by Bodenstein (18) in 1922 established that the rate constant for oxidation of nitric oxide to nitrogen dioxide actually increases for a decrease in temperature. The relationship for this situation he expressed as:

$$\log k = (641/T) - 0.725 \qquad (8)$$

Thus, in practice, cooling will directly increase the speed and amount of $NO_2$ generation.

Secondly, with increasing temperature in the constant pressure system at Sunshine, there is a rapid decrease in oxygen partial pressure in the head space of the vessel. This is due to the rapid increase of the vapor pressure of the contained liquid with temperature. Thus, in the regeneration reaction:

$$2NO + O_2 = 2NO_2 \qquad (9)$$

where it has been established that:

$$\text{Rate of } NO_2 \text{ produced} = k(P_{NO})^2 P_{O_2} \qquad (10)$$

oxygen concentration is depleted, resulting in lowered nitrogen dioxide production and ultimately in lowered silver leach rates.

Lastly, with increased temperature there is the increased possibility of argentojarosite and silver bearing plumbojarosite formation. X-ray diffraction work has identified these as reaction products under conditions of low free acidity (ie. less than 100 g/L) in the Sunshine pressure leach system. As found by Tourre (19) and Dutrizac (20), along with high ferric concentration, and low free acidity, increased temperature plays a key role in silver bearing jarosite formation. With increased temperature, in the range of 120-190° C, there is an increase in the potential for jarosite formation. Thus, in the research work, the effect of this jarosite formation is likely seen in the lowered silver recoveries at the higher maximum reaction temperatures.

## Effect of Oxygen Pressure

The results of variation in total pressure during the reaction is shown in Figure 10. These conditions were achieved by changing the pressures of the incoming oxygen gas. As the total

Figure 10
Effect of Pressure on Ag

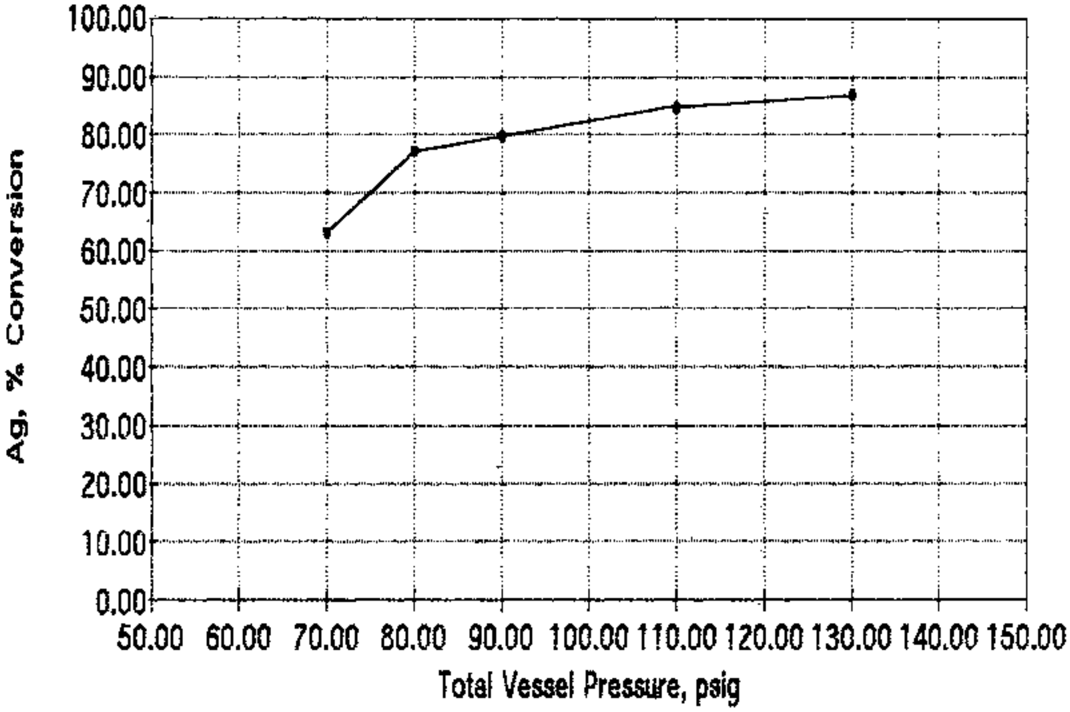
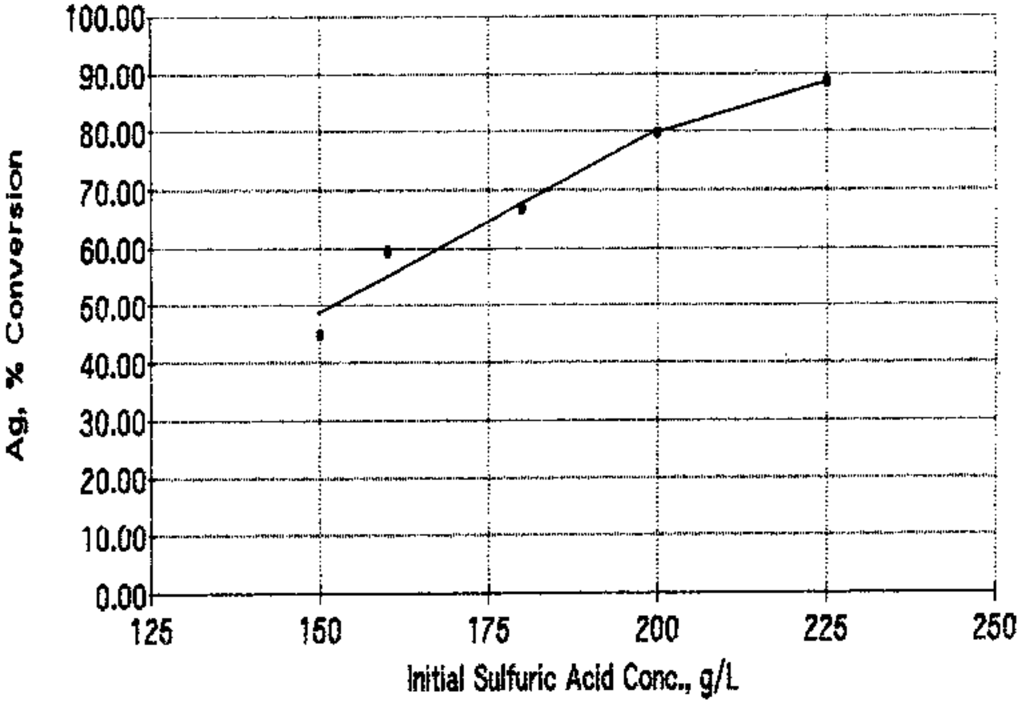


Figure 11
Effect of Free Acid on Ag



pressure was altered, the partial pressure (ie. oxygen overpressure) of the oxygen changed accordingly. As can be seen, the recovery of silver improved as the pressure increased. However, above 90 psig, the rate of improvement tended to diminish. This trend toward increased recovery is most likely due to the effects on the regeneration reaction:

$$2NO + O_2 = 2NO_2 \qquad (11)$$

and again

$$\text{Rate of } NO_2 \text{ produced} = k(PNO)^2 PO_2 \qquad (12)$$

Foremost, is the consequence of raised oxygen partial pressures on this reaction. As is easily seen, increased oxygen will drive the regeneration reaction to the right. Also, as this reaction sees a net decrease in the number of moles, the higher overall pressures will enhance this reaction. As with cooling, pressurization of this reaction during nitric acid manufacture is commonly practiced.

### Effect of Initial Free Sulfuric Acid Content

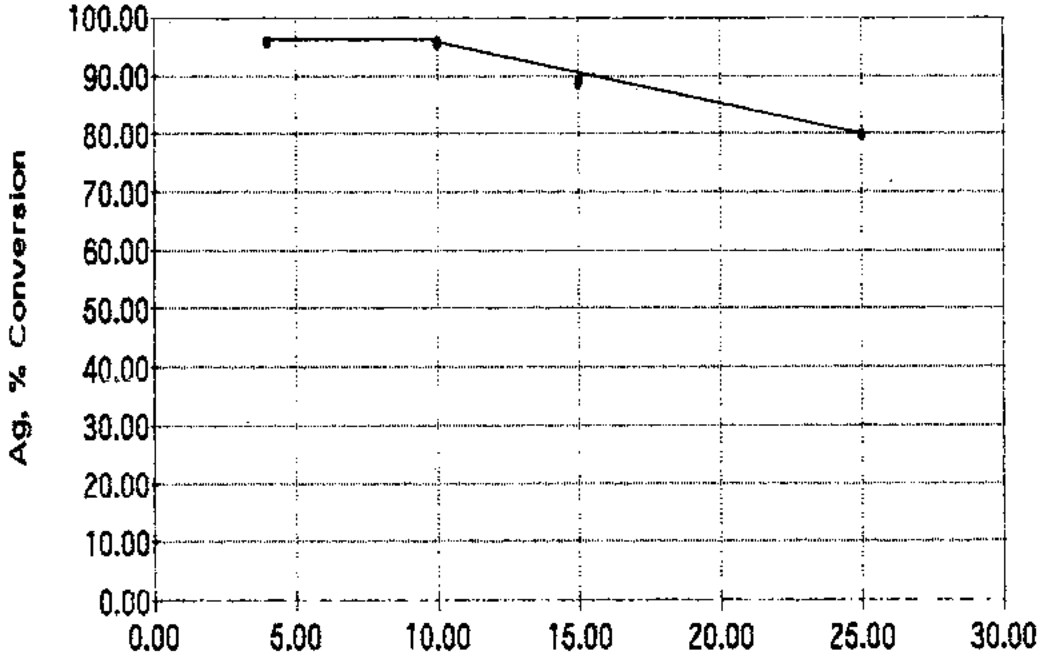
As was touched upon in the introductory passages, free acid content is a crucial element of each batch makeup. Due to the production of metastable sulfur, rather than sulfate ion, the leach reaction is a net user of free acidity. Thus, from a fundamental reaction standpoint, the presence of free acidity is essential for the process to progress. As Figure 11 illustrates, an increase in recovery with increased initial free acidity is no surprise. However, the most important reason for maintenance of a high free acidity in the leach is to stop silver robbing argentojarosite and plumbojarosite formation. Along with temperature and solution iron content, Tourre (19) and Dutrizac (20), defined free acidity as a key factor in jarosite formation. In practice, this must be balanced by the economics of eventual neutralization of any excess free acidity in the pregnant solutions. From actual operating experience, the leach is run to maintain a residual of approximately 115 g/L of free acid in the reacted slurry. This allows minimal jarosite formation, while lowering the amount of lime needed for eventual neutralization of tailings.

### Effect of Initial Particle Size

In plant practice, the residue from the antimony plant is directly leached in the silver refinery as received. A set of grinding tests were run to establish the effect of particle size on silver recovery. By using a laboratory ball mill and a slurry of 50% solids, different size distributions were obtained. These results are shown in Figure 12. A profound increase in silver recovery (from 80% to 95%) is found by reducing the material from $k_{80}$ = 25 micron to $k_{80}$ = 10 micron. Such dramatic improvement in conversion is usually indicative of a reaction controlled by diffusional effects. When a sulfur product layer is present, as is most certainly the case below 120° C in the Sunshine leach, diffusional effects will be a significant reaction barrier. Grinding to increase ultimate surface area is an effective means

Figure 12

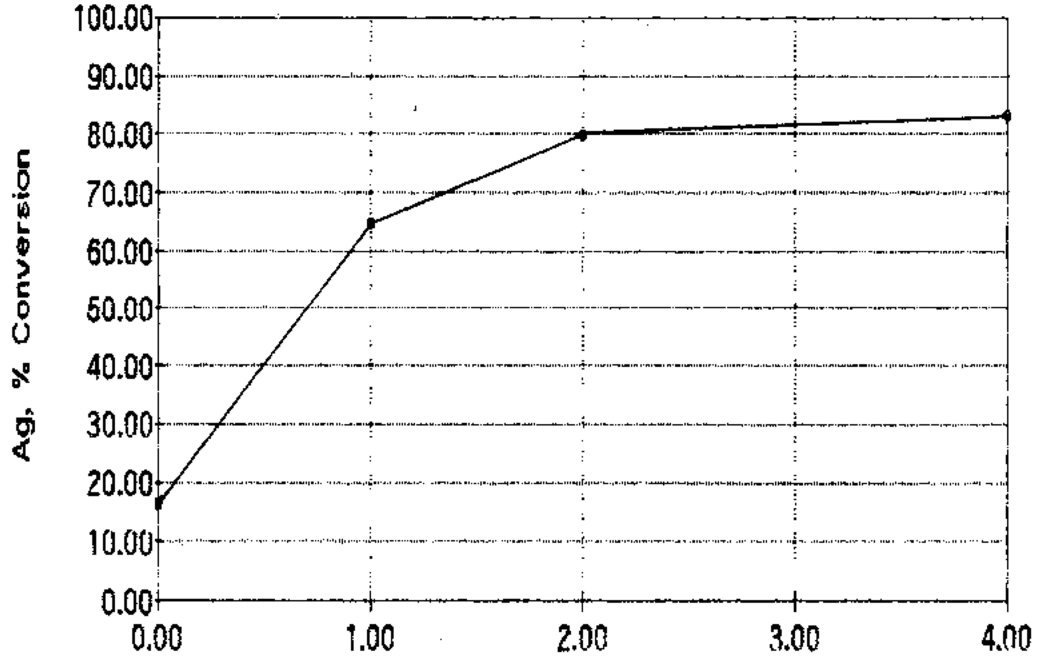
Effect of Regrinding on Ag



80% Passing Feed Size, Microns

Figure 13

Effect of Nitric Acid on Ag



Initial Nitric Acid Conc., g/L

of reducing this barrier. In the Sunshine system, both the sulfide oxidation reaction and the nitric acid regeneration reaction are enhanced by particle size reduction. This is because of the codependent nature of these reactions (ie. reaction products of one system serve to drive the other reaction). Thus, when the sulfide material is oxidizing faster and solubilizing more silver ,because of smaller particle size, more NO gas is produced. More NO gas serves to drive the regeneration reaction to higher rates and conversions. This in turn drives the sulfide oxidation reaction, and so on. This cycle continues throughout the entire leach time.

Moreover, because of the low temperatures where the sulfur product layer is occurring, this codependent cycle is also improved. As is known from the established regeneration reaction mechanism, low temperatures favorably increase nitric acid production rates, conversions and resolubilization. Thus, any increased sulfide oxidation realized through grinding is further driven by increased regeneration reaction phenomena when below 120° C. This fact would also help account for the dramatic rise in silver recovery experienced.

Furthermore, of all factors studied, particle size provided the most extreme improvement in silver solubilization. As will be discussed later, the possibility of a regrind circuit is a highly lucrative proposition, based on these test results. Besides the improvement in silver recovery, the effect on the solid by-products is substantial. The fine leach residue quality ,in terms of lowered silver content, is greatly improved, while its quantity, determined from overall mass balances, is diminished. In the sulfur pellets, very little silver remains. More importantly, the pyrite content is diminished since almost all of the iron in the feed is leached. Thus, deleterious cementation effects of pyrite on solubilized silver are all but eliminated. Also, from an overall process standpoint, this could be an important factor in lowering of reagent costs. A high purity sulfur product resulting from the silver refinery could be used as a reagent in the upstream antimony process. This would result in substantial savings in process costs. Thus, grinding to an optimum particle size is a very important parameter to adopt in plant operation.

## Effect of Nitric Acid Concentration

The results of varying initial concentrations of nitric acid are shown in Figure 13. As can be seen, little dissolution of silver takes place without the presence of nitric acid. Moreover, as its content is increased, silver conversion increases, though not as dramatically at higher levels. This is in accordance with the high oxidation potentials needed for leaching of silver sulfides. These elevated potentials, as well as greatly enhanced reaction rates, are provided by the properties of the nitric acid. As mentioned before, with increased nitric concentrations the probability of pressure spikes, caused by rapid reaction rates, is increased. Thus, though silver conversion does grow somewhat with greater additions of nitric, its level must be kept below the range of spike formation for safe leach operation. Additionally, one of the primary reasons for the economical operation of the Sunshine process is its reasonably small requirement of expensive nitric acid. The data in Figure 13 bear out the extent of this requirement. As can be seen, for the test slurry makeup used in the research, 2 g/L nitric acid concentration seems to be optimum.

## Effect of Initial Cupric Sulfate Concentration

To study the influence of cupric ion on silver conversion, varying amounts of cupric sulfate were added initially to the leach solution. As mentioned before, copper in solution is a vital element in plant leach makeup. Its presence stops the buildup of corrosive and toxic $H_2S$ in the preleach step. As can be seen in Figure 14, copper concentration has very little influence on silver recovery. Most certainly, this is due to the small influence it has on initial leach Eh. As cupric ion is a relatively weak oxidizer, compared to ferric ion, it does not enhance the oxidation potential of the slurry to any significant degree. Also, unlike ferrous ion, since the copper ion is already present in its highest oxidation state, it does not readily debilitate the oxidizing power of the nitric acid. Moreover, even in massive dissolved quantities, it does not limit the solubility of silver.

## Effect of Leach Pulp Density

Figure 15 represents the conversions experienced by an increase in percent solids in solution. Of all the variables studied this was the hardest to isolate. It also provided the most perplexing results. The difficulties associated with changing the pulp density involve its interdependence with other parameters in leach makeup. First, an increase in solids necessitates increase in cupric sulfate addition for preleach hydrogen sulfide elimination. Also, because of possible jarosite formation, more acid must be added initially to provide for the increased acid consumption of the higher solids. Thus, solids as a distinct variable cannot be isolated.

Nonetheless, experiments were completed which depicted an increase in recovery coinciding with an increase in percent solids. From the data, a doubling of percent solids in the vessel raises silver recovery up to near 90%. From a leach throughput and potential restart standpoint, this is a welcome result .

However, when pulp densities of 330 g/L were leached, almost no silver was solubilized. Copper, iron and other elements were affected little or not at all. Additionally, final free acidities were adequate to prevent jarosite formation. This strange phenomenon had been witnessed before in lab testing. At this time the exact mechanism fathering this result is not understood. However, knowledge of its existence does define, at least for now, a window of leachability for silver in terms of percent solids. For the present, the data indicates that doubling the amount of leach solids serves to increase recoveries and throughput. Successful adoption of doubling of leach percent solids, through some method of heat removal, remains as a lucrative goal to be achieved on a plant scale. However, more work should be done to identify the inhibition of silver leaching

Figure 14
Effect of Copper Conc. on Ag

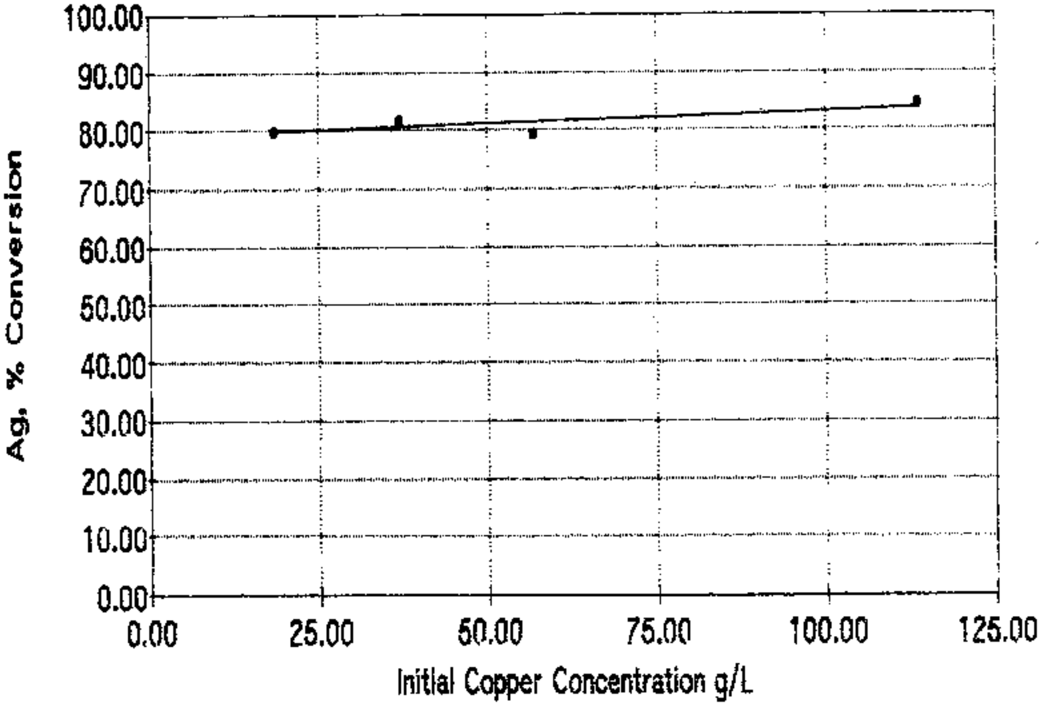
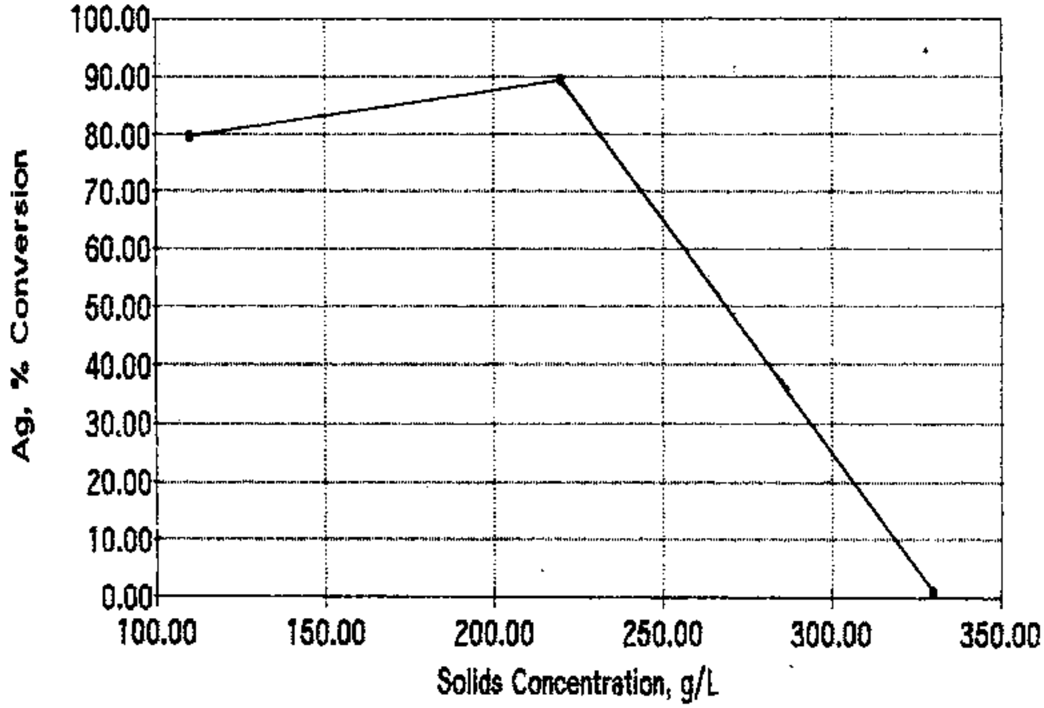


Figure 15
Effect of Solids Concentration on Ag



caused by extreme leach solids content. Someday, plant throughput expansion may hinge on knowledge of this mechanism.

## Restarting of Leaches

To investigate the possibility of stopping and restarting the leach reaction , a few tests were run. This concept could eventually be important on a plant scale for efficient heat removal, lowered reagent consumption and increased solids throughput. In a nutshell, what this encompasses is the notion that the rapid leach rates inherent with true nitric acid oxidation can be stopped or initiated whenever necessary. This is accomplished by adding a higher percent solids slurry to the pressure leach vessel. The normal amount of nitric acid is added and the leach is allowed to react to a temperature of 150° C. At this point, the reaction is stopped by relieving the oxygen overpressure and flashing the steam heat into the reactor feed tank. Upon cooling to 130° C, the vessel is again sealed and pressurized with oxygen. Another dose of nitric acid is added and the slurry again reacts to a temperature of 150° C.

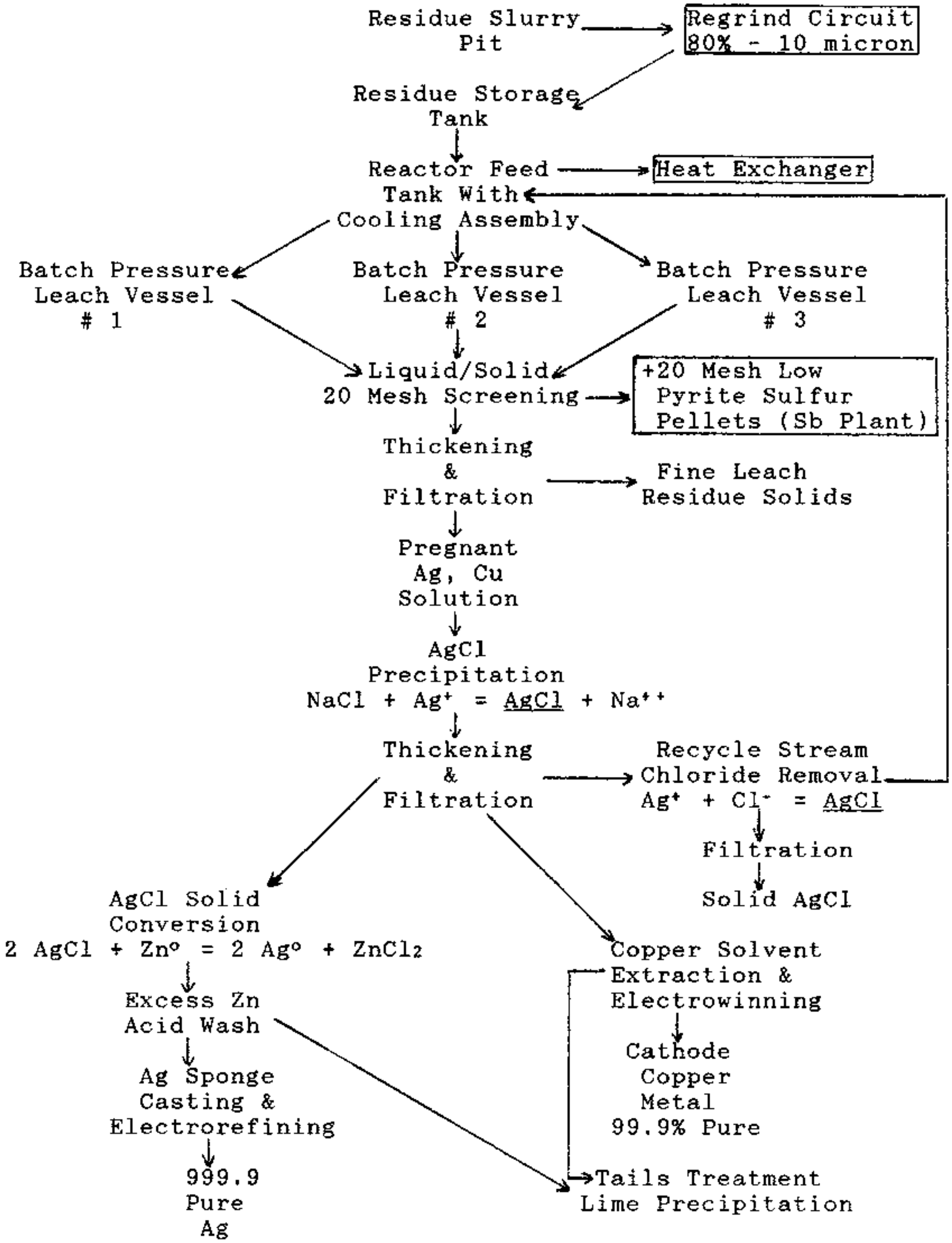
The advantages of this system are obvious. First, it allows increased throughput of solids in the batch reactors. This would allow plant scaleup and increased productivity to be achieved without increasing the reactor size. Next, it allows a fast and efficient method of heat transfer from the leach vessel into the preleach tank. This conveniently takes heat from where it is a process detriment and places it where it is a process asset. Also, this overcomes any complexities involved in designing and maintaining a cooling system in the existing pressure vessels. If need be, any required cooling could be done in the ambient pressure and low temperature reactor feed tank. Design, construction and maintenance of a cooling package in this vessel would be much simpler and more cost effective. Lastly, it is a method for considerable savings on sulfuric acid. For effective jarosite abatement, plant practice dictates a final leach concentration of 115 g/L of free sulfuric acid to be present in each batch. Ultimately, all of this sulfuric acid winds up in a waste stream requiring neutralization with slaked lime. This is costly. However, if this maintenance dose of final acidity is used in a higher than normal solids concentration leach batch, it is evident that savings could be realized.

Along these lines, testwork in the lab indicated that leach restarts could be achieved quite readily. It was important to scale up the leach batches appropriately in terms of solids, acidity and copper sulfate addition (for $H_2S$ abatement). On the lab scale, this is no problem. It was also found important not to flash off heat to below 125° C. Here sulfur begins to resolidify, which can cause lowered reaction rates of solids due to occlusion, preventing restart.

In the plant, the conditions for, and the results of, restarting were different. Although, several successful restarts have been accomplished using the lab parameters, in some cases silver conversions were somewhat diminished. This is thought to be a consequence of the lessened control over leach batch makeup and pressure vessel filling in the plant. Thus, in some attempts, too high percent solids were used in the batch makeup resulting in little silver dissolution. This phenomena is in

Figure 16

Sunshine Silver Refinery
Flowsheet



accordance with lab results generated when dealing with high percent solids. Unfortunately, the nature of this silver recovery depression is not yet understood. However, with better plant controls, restarts would be more readily instituted. Considering the initial success in plant restarts (a major achievement in itself) and the inherent potential in harnessing this concept, more work on maintaining a high silver conversion must be done.

## Conceived Plant Modifications

From the research work an understanding of the leach system was obtained. This knowledge can be directly applied to upgrade the plant process. A possible scenario is depicted in Figure 16.

As may be seen, only three changes are proposed to be necessary in the plant. This keeps to a minimum any massive restructuring of the existing facility. Of course, this is a priority, as more costs are always involved in plant restructuring of any consequence.

First of all, a regrind circuit is proposed to reduce the pressure leach feed size to an optimum value (ie. 80% - 10 micron). From lab test work, this will allow silver conversions up to 95 % and enhanced reaction rates.

Next, as a consequence of the enhanced recoveries from regrinding, the ensuing high grade sulfur pellet product can be recycled to the antimony plant. Here it could effectively be used as a reagent in the caustic sodium sulfide leach system. This would alleviate materials handling and smelter refining charges. Also, it would lower the unit antimony processing costs.

Lastly, is a heat exchanger attached to a cooling system in the reactor feed tank. Used in conjunction with the aforementioned plant leach restart scenario, this would be used as a heat sink for flashdown of the pressure vessels. This scheme would allow all of the previously mentioned benefits of restarting to be realized in a more efficient manner. It would also allow better capture and control of flash heat from the pressure vessels. In turn, this waste heat could be put to improved use in the refinery or antimony plant.

Overall, these changes could allow increased productivity with lowered processing costs. Although some still require more investigation (ie. control and conversion in leach restarts), for the most part, all could be integrated with the existing plant layout with a minimum of facilities revision.

## Results

1. A laboratory leach system was developed to accurately and reproducibly reflect the plant leach system. Using this, a baseline test of the system was established.

2. Regrinding of copper residue feed improved silver recoveries up to 95%.

3. Formation of silver robbing argentojarosite and plumbojarosite were identified as reaction products under specific leach conditions. Low free acidity greatly enhanced formation of these compounds.

4. The mechanism of pyrite cementation of solubilized silver was found to cause recovery losses in the leach. Proper flashing and discharging of the reactors avoids this problem.

5. Initial Eh values of the slurry, as modified by ferric and ferrous ion, were found to have a profound impact on ultimate silver recoveries.

6. The reaction was found to be very sensitive to temperature. An optimum temperature of 125° C, obtained by reactor cooling, was found to give the highest silver recovery.

7. Conditions necessary for stopping and restarting of leaches were found. This was verified in plant tests.

8. The amount of copper in solution had little bearing on silver recovery.

9. The amount of preleach time had little effect on silver recovery. For feed containing large quantities of siderite, this may not be true.

10. Extended times of leaching in the pressure vessel gave an increase in silver recovery. More importantly, keeping the reaction under pressure with nitric acid alleviated precipitation problems of silver caused by pyrite.

11. An increase in percent solids in the leach gave an increase in silver recovery. However, extremely high percent solids in the leach caused a dramatic decline in silver recovery.

12. An optimum value of 2 g/L of nitric acid was found adequate for the conditions of the baseline leach.

13. An increase in vessel pressure (and thus, a corresponding increase in oxygen partial pressure) increased silver solubilization. However, at pressures above 90 psig, the effect of raised pressure became diminished.

## Conclusions

1. Regrinding of copper residue feed should be incorporated in plant operation. Potentially, it will raise silver recovery to 95%, increase sulfide oxidation rates, and provide a high grade sulfur pellet product.

2. Flashing and restarting of leaches in the plant can be a potentially lucrative practice. In order to fully utilize it, more work must be done to define the mechanism behind hindered silver dissolution when leaching extremely high percent solids. Also better control of actual solids content must be established.

3. Cooling of the leach vessels is important for increased recovery and increased solids throughput. The most cost efficient

,in terms of ease of design, reactor volume loss and eventual maintenance requirements, should be put in place. If restarting of leaches in the plant proves viable, cooling is best done in the reactor feed tank.

4. The current requirement of 90 psig reactor pressure is adequate for current operations. This is true for two reasons. First, the effect of increased silver recovery corresponding with increased vessel pressure becomes lessened above 90 psig. Also, the building and maintenance costs associated with higher pressure vessels increase dramatically with increased pressure.

## Acknowledgements

The authors would like to thank the Sunshine Mining Company for supporting this work. In particular, this includes all of the people whose efforts in the startup phase allowed the refinery to be an operating entity today. Also, we would like to thank John Ackerman and John Allen for their constant suggestions and support. A thanks also goes to each of the leach operators (Tom Pouttu, Janet Carpenter, Ernie Hoffman, Brian Eifler, Cindy Mayo, Pat Williams and Bob Dionne), whose patience allowed the plant restart work to achieve its first success. Lastly, but most importantly, we would like to acknowledge the contribution of Ernie Krys. Without his speedy and accurate analysis of the samples, none of this work would have been possible.

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# A REVIEW OF PLATINUM GROUP METAL RECOVERY FROM US SOURCES

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# A Review of Platinum Group Metal Production From US sources

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## Abstract

U.S. platinum-group metals (PGMs) production remains modest relative to domestic demand but strategically important for automotive emissions control, chemical processing, and emerging hydrogen applications. After a brief literature survey of market structure and policy, this review presents case studies of current operations of Sibanye-Stillwater's Stillwater–East Boulder (MT) and by-product recovery linked to the Eagle Mine (MI), followed by prospective projects including Twin Metals Minnesota, NewRange Copper Nickel, and the Tamarack (Talon Metals) project. Contextual statistics highlight scale and reliance: in 2024, world mine output was approximately 190,000 kg palladium and 170,000 kg platinum; U.S. mine production accounted for roughly 8,000 kg Pd (~4%) and 2,000 kg Pt (~1%), leaving the United States ~36% import reliant for palladium and ~85% import-reliant for platinum on an apparent consumption basis. Recycling statistics show approximately 120,000 kg of palladium and platinum were recovered globally from scrap in 2024, including about 45,000 kg Pd and 8,500 kg Pt from U.S. automobile catalytic converters, evidence that domestic secondary supply already plays an outsized role. Across cases, the paper distills practical constraints (refining access, assay/settlement lag, logistics) and identifies actions that could materially strengthen U.S. PGM availability over the next decade without presuming large greenfield mines.

## ACKNOWLEDGEMENTS

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## 1: Introduction

The scope of this review article is to provide a summary of domestic USA PGM deposits, beneficiation, refining methods, and capabilities, with a focus on mining operations. Platinum group metals (PGMs) include platinum (Pt), palladium (Pd) rhodium (Rh), ruthenium (Ru), iridium (Ir), and osmium (Os). Their unique properties, such as high melting points, biocompatibility, corrosion resistance, and high specific catalytic activity make them essential for a variety of industrial, defense and medical applications. These elements have similar chemical and physical properties and are exceedingly difficult to separate.

The three most abundant and industrially used PGMs are platinum, palladium, and rhodium. There are a limited number of primary PGM mines, and they are often found around the world as a secondary product. Most primary Pt and Pd mines reside in South Africa and Zimbabwe [1]. The largest producer of PGMs since 1920 has been the Bushveld Complex in South Africa, with around 77% of the world's Pt supply and 41% of the world's Pd supply in 2010 [2]. The minor PGMs (Rh, Ru, Ir and Os) are only extracted as by-products, and there is no primary deposit source. Figure 1 details the sources of five of the most consumed PGMs. Since PGMs are often found in nickel and copper sulfide ores, they can also be extracted as a secondary product in association with other mines.

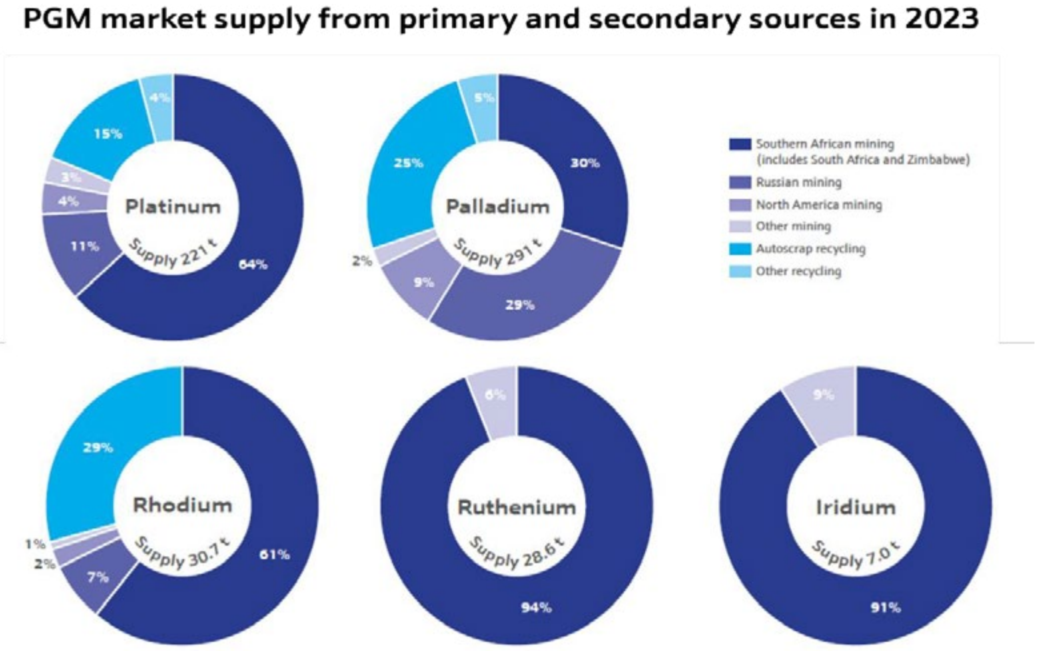

Figure 1. PGM market supply from primary and secondary sources in 2023 [1].

The United States is the 5th largest PGM producer in the world, but still stands at only a fraction of South Africa's production [3]. In 2022, the total global supply of platinum was 224.3 tpy (i.e. tonnes per year) , which was a 13.9% decrease from the previous year [4]. For palladium, the total global supply was around 300 tpy and has been stable for the last 8 years. Rh, Os, Ru, Ir, each had a global supply of 21.4 tons, 0.5 tons, 23.4 tons, 6.8 tons, respectively. Domestically, there was 10,000 kg of palladium and 3,000 kg of platinum produced from mining [5]. While domestic consumption increased by 8,000 kg in 2023, the increase in global supply drove the price down. Overall, the estimated annual average price for PGMs in 2023 decreased, by 51% for rhodium, by 30% for palladium, and by 19% for ruthenium when compared to average 2022 prices [5].

For the United States, the low domestic concentration translates into chronic import dependence and a supply-demand deficit. In 2024, the only significant domestic mine, Sibanye's Stillwater-East Boulder complex in Montana, produced about 426,000 oz of combined platinum and palladium, which makes up only 7% of U.S. consumption [6]. Even robust recycling at its Columbus refinery (~ 316,000 troy ounces of PGMs recovered from spent catalysts) is not enough to bridge that gap. The nation remained 83% import reliant for platinum and 37% for palladium in 2024, according to the latest USGS Mineral Commodity Summary [7].

Federal policy is pushing hard in that direction by providing massive amounts of funding for mining, tailing-reprocessing and recycling projects that could shorten PGM supply chains [8]. In 2022, The U.S. Secretary of the Interior included all of the PGMs in its list of critical minerals [9]. This list includes elements that are considered essential to the economic growth or national security of the United States; that have a supply chain vulnerable to disruption; and that serve an essential function in the manufacturing of vital industrial products.

## 1.1 PGM applications and global production

PGMs are used in a wide variety of products and industries, from computer screens to automobile airbag deployment mechanisms and cancer treatments [10]. Since 1979, the highest usage of PGMs is for catalysts used in catalytic converters in cars to reduce harmful emissions [11]. Other main uses include investments, jewelry, and other industrial and dental and medical products. Pt, Pd, and Rh also are highly resistant to chemical reactions, have a high melting point, and have stable electrical properties, making them popular materials for industrial products like crucibles. In addition, they are often used as catalysts in the chemical engineering industry to catalyze the partial oxidation of ammonia for nitric oxide production, in fertilizers.

Recently, PGMs have been increasingly used for synthetic organic chemistry and to refine crude oil and other petroleum products. While PGMs have a large breadth of use, most applications require a small amount of PGMs for use due to their efficiency [12]. The energy transition and growing popularity in renewable energy-based systems and electric vehicles may cause PGM demand to face massive decline soon. A breakdown of the demand for Pt, Pd, and Rh can be seen in Figure 2.

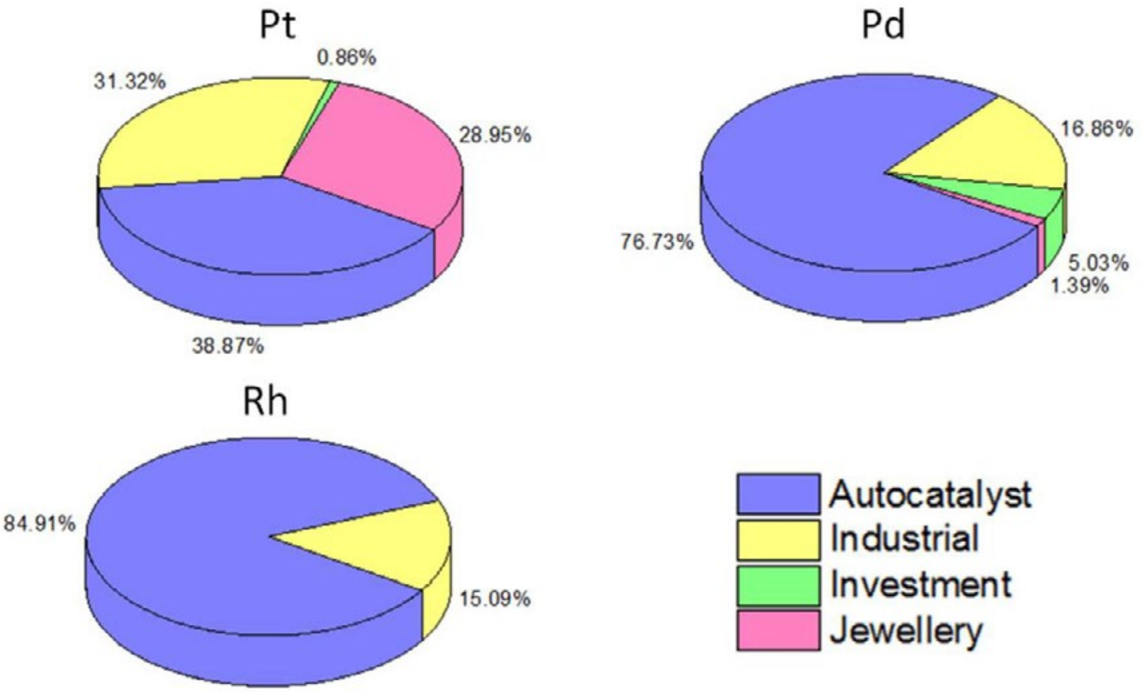

Figure 2. Breakdown of Pt, Pd, and Rh net demand, expressed as percent of total demand in 2019 [11].

To increase the concentration of minerals and obtain enriched products, flotation is the most widely used process in the beneficiation of PGMs. This process can be limited by several factors, one of which is the mineralogy of the ore body. Since PGMs vary in composition, different concentrates are produced through the flotation process. PGMs that are produced in mining operations are typically found in sulfide deposits, or in small-scale gravel and sand deposits. They are often accompanied by other metals such as nickel, iron, copper, chromium, tellurium, selenium, gold, and silver [13], [14], [15]. Over the last decade, the price of PGMs has increased dramatically, with Rh and Ir rising the most, as displayed in Figure 3.

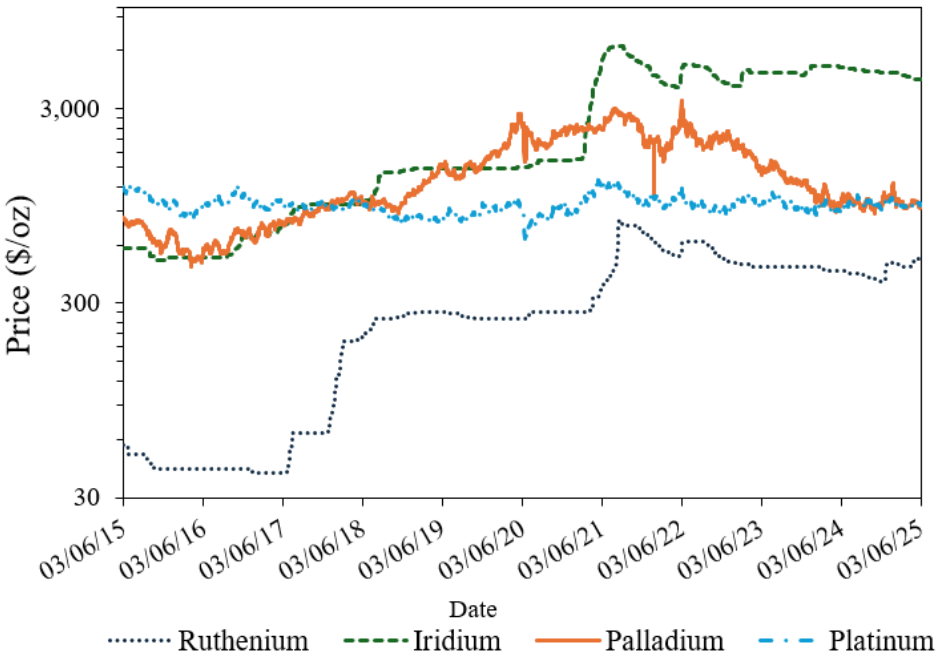

Figure 3. Daily PGM prices in $/oz from 3/06/2015 to 3/06/2025, based on the Johnson Matthey London Commodity Dataset on Factset.

### 1.2. Historical domestic PGM deposits

While Sibanye-Stillwater currently is the only major PGM producer in the United States, there have been many more deposits of varying grade and resources located. The last comprehensive report of PGM occurrences in the continental United States by USGS was in 1993, with the previous being in 1977 [16], [17]. A table of the reported deposits in 1977 and then in 1993, divided by state is shown below in Table 1. PGMs presence in many of these deposits, or if they were at an economically feasible levels to extract, was unverified. The minimum PGM concentration to make the confirmed deposit list was 100 ppb [17]. For a mining operation that produces PGMs as a primary product, a minimum grade of around 3 ppm is normally required to be economically feasible. To produce PGMs as a secondary product, a minimum concentration of around 250 ppb is required. The deposits that have been confirmed and plotted on maps are also in Table 1. Some of the deposit types are unknown, but the known ones include placer Au-PGMs, porphyry copper, podiform chromite, and low-sulfide gold-quartz veins [17].

The initial extensive literature search on PGMs was done in in 1977 by Blair et al., which looked into any sites where PGMs were mined, documented, analytically determined, or reported by individuals [16]. The results shifted in the 1993 report, as more sensitive analytical equipment was available allowing for a more precise analysis of PGM ores, specifically podiform chromite deposits. The search was limited to PGMs in the contiguous United States, leaving out Alaska, which has had several large-scale placer deposits recorded in the Goodnews Bay area [18]. Goodnews Bay district in Alaska first had placer deposits discovered in 1926, and several small scale mines operated there in the following years [18]. These small-scale operations produced around 3,000 troy ounces of PGMs over a period of 7 years. In 1937, Goodnews Bay Mining Company began operation as the first large scale commercial mine [19].

At the time, the Goodnews Bay Mining Company was the only domestic mine that produced PGMs as its principle commodity [19]. The mine mostly produced platinum and lesser amounts of iridium. It produced around 650,000 troy ounces of PGMs between 1928 to 1975, averaging 13,800 troy ounces (391 kg) per year [20]. Operations were then taken over by Hanson industries, which operated sporadically in the 1980's, until it stopped producing, and was eventually acquired by XS Platinum Limited in 2008 [21]. At first, XS Platinum Limited had plans for a tailings reclamation project to recover platinum from the waste stream. This eventually fell apart when the company and 5 of its officers were indicted by a federal grand jury for felony violation and conspiracy to violate the Clean Water Act [22].

In addition, before the Goodnews Bay platinum operation, there was the Salt Chuck Mine on Prince of Wales Island, which was the only palladium producer in the United States between 1915 to 1941 [23], [24]. The operation began as a copper mine in 1905, which was its primary product until the discovery of palladium in the deposit in 1915. The deposit consisted of mostly magnetite, bornite, and lesser amounts of chalcopyrite and malachite. Salt Chuck produced 326,000 tons of ore with copper, gold, silver and palladium, with an average palladium grade of 1.96 g/t existing mostly as kotulskite (PdTe) [25]. The processing mill at the time started at 30 tons per day, which then grew to 300 tons per day by 1923. In 1923, the name was changed to Alaskan Palladium Mining CO, before operations were suspended in 1926. It then went through a period of operation, suspension of operation, and selling until 1941 when it stopped operation for good. They had a reported palladium recovery of 65%.

Table 1. Summary of Reported and Confirmed PGM Deposits in the Continental United States, 1977 and 1993 [16], [17].

| **State** | **Number of Reported Deposits (1977)** | **Number of Reported Deposits (1993)** | **Number of Confirmed Deposits (1993)** |
|---|---|---|---|
| Arizona | 4 | 8 | 1 |
| Arkansas | 1 | 1 | 0 |
| California | 88 | 143 | 117 |
| Colorado | 9 | 10 | 5 |
| Georgia | 0 | 2 | 0 |
| Idaho | 12 | 17 | 9 |
| Maine | 0 | 1 | 0 |
| Maryland | 1 | 1 | 1 |
| Minnesota | 2 | 7 | 7 |
| Montana | 12 | 78 | 67 |
| Nevada | 9 | 18 | 11 |
| New Jersey | 0 | 1 | 1 |
| New Mexico | 6 | 7 | 1 |
| New York | 1 | 2 | 2 |
| North Carolina | 3 | 9 | 8 |
| Oklahoma | 0 | 2 | 1 |
| Oregon | 57 | 115 | 68 |
| Pennsylvania | 1 | 3 | 2 |
| Texas | 1 | 1 | 0 |
| Utah | 5 | 11 | 4 |
| Vermont | 0 | 1 | 1 |
| Virigina | 0 | 2 | 1 |
| Washington | 17 | 33 | 19 |
| Wyoming | 16 | 32 | 11 |
| Total: | 245 | 505 | 337 |

## 2. Current domestic PGM operations

The main producer of PGMs is Sibanye-Stillwater, in Montana, with a small amount also being produced by the Lundin Eagle mine in Michigan. Sibanye-Stillwater had an estimated $880 million worth of PGMs produced during 2022, which then dropped to \$510 million in 2023 and \$310 million in 2024 [5], [26].

## 2.1 Sibanye-Stillwater Complex

The Sibanye-Stillwater Complex in Montana accounts for the majority of PGMs produced in the United States, with platinum and palladium as its primary products [27]. They have two main mine sites: Stillwater and East Boulder mines, both of which are underground mining operations rich in PGMs [28]. With the Stillwater mine expected to be open until 2049 and the East Boulder mine expected to be open until 2059, the facility is expected to produce PGMs domestically for many years to come. They currently produce around 425 k troy ounces of platinum and palladium per year and process it in the Columbus smelter complex.

The deposits are found in the J-M reef, which is a high-grade primary magmatic ref-type Pd-Pt deposit that both the Stillwater and East Boulder Mines are operated out of [29]. It was first explored in the 1960’s, but industrial operation only began in 1986. The Stillwater mine has a measured and indicated concentration of 10.87 g/t of Pd and 3.14 g/t of Pt, while the East Boulder mine has a measured and indicated concentration of 9.41 g/t of Pd and 2.58 g/t of Pt. The mines have a processing capability of 700,000 tpy at Stillwater, and 600,000 tpy at the East Boulder facility. For mining, both use a mechanized ramp and fill method as their dominant mining method [29]. Once the material is mined, it is sent to each mine’s respective ore processing facility.

### 2.1.1 Stillwater concentrator

The Stillwater concentrator was opened in 1986, and can process around 3,400 tpd through the crushing, milling, flotation and filtration circuit [29]. The goal of the concentrator is to liberate and separate the PGMs in the sulfides from the surrounding silicate gangue. For primary crushing, they use a jaw crusher, which crushes the mined ore to minus 150 mm [30]. Their primary grinding consists of a SAG mill, pebble mill and a ball mill charged with steel balls. The product is a cyclone sized ore product of p80 145 μm. After material is ground to size, it is sent to a flotation circuit for upgrading. Their flotation circuit starts with a flash flotation circuit, rougher flotation circuit. In addition, they have a rougher-cleaner flotation circuit, a middling flotation circuit, a middling cleaner flotation circuit, scavenger flotation circuit, and a scavenger cleaner flotation circuit. They also have a regrind stage that helps to optimize smelter operation down the line by improving grade and recovery through increased liberation. Their reagents used for flotation include potassium amyl xanthate (PAX), di-thiophosphate (Cytec 3477), carboxymethyl cellulose (CMC), methyl isobutyl carbinol (MIBC), sulfuric acid for pH adjustment, and a flocculant [30]. For filtration, they use an 8-plate pressure filter, which produces a concentrate with 11-13% moisture. A simplified block flow diagram of the Stillwater Concentrator can be seen in Figure 4.

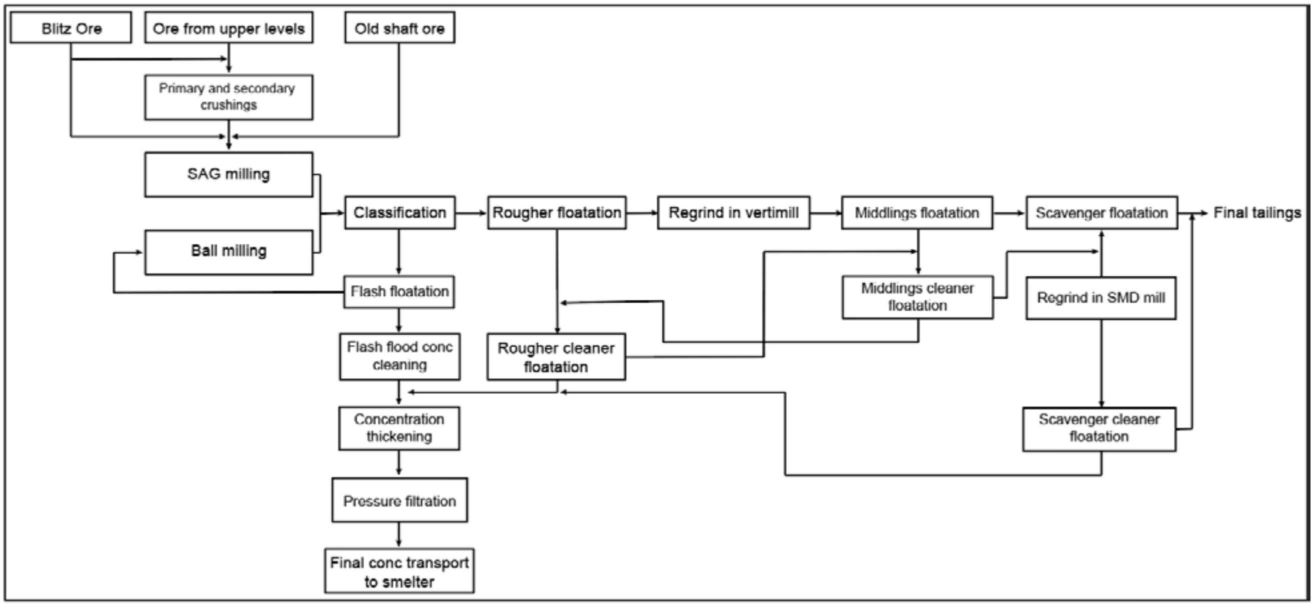

Figure 4. Block Flow Diagram of the Stillwater Concentrator [29].

### 2.1.2 East Boulder concentrator

The East Boulder concentrator operates a similar crushing, milling, flotation, and filtration setup to process 2,000 tons per day of PGM-rich sulfide material [29]. For primary crushing they utilize a jaw crusher that crushes the ore to minus 150 mm. Primary grinding consists of a SAG mill, pebble mill, and a ball mill, with similar capacities as the Stillwater Facility. Their flotation circuit consists of flash flotation, rougher and rougher-cleaner flotation, middling and middling cleaner flotation, as well as scavenger and scavenger cleaner flotation circuits. They also have a similar reagent scheme to the Stillwater facility. In addition, they have a regrind circuit to ensure recovery and concentrate grades are optimized for smelting. For filtration they use an 8-plate pressure filter, which produces a concentrate with 11-13% moisture. The East Boulder concentrator is similar in its process to the Stillwater concentrator seen in Figure 4.

### 2.1.3 Columbus Metallurgical Complex - Stillwater Smelter

Once the sulfides are concentrated at their respective facilities, they are shipped as a filter cake to the Stillwater Smelter at the Columbus Metallurgical Complex (Columbus, Montana) for processing [29]. The smelting facility has expanded over the years to include base metal refining, as well as PGM recycling operations. The complex is one of the largest recyclers of PGMs derived from spent catalytic converters and other industrial sources. In 2024, Sibanye-Stillwater processed and recycled a total of 316,470 troy ounces of PGMs from spent recycled catalytic converters at the Columbus metallurgical facility [31].

The smelter flowsheet consists of a concentrate dryer, two electric submerged arc furnaces (EAFs), two top-blown rotary converters (TBRCs), and an off-gas treatment system [29]. Upon arrival, the primary concentrate is processed through a fluidized bed dryer, reducing its moisture content to below 1% to prevent operational hazards and slag foaming. Dried concentrate is then blended with pulverized and sampled spent automotive catalysts before being fed into the two 150-tpd electric furnaces. The furnaces receive a mixture of concentrate, limestone flux, and recycled dust through air-slide conveyors [30]. During smelting, limestone decomposes according to:

$$CaCO_3 \rightarrow CaO + CO_2 \qquad [1]$$

producing CaO that combines with siliceous gangue to form calcium silicates

$$CaO + SiO_2 \rightarrow CaSiO_3 \qquad [2]$$

As temperatures exceed 900 °C in the EAF, the sulfide minerals contained in the concentrate undergo thermal decomposition and re-equilibration to form a homogeneous molten matte. For example, chalcopyrite breaks down to copper and iron sulfides:

$$CuFeS2 \rightarrow Cu2S + FeS + S \qquad [3]$$

while pentlandite decomposes to nickel and iron sulfides:

$$(Fe,Ni)_9S_8 \rightarrow 3Ni_3S_2 + FeS \qquad [4]$$

and pyrrhotite disproportionated toward FeS:

$$Fe_{1-x}S \rightarrow FeS + x\,Fe_{(l)} \qquad [5]$$

with dissolved Fe contributing to the matte phase. These reactions form a molten matte composed primarily of $Cu_2S$, FeS, and $Ni_3S_2$, which collects the PGMs. Slag, consisting mainly of silicon and iron oxides, is tapped several times per day at 1500–1550 °C, air-cooled, and recycled back into the furnace feed. The matte is tapped every eight hours into ladles and granulated before conversion. The furnace off-gases pass through a primary and secondary baghouse system, achieving ≥99.7% $SO_2$ capture efficiency [30]. The granulated matte is charged into the top-blown rotary converter, where iron and sulfur are selectively oxidized. The dominant reactions include oxidation of iron sulfide:

$$3FeS + 5O_2 \rightarrow Fe_3O_4 + 3SO_2 \qquad [6]$$

followed by formation of calcium ferrite slag in the presence of added lime:

$$2Fe_3O_4 + \frac{1}{2}O_2 + 3CaO \rightarrow 3CaFe_2O_4 \qquad [7]$$

These reactions remove iron and sulfur from the matte and transfer them to the converter slag, while enriching the remaining matte in Cu, Ni, and PGMs. The converter slag is granulated, dried, and recycled back to the electric furnaces, while the upgraded matte typically containing 350–700 oz/t Pt+Pd, along with around 29% Cu, 41% Ni, 21% S, 2% Fe, and the rest Co, Au, Ag, Rh, Te, or Se is granulated and sent to the Base Metal Refinery for further processing.

To improve the throughput capacity of the smelter in response to increasing mine production, several upgrades have been made. This includes a new concentrate receiving facility, improved smelter and gas cleaning, a granulation facility, upgraded top blown rotary converter drums, sulfur dioxide off-gas scrubbing, and a sodium hydroxide regeneration train. A simplified block flow diagram of the smelter can be seen in Figure 5.

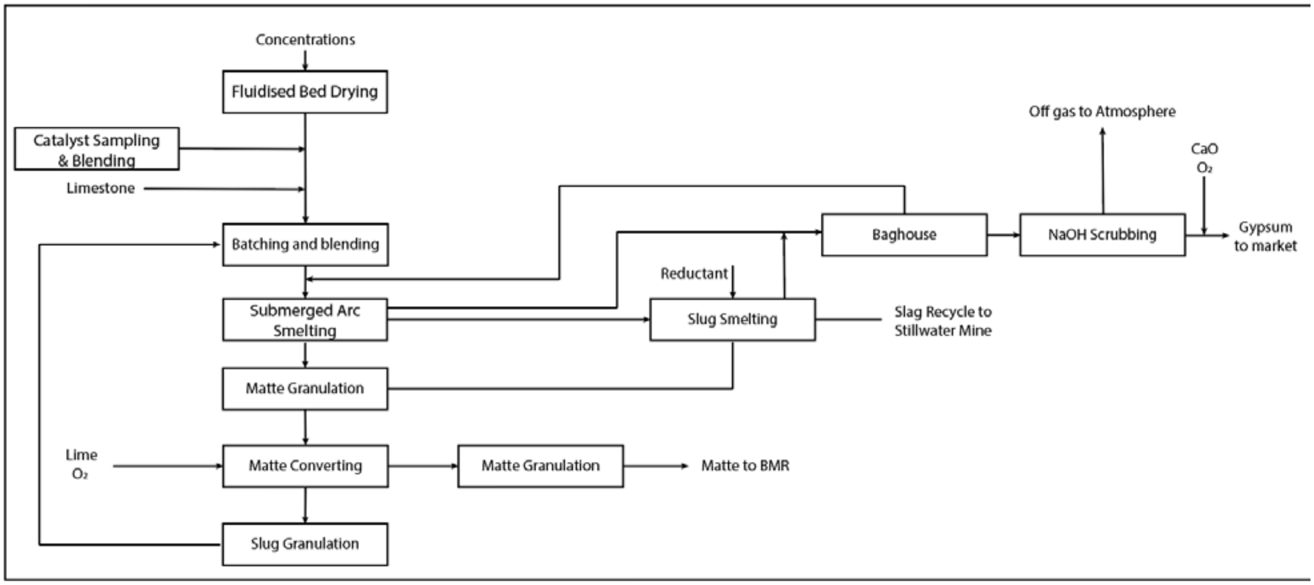

Figure 5. Block Flow Diagram of Stillwater Smelter [29].

The Columbus Metallurgical Complex also hosts a Base Metals Refinery (BMR) to take the smelter matte and produce a high grade PGM filter cake [29]. It consists of grinding, atmospheric leaching, pressure leaching, PGM concentrate separation, iron precipitation, nickel sulfate salt production, and copper cathode production. The fundamental process follows the patented two-acid Sherritt Leach Process. It was built in 1996 and outfitted to manage 300 kph of smelter matte but has been upgraded to manage 600 kph of smelter matte. It was first built to extract the base metals, which reported to a mixed copper-nickel solution for treatment off-site, but was later modified to separate the nickel and copper to produce a nickel sulphate salt and cathode copper. The capacity increase was primarily a result of improved process optimization and an expansion of the copper electrowinning circuit, which had historically been the bottleneck. Its current configuration consists of a nickel atmospheric leach circuit, a copper dissolution autoclave, a nickel sulfate crystallizer, a copper electrowinning plant, and two polishing leach autoclaves.

When the granulated converter matte is received at the BMR facility, it is milled in a batch-wise process in a tower mill to produce material with a p80 of 75 μm and a 80% solid slurry [30]. It is then leached with sulfuric acid and oxygen at atmospheric conditions, to remove nickel as a sulfate crystal product [29]. The atmospheric metathesis based leach system has a series of five cascading agitated tanks, and typical reactions that occur are as follows:

$$Ni^o + H_2SO_4 + 0.5O_2 = NiSO_4 + H_2O \tag*{[8]}$$

$$Ni_3S_2 + H_2SO_4 + 0.5O_2 = NiSO_4 + 2NiS + H_2O \tag*{[9]}$$

$$Ni_3S_2 + 2CuSO_4 = Cu_2S + 2NiSO_4 + NiS \tag*{[10]}$$

$$Fe^0 + H_2SO_4 + 0.5O_2 = FeSO_4 + H_2O \tag*{[11]}$$

Typically, any PGMs that are present will co-precipitate with the copper, and any unleached residue will be separated via a thickener and sent to the pressure leaching stage. These solids are treated with sulfuric acid and oxygen in a pressure leach to dissolve the copper, selenium, and tellurium. This includes the nickel and copper sulfides, as well as iron in the form of magnetite and hydrated ferric oxide. The pressure leach stage operates in lower acid concentrations of 20-25 g/L and temperatures around 135°C. No magnetite will dissolve under these conditions. The reactions present in the pressure leach autoclave are:

$$NiS + 2O_2 = NiSO_4 \tag*{[12]}$$

$$Cu_2S + H_2SO_4 + 2.5O_2 = 2CuSO_4 + H_2O \tag*{[13]}$$

$$Fe_2O_3 + 3H_2SO_4 = Fe_2(SO_4)_3 + 3H_2O \tag*{[14]}$$

The selenium and tellurium are subsequently cemented out of solution, and the copper-rich solution is sent for electrowinning. The underflow from the pressure leach is treated with a polishing leach to remove any remaining impurities or base metals from the PGM concentrate. The copper electrowinning circuit operates continuously, and was expanded in 2021 to add six cells, which eliminated a historic bottleneck to the BMR process. This allows the BMR to produce 750 tpy of copper cathodes. Any remaining solids then form the PGM filter cake, which is washed, filtered, and dried.

The nickel sulfate present in the atmospheric leach thickener overflow is vacuum crystallized and dried in a fluidized bed before being sold as a nickel sulfate salt. Some minor PGM metals, such as Rh, Ru, and Ir go into solution and follow the nickel, which are then precipitated via the Thakadu process to prevent any losses [32]. This process generally involves pre-treatment, solvent extraction, and crystallization, which then increases recovery of PGMs and produces pure nickel sulfate salt.

The BMR can produce a bullion concentrate containing about 60% PGMs, as well as nickel and copper co-products. It is then sent to Johnson Matthey Company for any further separation and refining that is required. A simplified block flow diagram of the BMR can be seen in Figure 6.

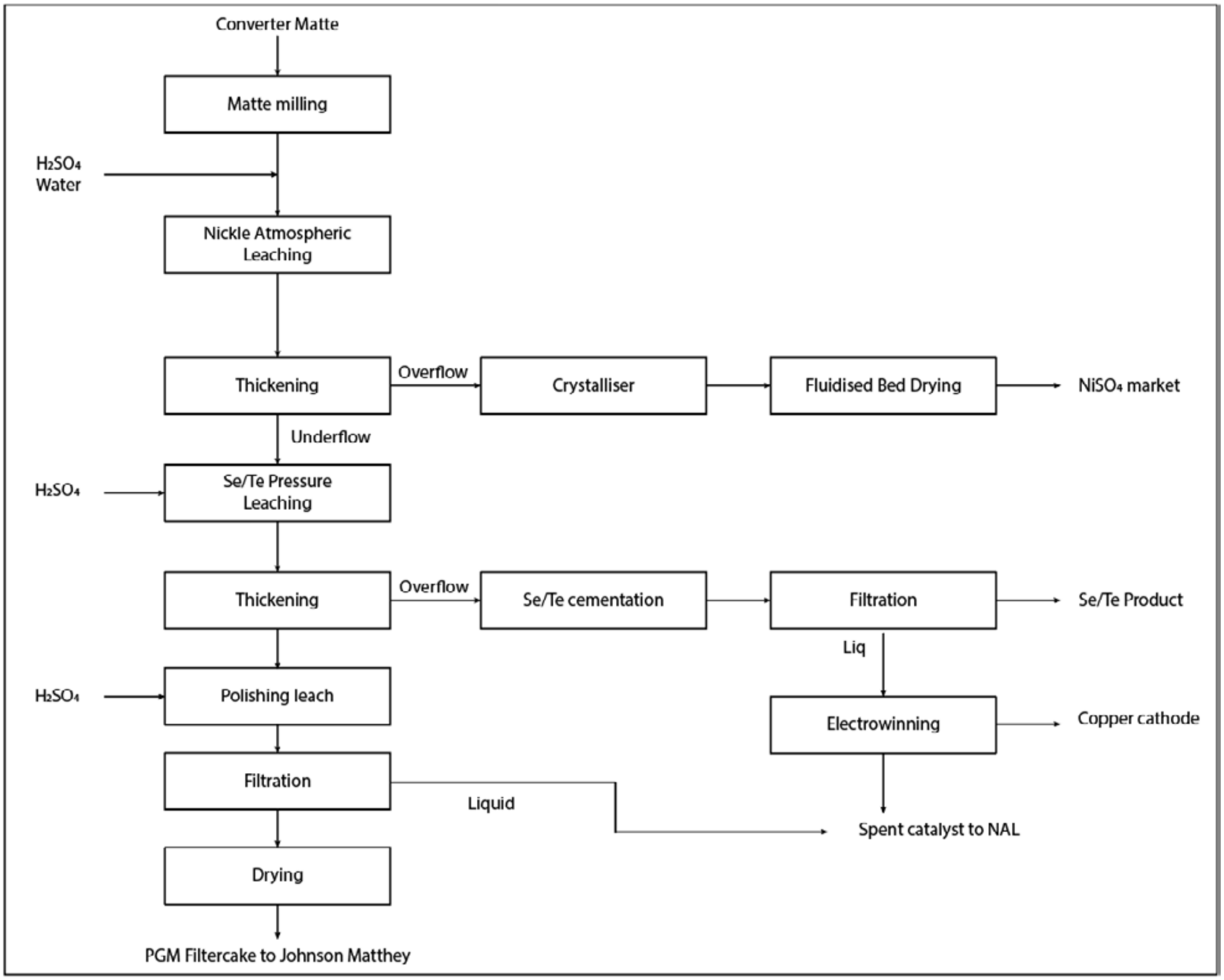

Figure 6. Block Flow Diagram of Stillwater Base Metal Refinery [29].

## 2.2 Lundin Eagle Mine

The Lundin Eagle Mine, located in the Upper Peninsula of Michigan, is the United States only domestic primary producing nickel mine [33]. The magmatic sulfide deposit was first discovered in 2002, and production first began in 2014. It produces nickel and copper concentrates via flotation, while also having minor amounts of cobalt and PGMs in the nickel concentrate that produce additional profits. The mine is expected to close in 2029 as it depletes its two main ore bodies, Eagle and Eagle East, but is looking to start producing from a third zone, Keel, which may extend its mine life [34]. They have also recently partnered with Talon Metals conduct exploration drilling at nearby properties in Michigan's Upper Peninsula, in order to try and find additional nickel and copper mineralization [35].

The ore body hosts high grade magmatic Ni-Cu sulfide mineralization in the form of pentlandite and chalcopyrite minerals. Their measured and indicated mineral resources of PGMs were around 0.38 g/t of platinum and 0.27 g/t of palladium [34]. While they process a higher grade currently (0.57 g/t of platinum and 0.4 g/t of palladium), it is expected to drop over the next few years as they process fewer valuable ore before mine closure, resulting in the averages above. Because of the significantly lower PGM grades and smaller deposit when compared to Sibanye-Stillwater, it is more economical to sell the nickel and copper flotation concentrates to a smelter than to refine it themselves. The Eagle mine sells to a smelter in Canada for further processing, and get paid for the copper, nickel, PGM, and cobalt in the concentrates.

All three ore bodies are underground mines that produce ore which is transported to their Humboldt Mill, where it undergoes a three-stage crushing process [34]. The Humboldt mill processes around 2,000 tpd and has the capabilities to crush, grind, and concentrate the ores. This circuit involves a primary jaw crusher and a secondary and tertiary cone crushing circuit to reduce the ore size down to 80% passing 8 mm [36]. Then, it is sent to two ball mills working in parallel that target an 80% passing 100 µm product. Sodium carbonate is then added for pH control further down the line. The feed is then sent to a rougher-cleaner-scavenger flotation circuit to separate the copper and nickel minerals from the rest of the gangue. They use SIPX, MIBC, soda ash, lime, CMC, and flocculant to achieve the best recovery and separation during this process. The copper-nickel rougher concentrate is reground before cleaning to remove as much gangue as possible. The bulk cleaner concentrate then goes through another round of flotation to separate the copper and nickel minerals through the addition of lime to increase the pH further. By increasing the pH to between 10.5 and 11.5, the nickel pentlandite particles are passivated and sink, while the copper floats. The resulting copper and nickel concentrates are then dewatered with thickeners and filter presses and sent off directly to a smelter facility. The platinum and palladium follow the nickel through this process and end up reporting to the nickel concentrate. This process achieves recovery of around 76% platinum and 84% palladium, resulting in 7,000 oz of platinum, and 5,500 oz of palladium produced each year [36]. The tailings produced are sent to a subaqueous tailings pond next to the Humboldt mill to prevent any acid mine drainage from occurring due to sulfur content of the pyrrhotite in the tailings. A flowsheet of the Humboldt mill process can be seen in Figure 7.

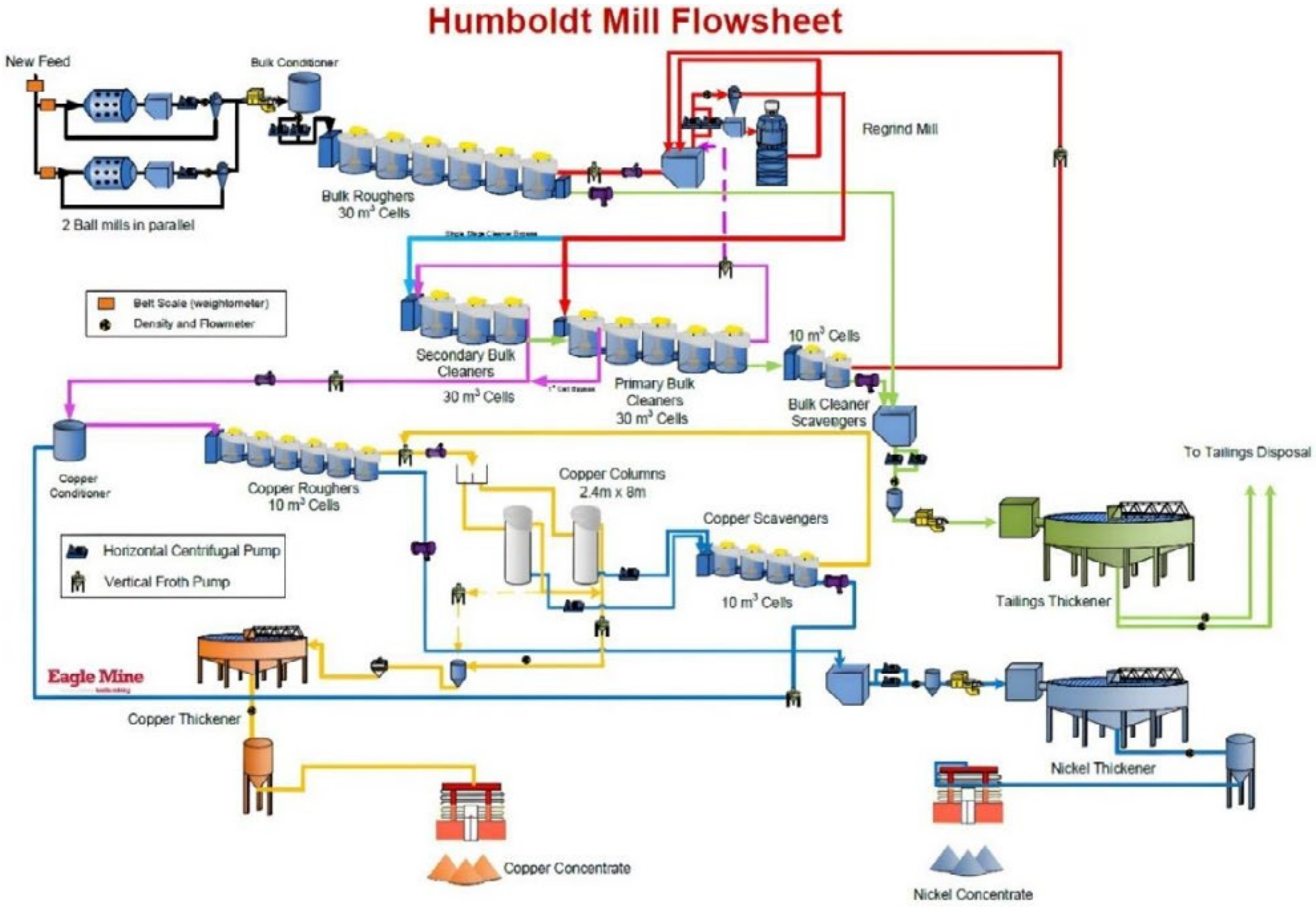

Figure 7. Simplified Humboldt Mill Flowsheet [36].

## 3. Current domestic recycling operations

Recycling of PGMs is a well-established industry, both domestically and globally, that is increasing in volume each year. The secondary supply of platinum, palladium, and rhodium is derived primarily from the recycling of spent automotive catalysts, waste electrical and electronic equipment, as well as jewelry. Recycled PGMs represent a significant share of the global supply, with these secondary sources playing a crucial role in bridging the gap between primary mine production and total global consumption. Catalytic converters account for about 60% of PGM use and, as such, are the dominant secondary sources with high PGM content and relatively few impurities [14], [37].

The U.S. is strong in collection, pre-processing, and primary recovery; while final PGM separations often occur at a smaller number of specialized refineries, often offshore, introducing supply-chain exposure. In 2024, approximately 120,000 kg of palladium and platinum were recovered worldwide from new and recycled scrap, including about 45,000 kg of palladium and 8,500 kg of platinum sourced from automobile catalytic converters in the United States [38].

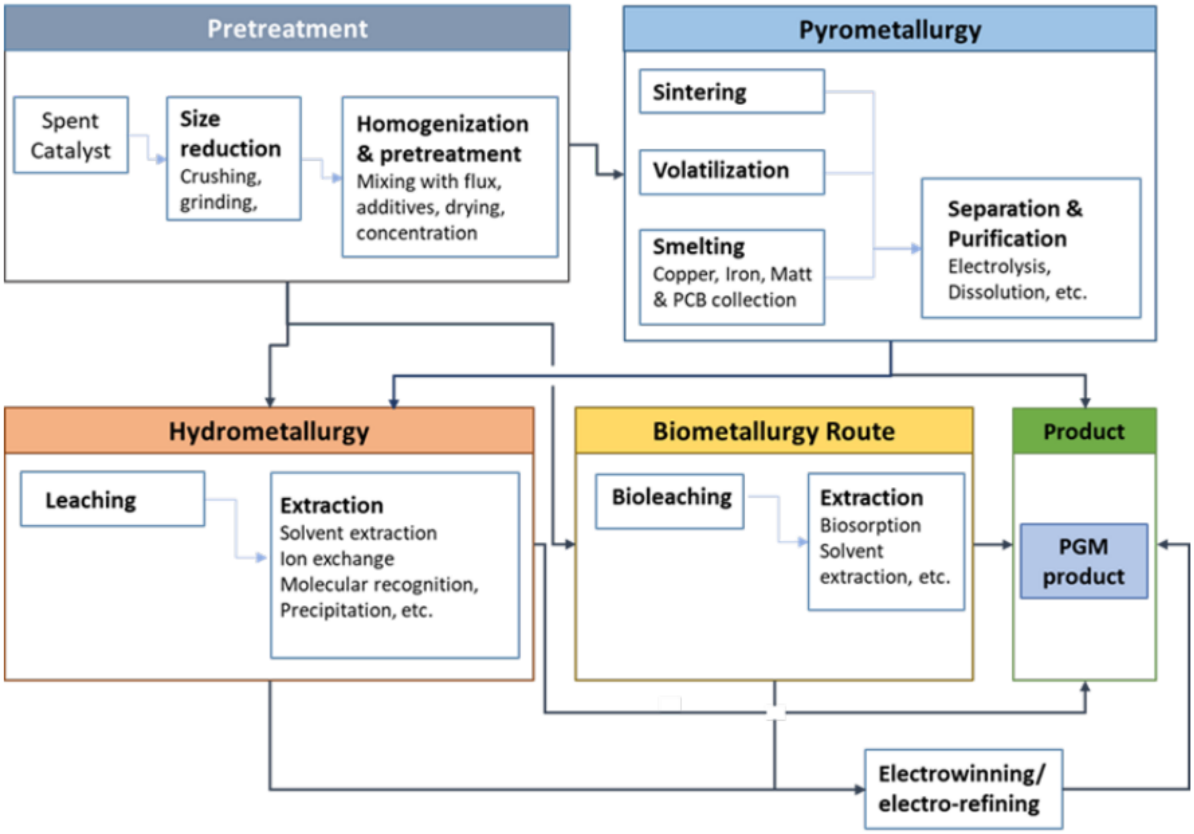

Figure 5: PGM catalyst recovery pathways [39]

The recovery of PGM materials typically involves two main stages: pre-processing and metal recovery. Pre-processing makes heterogeneous scrap safe, homogenous, assayable, and chemically ready for recovery. The goal at this stage is to remove oils, plastics, and loose tramp metal that could foul furnaces or leach circuits. Metal recovery involves conversion of pre-processed feedstock into PGM products by either pyrometallurgical or hydrometallurgical techniques [40], [41], [42]. Research in bio-metallurgical techniques such as bioleaching and biosorption methods is growing, but is currently limited to lab and pilot scale recovery, with potential to complement mainstream circuits over time [15], [41].

Thermal treatment enables the separation of PGMs and PGM-containing alloys from their base materials for subsequent refining [41]. Typically, the process involves melting the PGM-bearing substrate in the presence of a flux and a metallic collector. Pyrometallurgical methods have a high tolerance for heterogeneity/contaminants in the feedstock and are ideal for large scale operations with mixed feed. However, because these are high-temperature operations, they are inherently energy-intensive and can generate emissions.

Hydrometallurgical extraction operates at significantly lower temperatures and allows for selective metal by metal leaching. A typical hydrometallurgical process includes leaching, concentration, extraction, stripping, and recovery of the metals either in metallic form or as salts. Various leaching systems have been investigated for PGM recovery, including aqua regia ($HCl/HNO_3$), cyanide-based solutions ($CN^-$), mineral acids ($H_2SO_4$, HCl, $HNO_3$), chloride-based media ($NH_4Cl$, $CuCl_2$, $FeCl_3$), alkaline hydroxides (NaOH, $NH_4OH$), and carbonate systems ($Na_2CO_3$, $(NH_4)_2CO_3$) [41]. Overall, leaching performance depends on several factors, including reagent concentration, solid-to-liquid ratio, particle size, entrapment of soluble species within less soluble phases, pH, temperature, stirring intensity, and leaching duration [41], [43], [44].

Companies that recycle PGM catalysts in the United States include Sibanye-Stillwater, BASF, Umicore, Tenneco, Continental AG, APC Automotive Technologies, Metallix, Reldan, and others [15]. As noted in Section 2.1.3, the Columbus Metallurgical Complex is one of the world's largest recyclers of PGMs. This material primarily comes from automobile repair facilities and scrapyards that process end-of-life vehicles. In addition to automotive catalysts, PGMs are also recovered from spent petroleum and industrial catalysts, as well as refinery sweeps and cuttings [45]. Accurate sampling and analysis are very difficult, but critical under to ensure operational reliability and integrity [29].

Table 2. Summary of a few PGM recycling operations in the US

| Operator / site | State | Role in chain | Main feedstocks | Notable capabilities |
|---|---|---|---|---|
| Sibanye-Stillwater, Columbus | MT | Pre-processing, smelting | Auto catalysts | • Large U.S. auto catalyst recycler<br>• Third-party final refining [28]<br>• Recycled roughly 10 tons of 3E PGMs in 2024 [31] |

| | | | | |
|---|---|---|---|---|
| BASF ECMS, Seneca | SC, TX | Pre-processing, final refining | Auto catalysts | • Major North American PGM refinery<br>• Manufactures new industrial catalysts from spent auto catalysts [46] |
| Heraeus, Wartburg | TN | Pre-processing, sampling, smelting | Mixed PGM scrap | • New smelter added in 2023<br>• Renewable-powered [47] |
| Sabin Metal West, Williston | ND | Pre-processing, sampling | Industrial catalysts | • Pyro-based recovery<br>• Full-lot sampling [48] |
| PMR USA, Gastonia/Las Vegas | NC / NV | Pre-processor (de-can, mill, assay) | Auto catalysts | • High converter throughput [49]<br>• Specializes in homogenization and sampling |
| Metallix Refining Inc., Maxton/Greenville | NC | Pre-processing, sampling, refining | Ceramic and metallic auto catalysts | • High converter throughput facility<br>• Configured for metal-foil autocats<br>• Leaching/wet chemistry based recovery |

BASF Environmental Catalyst and Metal Solutions operates recycling facilities in the United States that recover and refine precious metals used in catalyst manufacturing. The Seneca site in South Carolina functions as a large-scale operation that recycles, concentrates, and refines PGMs essential for various catalytic applications [39], [42]. The Caldwell facility in Texas supports BASF's precious metal recovery network by recycling chemical catalysts and processing industrial scrap. Recovered materials from the Caldwell site are directed to downstream smelting operations. The overall process employed across these sites combines smelting, chemical leaching, and refining to recover platinum, palladium, rhodium, and other associated metals [50], [51].

Heraeus Precious Metals is a major company in the precious metals industry, producing materials and components used across sectors such as automotive, chemical, semiconductor, pharmaceutical, hydrogen, and jewelry manufacturing. In North America, Heraeus operates recycling and refining facilities in Santa Fe Springs, California, and Wartburg, Tennessee; the latter was expanded in 2022 with the installation of a new smelter. Heraeus is among the largest refiners of PGMs and has extensive experience in recycling components from electrolyzer and fuel cell systems, such as including membrane electrode assemblies and catalyst-coated membranes.

The company applies pyrometallurgical and hydrometallurgical techniques to recover platinum, palladium, iridium, and ruthenium from secondary sources, continuously improving its processes to enhance recovery efficiency and meet the growing demand for recycled PGMs [47], [52], [53].

Another important company in the global PGM recycling sector is Johnson Matthey, which operates multiple facilities worldwide. In the United States, its principal recycling and refining operation is in West Deptford, New Jersey. This site not only refines PGM concentrates but also processes a broad spectrum of secondary materials, including automotive and industrial catalysts, chemical catalysts, dental and electronic scrap, fuel cell components, and jewelry residues. In other words, Johnson Matthey's recycling activities cover a wide range of industries and material streams [54]. The refining workflow includes four major stages, combining pyrometallurgical and hydrometallurgical techniques.

Figure 8. Simplified workflow for PGM recycling and refining operations [55]

The first step involves determining the precious metal content and estimating the required processing time. Johnson Matthey employs highly accurate and long-established assay methods, supported by advanced sampling protocols and diverse analytical techniques tailored to each material type. Chemists evaluate the full matrix of the feed, both precious and non-precious elements to identify the most efficient and economical refining pathway.

Once terms are agreed, compatible feeds are blended and sent to large reverberatory furnaces, where they are melted at temperatures above 1200 °C for roughly 12 hours. The smelting step separates non-metallic components and produces two bullion types:

- a silver-rich bullion containing mainly platinum and palladium, and
- an iron-rich bullion containing all five PGMs.

Both bullion streams then undergo acid leaching to dissolve and concentrate the precious metals. The silver-based bullion is treated with concentrated acid to dissolve silver, platinum, and palladium while separating any gold for further refining. The iron-based bullion follows a different chemical route due to the distinct solubilities and chemical behaviors of the PGMs it contains. The dissolved PGMs are then subjected to a multistage chemical separation process, involving solvent extraction, precipitation, filtration, and evaporation steps. These operations isolate each metal and convert them into high-purity salts or sponge. Subsequent heat treatment produces the final precious-metal products, suitable for reuse in catalytic, chemical, and industrial applications.

## 4. Future Domestic PGM Operations:

Domestically, there are plans for starting several PGM producing primary operations in Minnesota, along with the potential for some in Alaska. The facilities of interest in Minnesota are the Twin Metals Minnesota operation, The NewRange Copper Nickel operation, and Talon Metals' Tamarack Project. These operations can provide a future source of domestically produced PGMs, which can also decrease the reliance of overall domestic reduction on the price of palladium. Since these projects will all produce PGMs as byproducts, as opposed to primary products like Sibanye-Stillwater, their overall profitability and production will not be tied to the price of PGMs.

### 4.1 Twin Metals Minnesota

The origin of the Twin Metals Minnesota (TMM) operation began in 2006 at the start of a hydrogeological study for the Duluth Metals Limited project [56]. Following their partnership with Antofagasta PLC in 2010, TMM was started, which plans to develop an underground copper, nickel, platinum, palladium, gold and silver mine in the Iron Range region of northeast Minnesota [57]. The deposit is composed primarily of plagioclase, olivine, pyroxenes, and oxides, which make up 85% of the total minerology. The sulfide content ranges from 1 to 6%, and is comprised primarily of chalcopyrite, cubanite, pentlandite, and pyrrhotite. The platinum within the deposit does not deport as a constituent to any of the sulfides and occurs only in discrete Pt minerals. The palladium is similar, but with around 20% of the Pd deporting in the pentlandite.

The TMM project estimates a 30-year mine life, with an averaged processing rate of 20,000-50,000 tpd [57]. Overall, they are expecting a necessary initial capital expenditure of \$2.77 billion, with \$5.41 billion capital expenditure over the 30-year lifespan [58]. TMM estimates a total production of 180 million tons of ore over the 25-year production window. Broken down annually and by concentrate, this is 174,000 tons of copper concentrate, 84,000 tons of nickel concentrate, and 550 tons of gravity concentrate, which will include the PGMs. In their pre-feasibility study, they estimate a life-of-mine (LOM) production of 46,500 kg of platinum and 125,000 kg of palladium [59]. While the main products will be nickel and copper, and significantly more work has been done on the sulfide flotation aspect of the project, the PGMs are still a valuable component and can produce around 10 years' worth of domestic production [60] [61].

The mining process starts with deep underground drilling, where the ore is accessed through a system of draw points and drifts, and is initially broken up and collected in an orepass [58]. The deposit would be mined using a longitudinal long hole retreat mining method, so less excavation is required and the need for above ground waste rock stockpiles is eliminated. The ore would be processed through comminution, gravity concentration, flotation, and concentrate dewatering. Ore will first be sent to a SAG mill, before going to a ball mil that will grind the ore to an 80% passing 135 μm, before finally being pumped to the flotation circuit. A portion of the material is recirculated to be ground further, and a portion is split and fed to the gravity concentration circuit before returning to the ball mill circuit.

The concentrator would produce copper, nickel, and gravity concentrates, as well as a tailings, which will be sent to a tailings dewatering plant. 50% of the tailings will produce tailings backfill for the mine, with the rest producing a tailings filter cake for a dry stack facility, which will provide permanent ground storage tailings.

In order to recover the PGMs, they plan to use the gravity concentration circuit to recover up to 65% of the precious metals with low mass pull products [59]. As PGMs and precious metals have a higher specific gravity than most other metals, they can be separated out using gravity separation. In the mine plan of operations, the description of the gravity concentration circuit is sparse, but details how a portion of the ball mill recirculating load would be split off to feed multiple gravity concentration units operating in parallel, which would then recover the gold, platinum, and palladium. This concentrate would be dewatered and then sent in batches to a gravity concentration holding tank, and then shipped off to sell [62]. A depiction of their initial milling flowsheet can be seen in Figure 8, which was made in 2014. This does not include the gravity concentration stage, as it was included in the mine plan later.

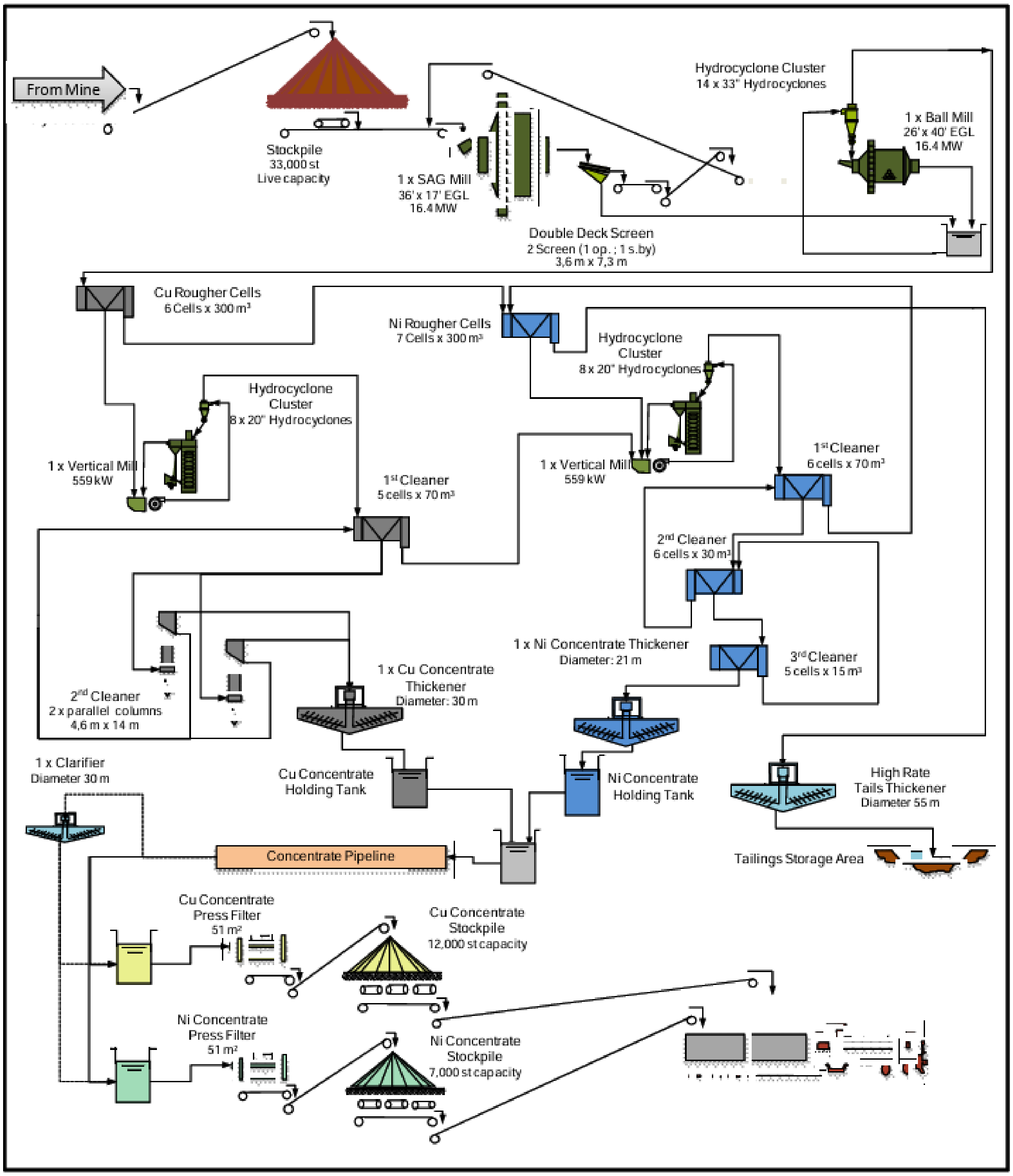

Figure 9. TMM Preliminary Mill Flowsheet [59].

## 4.2. New Range Copper Nickel Operation

Another potential source of domestic PGMs is the New Range Copper Nickel project, which is a joint venture between PolyMet mining and Teck Resources [63]. After the merger in 2023, the venture now controls over half of the reserves of the Duluth Complex, which is the largest known untapped deposit of copper and nickel in the world [64]. While this deposit is nearly entirely copper, nickel, and cobalt, New Range does have access to significant amounts of palladium, along with some platinum and gold. The deposit has a platinum grade of 0.067 g/t, and a palladium grade of 0.243 g/t. The deposit is considered a magmatic deposit with disseminated accumulations of sulfides within a dominant silicate framework.

Their initial mine plan includes a 32,000 tpd open pit mining operation [65]. With a 20-year permitted life, they plan to produce around 2,000 kpy of palladium and 500kpy of platinum contained in a nickel concentrate or PGM concentrate, depending on the project phase [66]. They plan to retrofit existing crushing and milling infrastructure at brownfield sites to do the initial processing of the ore [65].

There are two phases to the New Range beneficiation plan. Phase 1 consists of crushing, grinding, flotation, concentrate thickening, and concentrate filtration [67]. Phase 1 will produce the concentrates that can be sold to the market. In Phase 2 in mining year 2, they plan to commission a hydrometallurgical processing plant to produce nickel sulfide and pyrrhotite concentrates. The concentrates will be processed through a single autoclave to produce a copper concentrate, nickel-cobalt hydroxide, and PGM and precious metal precipitate. This involves a HPAL process and solution purification to extract and isolate the PGMs. While this process has been widely discussed and is currently included in the mine plans, there is ongoing discussion on the feasibility and implementation of Phase 2.

In Phase 1, the primary comminution and flotation circuits begin operation, with the goal of producing marketable concentrates of copper, nickel, cobalt and PGMs. For the comminution, they plan to use a combination of gyratory crushers, a pebble crusher, a SAG mill, a ball mill, and 3 stirred mills [67]. They plan to use the existing primary crusher, with a new downstream crushing circuit. The milling intends to produce a product 80% passing 120 μm, which is mixed with process water during the SAG milling to be processed further. This consists of three main sections: bulk Cu-Ni circuit, Cu-Ni concentrate separation circuit, and a pyrrhotite (Po) circuit. After the material passes through these circuits, they are dewatered by thickening and filtration to achieve a cake moisture of 12.1%, with the process water being recycled in the plant. After this, the concentrates can be sold, or later in the mine life, the nickel and pyrrhotite concentrates can be treated via Phase 2 [67]. An image of the Phase 1 process can be seen in Figure 9.

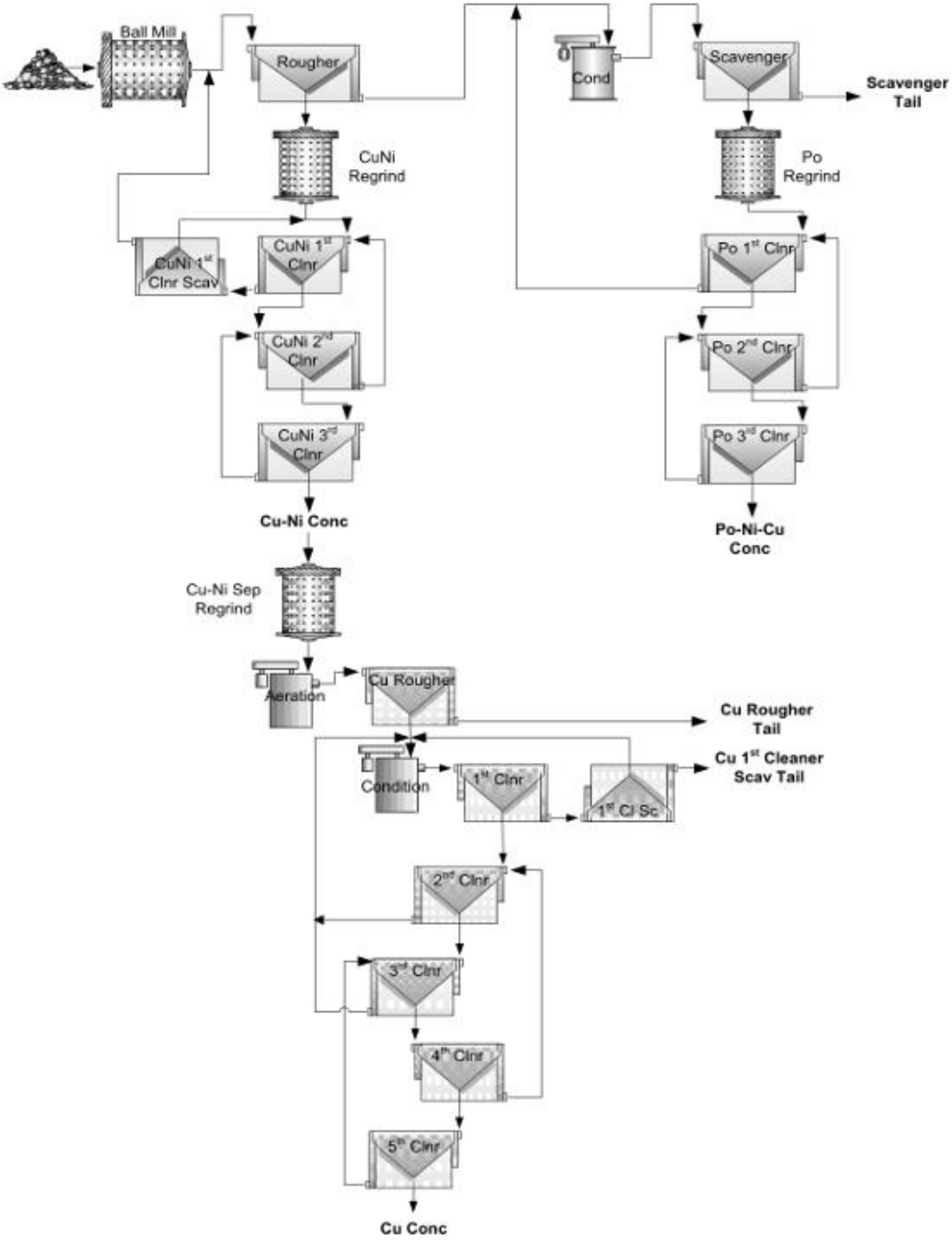

Figure 10. New Range Phase 1 Process Block Flow Diagram [51].

In Phase 2, the material will be processed hydrometallurgically using an autoclave at 1,000 tpd. The leach will operate in an oxygen rich environment with added chloride ions and around 440°F and a pressure of 504 psi, to ensure complete oxidation of the sulfides. An oxygen overpressure of 100 psi will be maintained. Hydrochloric acid will be added to maintain steady chloride concentration, which is necessary for leaching of the PGMs. The autoclave will oxidize the sulfide minerals, making it into sulfates and liberating the entrained PGMs, which then form soluble chloride complexes. The 9.2% solid slurry is discharged, cooled using heat exchangers, and sent to a thickener in which the PGMs reside in the thickener overflow. The PGM precipitation circuit includes a preheater, two PGM precipitation tanks and an $SO_2$ reduction tank. The $SO_2$ reduction tank is used to reduce the ferric ions, before sending the slurry to the other tanks. To precipitate the PGMs, CuS is added, as CuS is less noble than the PGM sulfides, so the PGMs in solution will precipitate in exchange for Cu solubilizing. The precipitation reactions are shown in Equations 8 and 9 [67].

$$PdCl_4^{2-} + CuS = PdS + Cu^{2+} + 4Cl^- \qquad [15]$$

$$PtCl_6^{2-} + 2CuS = PtS + 2Cu^{2+} + 6Cl^- + S \qquad [16]$$

Through initial studies, a recovery rate of 78.6% for palladium and 74.5% for platinum can be achieved. A simplified flowsheet of the hydrometallurgical process can be seen in Figure 10.

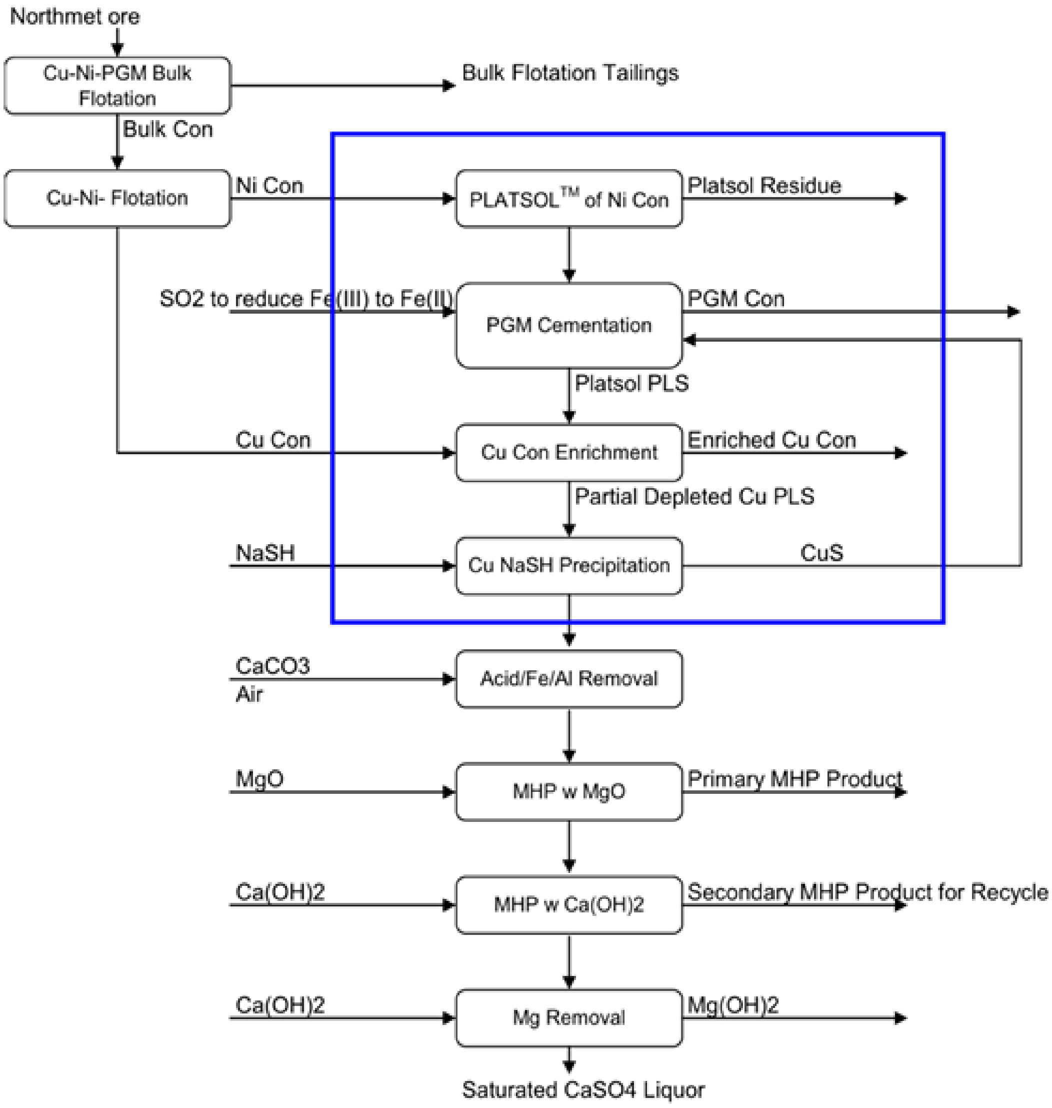

Figure 11. New Range Project Simplified Hydrometallurgical Process [67]. The blue zone contains the PGM hydrometallurgical steps.

### 4.3. Tamarack Talon Metals Project:

Talon Metals, in collaboration with Kennecott, have a proposed underground nickel sulfide mine near Tamarack, Minnesota [68]. With a projected 7 to 10-year production plan, there is an estimated 8.2 million tons of ore that can be processed. Construction is planned to begin in 2027 at the earliest, with a 2-year construction period. They have an indicated grade of 0.34 g/t of platinum and 0.21 g/t of palladium, with some sections having up to 3.65 g/t of PGMs, and is counted as a byproduct [69]. The ore deposit is considered an ultramafic intrusive complex that has nickel, copper, and cobalt sulfide mineralization, with pockets of platinum and palladium entrained in the sulfides [70].

The mine is planned to be an underground mine that is accessed via a vertical shaft. Once the ore is mined and hoisted to the surface, it will be processed at an average daily mill feed rate of 2,000 tpd [71]. While the mine site will be in Minnesota, Talon Metals has decided to locate their mill, the Beulah Minerals Processing Facility, in North Dokota. The mineral processing facility will consist of comminution, flotation, and dewatering circuits. The primary crushing involves jaw crushing, followed by secondary cone crushing and ball mill grinding. The product that, which is 80% passing 100 µm, will then be sent to the flotation circuit. This is a typical rougher-cleaner-scavenger flotation circuit to produce copper and nickel concentrates. They plan to use sodium isopropyl xanthate (SIPX) as the sulfide collector, and methyl isobutyl carbinol (MIBC) as the frother. The bulk flotation is done in five tank cells of 30 $m^3$ each. For the scavenger flotation, potassium amyl xanthate (PAX) is used in three tank cells to recover the remaining sulfide material. The rougher concentrate is sent to a regrind stage to reduce the particle size to 80% passing 60 µm. They plan to increase the pH to 10 with the addition of lime, and using CMC as a depressant, thereby increasing the selectivity of the flotation. The concentrates are then thickened and filtered separately and sent to smelters for further processing. The tailings from the flotation circuit are planned to be used for paste backfill in the mine.

As the nickel and copper content of the deposit constitutes a large majority of the potential profits of the Tamarack operation, there is no additional processing that will be done for further extraction of the PGMs. Instead, the PGMs are expected to report to the nickel concentrate stream and will be sold along with the nickel concentrate to the smelter for credit. While this is the current planned method, they are also considering a further hydrometallurgical processing facility, which could process lower grade nickel concentrate [71]. This would involve a pressure oxidation leach, followed by leach solution neutralization and impurity removal, copper recovery, nickel and cobalt solvent extraction, and final scrubbing and stripping stages. This would alter how any PGMs are recovered from the Tamarack operation, but there is currently no test work showing the feasibility of this stage. A depiction of the general flowsheet of the Tamarack project can be found in Figure 11.

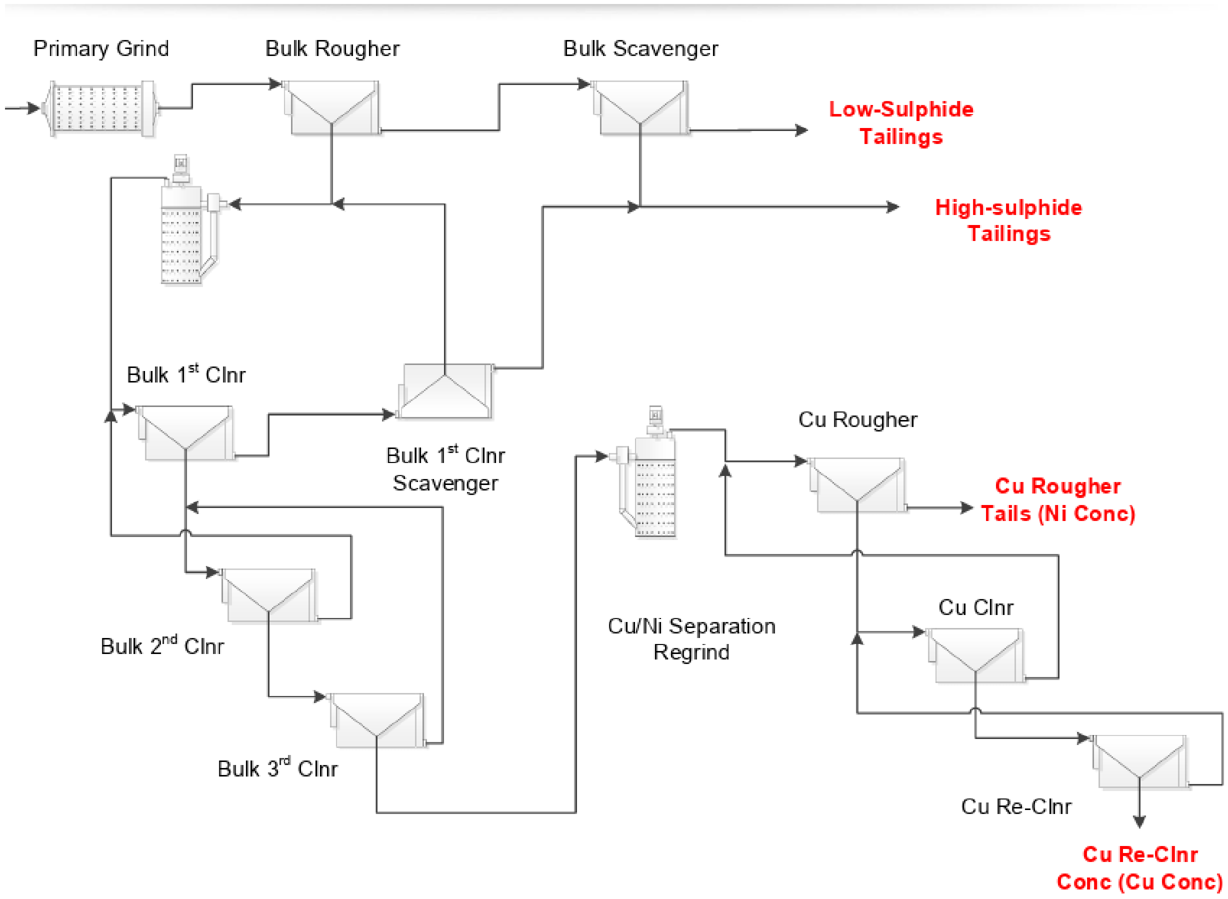

Figure 12. Simplified Tamarack Flowsheet [71].

Overall, with the Twin Metals Minnesota, NewRange, and Tamarack projects, the domestic primary production of PGMs has the potential to increase by 65%, as shown in Figure 12. This could help decrease the net import reliance on PGMs and establish a more robust domestic supply chain. In addition, due to the byproduct nature of the PGMs in these operations, any future volatility in the price of palladium or platinum will have significantly reduced effects on domestic production. A summary of the estimated average yearly production of PGMs and the grades of these deposits is shown in Table 3.

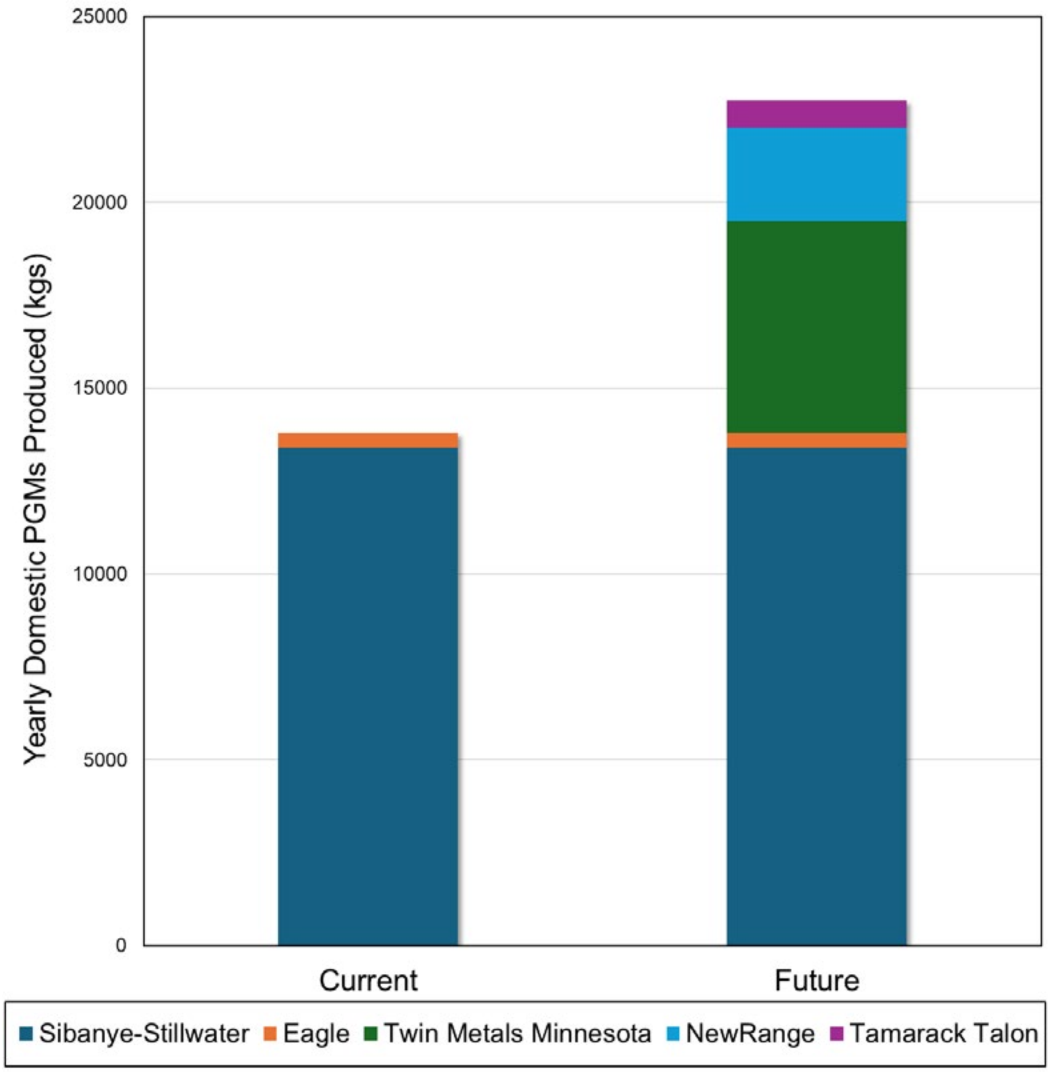

Figure 13. Estimated Average Yearly Production of Domestic PGM Projects, Comparing Current and Future Projects.

Table 3 - Summary of US PGM production.

| Operation | Estimated Average Yearly PGM Production (kg) | PGM Indicated and Measured Grade (ppm) |
|---|---|---|
| U.S. Yearly Production | 13800 | n/a |
| Stillwater | 13400 | 13.16 |
| Eagle Mine | 400 | 0.65 |
| Twin Metals | 5700 | 0.48 |
| NewRange | 2500 | 0.30 |
| Tamarack | 750 | 0.68 |

## 5. Promising domestic deposits

While the amount of active PGM producing mines and the number of opening projects is limited, there are promising deposits being investigated across the United States.

### 5.1. Nikolai Project

The Nikolai project is a polymetallic exploration initiative spearheaded by Alaska Energy Metals Corporation, which looks to recover nickel, copper, cobalt, platinum, palladium, and gold [72]. First investigated in 1995 for Inco, the deposit was found to be like the Norilisk area of Russia and has the potential to provide a key source of domestic critical minerals. With metal prices and mining technology at the time, the deposit was thought to be unprofitable, but with lower grade orebodies around the world, there has been a renewal of interest in the project [73]. Pure Nickel Inc. completed preliminary drilling between 2007 to 2014 across the length of the Eureka zone of the deposit, where they found a thick zone of disseminated sulfide mineralization with consistent nickel, copper, cobalt, and PGM grades. They found an average platinum grade of around 0.048 g/t and an average palladium grade of around 0.094 g/t. Alaska Energy performed more drilling in 2023 to confirm similar grades. Overall, they confirmed a indicated resource of 1.22 million oz of palladium and platinum the Eureka zone [74]. While the project is still new and no progress has been made to develop the deposit into a mining operation yet, the drilling shows promise for the project [75].

### 5.2. Duke Island Project

Duke Island is another Alaskan deposit that has promise to be a domestic source of copper, nickel, and PGMs [76]. It is an ultramafic intrusion that has sulfide mineralization indications, and the surface rock samples indicate concentrations up to 1 g/t of PGMs. While there has been preliminary drilling in this area, it has not been deep enough to evaluate the potentially PGM enriched sulfide zone. Sixteen drill holes were done in only one of the zones present in the project zone. It has been owned by Stillwater Critical Minerals, but in 2024 it was announced that they had entered into a letter of intent to sell the Duke Island project to Granite Creek Copper [77]. While this project is in the preliminary stages of exploration and development, it has gained interest as a potential source of PGMs in Alaska.

### 5.3. Other deposits

There are numerous other PGM deposits across the United States that are technically feasible to recover but have not been pursued thus far. Many of these are located in Alaska, such as the Union Bay Project, Last Chance, Brady Glacier, Genesis, Misheguk Mountain, Valdez Creek Mining District, and more [23], [78], [79], [80], [81]. These investigations show potential for increasing critical mineral independence for the United States in the world of PGMs.

## 6. Conclusion

This review shows that although the United States is a limited primary producer of platinum-group metals relative to global supply, domestic operations remain a critical component of national mineral security. In 2024, U.S. mines produced approximately 8,000 kg of palladium and 2,000 kg of platinum only a small fraction of global output leaving the country heavily dependent on imports.

Sibanye-Stillwater US operations continue to be the foundation of U.S. primary PGM production, supported by well-established smelting and refining facilities at the Columbus Metallurgical Complex. Additional PGMs are recovered as by products from the Lundin Eagle Mine, and several emerging projects in Minnesota and Alaska, including Twin Metals, NewRange Copper Nickel, and the Tamarack project, could increase domestic by product PGM production by as much as 65% if developed as planned. These projects highlight a promising pathway to diversify and strengthen the U.S. PGM supply chain.

Recycling has become the most significant contributor to the domestic PGM supply, recovering roughly 45,000 kg of palladium and 8,500 kg of platinum from spent automotive catalysts far exceeding primary mine output. The sector is supported by a mature network of companies, such as Sibanye-Stillwater, BASF, Heraeus, and Johnson Matthey, which operate advanced pyrometallurgical and hydrometallurgical processes capable of treating increasingly complex secondary materials. Expanding domestic refining capacity and reducing offshore dependence in the final separation steps represent important opportunities to improve supply-chain resilience.

Although new mining projects and robust recycling capabilities offer meaningful potential to improve domestic availability, the United States still faces challenges related to processing capacity, permitting timelines, and reliance on foreign refining for certain high purity PGM products. Addressing these issues will be essential for achieving a more secure and competitive supply chain.

Overall, strengthening PGM supply security in the United States will require coordinated progress across primary production, recycling, refining, and policy development. Strategic investments in permitting, domestic refining infrastructure, advanced recovery technologies, and integrated supply-chain planning would significantly reduce vulnerability to global market disruptions while leveraging the country's existing strengths in recycling and metallurgical innovation. As the energy transition accelerates and industrial applications continue to evolve, a resilient domestic PGM ecosystem will be essential for national security, economic stability, and long-term technological leadership.

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