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NI 43-101 Technical Report
Ochoa Project Feasibility Study
Lea County, New Mexico, USA

Effective Date: January 9, 2014
Report Date: March 7, 2014

Prepared for

ICPOTASHCORP

Compiled By

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NATIONAL INSTRUMENT 43-101 TECHNICAL REPORT
OCHOA PROJECT FEASIBILITY STUDY
LEA COUNTY, NEW MEXICO
USA

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ITEM 1: SUMMARY

1.1 Location, Access, and Infrastructure

IC Potash Corp. (Company) through its wholly owned subsidiary Intercontinental Potash Corp. (USA) has completed a Feasibility Study (FS) on its Ochoa Mine Project property (Property). ICP refers to the Company and/or all affiliates, and is used interchangeably. The Property is located in Lea and Eddy Counties, New Mexico.

The Ochoa Mine Project (Project), a greenfield underground polyhalite mine and surface processing facility development, is designed to mine approximately 3.7 million tons per year (Mtpy) of polyhalite and process it into approximately 714,400 tons per year (tpy) of Sulfate of Potash (SOP) fertilizer product. The main mine shaft, slope, and processing plant facilities are located in Lea County, New Mexico, approximately 60 miles east of Carlsbad, New Mexico, and approximately 70 miles south-southwest of Hobbs, New Mexico.

Adequate interests in the Property have been obtained to support the planned mining and processing operations on the Property. Sources for water, electric power, and natural gas are accessible locally. Access to the Property is attained via State Highways (SH) 128 and 18. The Texas-New Mexico Railroad (TNMR) is located approximately 24 miles east of the mine and processing plant sites, north of the town of Jal, New Mexico. A product transloading facility will be constructed near Jal. SOP will be trucked from the processing plant to the Jal rail loadout.

Permitting for the Project is well underway with a Final Environmental Impact Statement (FEIS) Record of Decision (ROD) expected by April 2014.

1.2 Tenure and Surface Rights

The Property encompasses 89,787 acres, more or less, with 28 United States (US or USA) Department of Interior (DOI) prospecting permits administrated by the DOI's Bureau of Land Management (BLM), and 18 New Mexico State Land Office (NMSLO) mining leases. The BLM has offered the Company an additional seven prospecting permits totaling about 12,484 acres. Options for necessary surface leases and rights-of-way (ROWs) have been acquired. ICP has been advised by New Mexico's State Engineer's office that it has the right to withdraw sufficient water from the Capitan Reef aquifer to support the Project. Reasonable prospects exist for ICP to obtain the required permits and approvals to conduct mining and processing operations on the Property.

1.3 Geology, Geochemistry, and Hydrogeology

1.3.1 Geology

The Property lies at the northeastern margin of the Delaware Basin. The Delaware Basin is a structural sub-basin of the larger Permian Basin that dominated the region of southeast New Mexico, west Texas, and northern Mexico from 265 to 230 million years before present (MYBP). The Permian Basin is an asymmetrical depression formed on top of the Precambrian basement rocks. Marine sediments accumulated in the Basin throughout the Paleozoic era. The slow collision of the North American and South American crustal plates resulted in tectonic subdivision of the Permian Basin into numerous sub-basins, of which the

Delaware and Midland Basins are the largest (Ward, Kendall, and Harris 1986). The Delaware Basin has been extensively studied, in part because of extensive gas and oil exploration, but also because of the Waste Isolation Pilot Project (WIPP) in the northern part of the Basin. WIPP is a geologic repository to permanently dispose of radioactive waste, and has been the subject of extensive study.

The sedimentary sequence of the Delaware Basin is composed of deep water siliciclastics, shelf carbonates, marginal marine evaporites, and terrestrial red beds. The deep water siliciclastics and shelf carbonates occur well below the horizon of interest. Extensive and thick evaporite deposits occur throughout the late-Permian period (Ochoan-age) rocks within the Basin. The Upper Permian Series consists of Ochoan-age sedimentary deposits, specifically the Castile, Salado, and Rustler Formations.

The primary lithologic units present within the Property are, in order from oldest to youngest, the Castile Formation, the Salado Formation, the Rustler Formation, of which the Tamarisk Member is the location of the Ochoa polyhalite bed of interest, and the Dewey Lake Formation.

The Tamarisk Member is composed of three sub-units: a lower basal anhydrite, a middle halitic mudstone, and an upper anhydrite. The Ochoa polyhalite occurs within the basal anhydrite.

Potash is a general term for a potassium-bearing, chemical sedimentary mineral deposit that is the result of low-temperature chemical processes governed by evaporative concentration of a fluid such as seawater or freshwater. Bedded potash deposits commonly occur in sedimentary basins that have restricted connection to more dilute fluid.

Polyhalite is a hydrated sulfate of potassium, calcium, and magnesium ($K_2Ca_2Mg(SO_4)_4 \cdot 2H_2O$). Polyhalite may be white, light or medium gray, salmon colored to orange to brown, or reddish. When pure, it has 15.6 percent (%) potassium oxide (K_2O), 6.6% magnesium oxide (MgO), 18.6% calcium oxide (CaO), and 53.2% sulfur trioxide (SO_3) with 6.0% water (H_2O). It is usually finely to medium crystalline, massive, and compact.

1.3.2 Hydrogeology

Extensive hydrogeologic analysis was conducted to develop a source of water for the Project. The Capitan Reef Complex aquifer was tested by drilling two wells, and modeling was undertaken to demonstrate that the aquifer could supply the Project with an adequate quantity of water without detrimental effects on the Pecos River Basin. The wells are located about 18 miles east of the Project site. The Capitan aquifer is saline and this water will need to be treated for certain uses in the Project's facilities.

Multi-stage groundwater packer testing was conducted on drill hole ICP-092 to determine potential groundwater flows near the shaft and slope sites. Limited aquifers exist in the sedimentary rocks above the salts and steady-state groundwater flows are expected to be less than 50 gallons per minute (gpm) for all aquifers combined.

1.4 History

The Property does not have any mining history. The Delaware Basin has been explored for hydrocarbons since the early 20th century, but it has not been previously explored for polyhalite. ICP's planned commercial mining and processing operation to produce SOP and potentially other potassium/magnesium fertilizers is based on work that was performed in the 1920s and 1930s by the United States Bureau of Mines (USBM) and Potash Corporation of America (PCA) in the 1950s. The large-scale development of economic production of potash from potassium chloride and langbeinite in the Carlsbad, New Mexico, area significantly reduced interest in the use of polyhalite to produce potassium-based fertilizers. ICP began preliminary polyhalite exploration in 2008 when they applied for exploration permits and initiated a scoping study. That study was prepared by Micon International Limited (Micon) (2008, 2009) and it indicated that the Property had good potential for a sizeable polyhalite deposit.

In the 1930s and 1940s, the USBM (1930a and b, 1933, 1944) was tasked with performing scientific and engineering research regarding polyhalite processing to produce SOP. PCA conducted pilot plant testing in the 1950s. This work formed the basis of the process that ICP has developed for commercialization.

ICP validated the USBM and PCA results during the Ochoa Project Preliminary Feasibility Study (PFS) (Gustavson Associates [Gustavson] 2011d) and FS (SNCL 2014) via process testing, verifying, and validating the earlier work, while collecting data regarding equipment design for processing the Ochoa polyhalite.

1.5 Exploration

ICP successfully drilled, cored, logged, and plugged and abandoned 32 vertical exploration holes throughout the permit area during a three-phase exploration drilling campaign. Data from an additional 855 petroleum wells were used to establish regional correlations. Geophysical logs were run in all exploration drill holes. Core recovery in the polyhalite and anhydrite zones was excellent in terms of length and minimal alteration of the rock by the salt-based drilling fluid. Site visits were made by Qualified Persons (QPs) during the exploration program.

Phase 1 consisted of 6 holes, Phase 2, 7 holes, and Phase 2B, 7 holes. Phase 3A began in August of 2012 and completed 12 holes, 11 of which were in the main resource area. Early phases of drilling recovered smaller diameter core (3 inches). The need for bulk samples for metallurgical testing drove the acquisition of 6-inch-diameter core for most of Phase 3A. Stewart Brothers Drilling Company of Milan, New Mexico, drilled all 32 exploration holes. Each drill hole was drilled in two sections. The upper portion of each hole, from ground surface to within 50 to 75 feet (ft) of the top of the polyhalite, was drilled using rotary drilling techniques. The lower portion of each hole was cored in order to obtain samples for grade and engineering analyses.

Core recovery in the polyhalite and anhydrite zones was excellent in terms of length and minimal alteration of the rock by the salt-based drilling fluid.

A combined total of over 70,000 ft have been drilled in the 32 holes, of which approximately 3,528 ft were cored. During the Phase 3A program, a number of sidetrack cores were drilled to get more polyhalite material for metallurgical testing.

Based on a review of the exploration program, the QPs are confident that the exploration dataset meets the criteria for resource estimation use under Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards on Mineral Resources and Mineral Reserves (CIMDS) and National Instrument (NI) 43-101. ICP's quality assurance/quality control

(QA/QC) program is designed with aggressive duplication and insertion. Procedures are well documented and have been followed accordingly.

1.6 Metallurgical and Processing

1.6.1 Process Background and Testing

Process design began by confirming that prior research performed by both the USBM and PCA could be duplicated and was thoroughly understood. Test work during the FS further defined the details of the process including crushing and washing tests, calcination tests, development and testing of a bench-scale counter-current leach circuit, and a detailed comparison of different crystallization circuit options. Pilot plant test work was performed to confirm that the processes developed during bench-scale testing were technically viable on a continuous basis.

A large-scale pilot plant test was conducted with both gray and Ochoa ore. The test included process steps through crystallization with actual production of SOP and leonite crystals from Ochoa ore. Marketable grade SOP was produced.

Tests were conducted to determine granulation parameters and potential binders for SOP granulation. Further testing will determine parameters for selecting the type of binder for plant operation and the optimal amount necessary to obtain market-grade granulated SOP.

1.6.2 Design Criteria

The processing plant is based on design parameters of a run-of-mine (ROM) feed rate of 3.69 Mtpy at an average grade of 80% polyhalite; a plant utilization rate of 90.92% (7,912 hours per year); an overall recovery rate of 82.28%; and a SOP production rate of 714,400 tpy at a minimum level of 50% potassium sulfate (K_2SO_4). The drying and granulation circuit is designed to allow flexibility in production for each of the three SOP products. Soluble-grade SOP can vary between 0 and 100,000 tpy, granular product can fluctuate between 250,000 and 385,000 tpy, and standard grade product can fluctuate between 250,000 and 503,000 tpy.

1.6.3 Process Description

ROM ore will be conveyed from the mine to the plant via a series of belt conveyors. The ore will be sent to a roll crusher that discharges to a pulping tank where recycled water will be added to produce a slurry. The slurry passes to wet screening with oversize material sent to a Cage-Paktor for reduction and recycling. Salt is then removed from the ore. Salt dissolution begins in the wet portion of the crushing circuit. Additional dissolution occurs in a separate salt leach tank that provides complete dissolution of the salt particles.

The polyhalite ore must be heated to result in leachable ore. Fluid-bed thermal processing units have been selected as the preferred equipment for calcining the ore because they allow for excellent control of temperature and residence time, which are the main factors controlling the efficacy of the calcination reaction.

The process uses a two-stage counter-current leach circuit. Calcined solids are fed to the first tank in the primary stage and mixed with brine produced from the second-stage circuit to produce the primary leach slurry. The primary leach brine reports to the crystallization circuit. The second-stage leach circuit recovers essentially all of the potassium sulfate contained in the solids from the first-stage leach circuit and recycles it back to the first-stage leach circuit as brine. The separated solids are collected and transported to tailings disposal.

The crystallization circuit is designed to optimize recovery of SOP from brine produced in the leach circuit. Leonite, which is precipitated in the last stage of the process is dissolved in the leach brine. Dissolving this material in the leach brine does two things: (1) it increases the concentration of the brine, thus reducing the amount of evaporation required to reach the SOP crystallization point; and (2) it increases the amount of potassium sulfate contained in the feed to the SOP crystallizers, thus increasing the production of the desired end product.

Polyhalite seed material is added to the leonite dissolution tanks to aid in the removal of calcium oversaturation in the produced leach brine. The pre-concentration circuit further increases potassium and magnesium concentration in the brine which is fed to a clarifier which produces almost clear overflow brine for feed to the SOP crystallizer.

The SOP crystallization circuit uses two forced circulation mechanical vapor recompression (MVR) vessels configured in parallel to evaporate water from the clarified brine, resulting in the precipitation of about 30% of the potassium in the feed stream. SOP crystals are removed and sent to product drying. Mother liquor from the SOP crystallizer serves as the feed brine for the leonite crystallization circuit. The leonite circuit produces leonite crystals for recycle to the beginning of the crystallization circuit.

The crystal cake is first dried in a fluid bed dryer to remove any residual moisture. A series of multi-deck screens are used to size the crystals to meet specifications of soluble and standard products. Soluble and standard products are sent to the product day bins with the remaining material being sent to the granulation circuit to produce granular product.

The three SOP products are conveyed to their own dedicated storage bins at the site loading area. The products are then loaded and trucked approximately 22 miles over public roadways to the product storage and loadout facility where they are loaded onto rail cars or trucks for delivery to market.

A multi-story steel structure houses the process areas. Partial roof and wall enclosures have been added in the drying, sizing, and granulation areas for wet weather protection.

The gypsum tailings separated from the leach brine in the leaching circuit as precipitated solids are transported by trucks to the calcium sulfate storage pile. The magnesium sulfate ($MgSO_4$) bleed stream from the evaporation/crystallization circuit is delivered through a pipeline to the magnesium sulfate evaporation ponds. The sodium chloride (halite, NaCl) wash circuit bleed, boiler blowdown, and the reverse osmosis (RO) bleed streams are delivered by pipeline to the brine holding pond. Excess brine from the evaporation ponds is injected into an underground aquifer.

The water source for the plant operations is the Capitan Reef water well field located approximately 13 miles away. The plant will use both raw and treated water for different operations within the process. An RO water treatment plant located adjacent to the process plant will provide treated water for process and potable water needs.

1.7 Mineral Resource and Mineral Reserve Estimates

The mineral resource for the Ochoa Property comprises polyhalite mineralization within the Ochoa polyhalite bed, which is contained in the Tamarisk Member of the Rustler Formation. The Ochoa polyhalite bed occurs over most of the Property, with the exception of various detached leases to the east. The mineralization occurs as a generally undisturbed, flat-lying bed ranging between 4 and 6 ft thick inside the margins of the depositional basin. The bed dips gently to the southeast within the boundaries of the Property, flattening from a dip of up to 2 degrees (°) in the north to less than 0.5° in the south. Local steepening can occur at the basin margins.

The Ochoa polyhalite bed is the subject of this Technical Report (TR). This section identifies that portion of the Ochoa bed which qualifies as a NI 43-101 Mineral Resource and Mineral Reserve. The Mineral Reserve represents that portion of the Mineral Resource projected to be recoverable by the room-and-pillar mine plan developed in the FS. This method is similar to the methods commonly used in potash, coal, and trona underground mines. All tons are short (2,000 pounds [lbs]) tons (t).

1.7.1 Mineral Resources

Mineral Resources were estimated using a kriged gridded-seam computer geologic model constructed with Carlson Mining 2013 Software™. The Mineral Resource calculations are compliant with CIM Best Practice Guidelines for Industrial Minerals (2003).

Resource cutoffs of a 4.0-ft bed thickness and 65.0% polyhalite grade are considered reasonably conservative lower limits for potentially economic conventional underground mining in the Ochoa bed. A 65.0% polyhalite cutoff is equivalent to 10.0% K_2O, which is an economically reasonable cutoff commonly applied to potassium projects. These resource cutoffs do not preclude the possibility that thinner and/or lower grade polyhalite could be mined locally and remain economic as part of a larger mining operation.

Table 1-1 summarizes the Ochoa bed polyhalite Mineral Resources for the Property. The resources are reported on a dry tonnage basis. **Mineral Resources are inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.**

No reduction has been applied to the resource for possible undiscovered localized geological features, including faults, scours, channels, and other structural disturbances which may or may not affect economic mining. The presence of such structures at the prospective mining horizon and the extent to which these features could impact mining are risk factors. The relatively flat structure indicated by the high density of petroleum wells across the Property suggests that such risk is generally low.

1.7.2 Mineral Reserves

Table 1-2 states the Measured and Indicated (M&I) tonnage converted to Proven and Probable Reserve tonnage based on the 50-Year Mine Plan. No Inferred tons were included in the Reserve estimate. **Resource totals stated in Table 1-1 are *inclusive* of the Reserves stated in Table 1-2.**

Table 1-1. Ochoa Project Mineral Resource (effective date May 31, 2013)

	Average Thickness (ft)	Resource Area (acres)	In-Place Tons[1,2,3] (millions)	Polyhalite (wt %)	Equivalent K_2SO_4 (wt %)	Anhydrite (wt %)	Halite (wt %)	Magnesite (wt %)
MEASURED[4]	5.2	26,166	511.7	84.5	24.4	4.02	3.27	7.94
INDICATED[5]	5.0	26,698	506.0	83.3	24.1	4.00	3.30	8.61
TOTAL M&I	5.1	52,865	1,017.8	83.9	24.2	4.01	3.28	8.27
INFERRED[6]	4.8	15,634	284.0	82.6	23.9	4.11	3.37	8.82

[1] Average in-situ bulk density of 173.5 pounds per cubic foot (pcf).

[2] Bed thickness cutoff 4.0 ft, composite grade cutoff 65.0% polyhalite, excludes out-of-seam dilution.

[3] Mineral Resource includes Mineral Reserves.

[4] Measured Resource located within 0.75-mile radius from an exploration core hole.

[5] Indicated Resource located between 0.75-mile and 1.5-mile radius from an exploration core hole.

[6] Inferred Resource located between 1.5-mile and 3.0-mile radius from an exploration core hole.

Note: Gypsum weight percent negligible for all resource classifications.

Table 1-2. Ochoa Project Mineral Reserves (effective date January 9, 2014)

	Average Mined Thickness[1] (ft)	50-Year Mine Plan Mined Area (million ft^2)	ROM Mined Tons[2,3] (millions)	Mining Recovery[4] (%)	Polyhalite (wt %)	Equivalent K_2SO_4 (wt %)	Anhydrite (wt %)	Halite (wt %)	Magnesite (wt %)
PROVEN	5.9	246	125.0	47.1%	78.42	22.66	11.29	3.66	7.79
PROBABLE	5.9	113	57.4	64.8%	77.20	22.31	11.60	3.65	8.30
TOTAL P&P	5.9	359	182.4	51.5%	78.05	22.55	11.39	3.66	8.08

[1] Bed thickness cutoff 4.0 ft, composite grade cutoff 66.0% polyhalite, includes out-of-seam dilution.

[2] Average in-situ bulk density of 173.5 pcf.

[3] No inferred tons mined

[4] Areal recovery (mined area) inside 50 Year Mine Plan boundary

Note: Gypsum weight percent negligible for all resource classifications.

Mineral Reserves are included in Mineral Resources

Various risks are associated with mining the Reserves, which are independent of geologic confidence. Mineral Reserves could be adversely affected by mining conditions. Ore grade could be adversely affected by mining conditions and continuous miner operator differentiation of polyhalite vertical extents. Permitting delays would adversely impact the Ochoa Mine Project implementation schedule, but should not impact Mineral Reserves. Legal challenges could reduce available Mineral Reserves. Reduced productivity would increase operating (OPEX) and capital (CAPEX) costs adversely, affecting Project economics. Unfavorable court decisions on permit challenges could result in not receiving necessary permits.

The QPs have reviewed the FS and are satisfied that the CIMDS modifying factors have been adequately addressed; therefore, all Measured tons within the FS' 50-Year Mine Plan are presently classified as Proven tons.

1.8 Conclusions

1.8.1 Mineralization and Mining

ICP's Ochoa Property contains significant polyhalite mineralization in sufficient quantities and of sufficient grade to be attractive for mining and processing into SOP under current market conditions, notwithstanding the risks inherent to proving and developing any mining property. Geologic continuity in the mineralize polyhalite bed is strong throughout the Property.

The Property is suited to underground mining because of the depth to mineralization. Room-and-pillar mining methods are typical of the mining methods practiced in the nearby Carlsbad potash mines. The Ochoa Polyhalite Project Feasibility Study Report (FS) concludes that mining of the polyhalite is feasible and economic.

1.8.2 Processing

The processing plant design has been based on bench-scale and pilot plant testing and has confirmed the USBM research work of earlier decades. The Ochoa Polyhalite Project Feasibility Study Report (FS) concludes that processing the polyhalite into SOP is feasible and economic.

1.8.3 Economics

Table 1-3 summarizes the economic analysis for the Project. All costs are in 2013 US dollars (USD). Taxes and royalties are included in the cash flow.

Table 1-3. Financial Results (USD)

Full Equity Basis (i.e. No Debt)	Before Tax	After Tax
Capital Cost	$1,018 million	$1,018 million
Operating Cost per Ton SOP at Steady State	$195	$195
Internal Rate of Return (IRR)	17.8%	16.0%
Net Present Value (NPV), 8% Discount Factor	$1,502.3 million	$1,018.9 million
NPV, 10% Discount Factor	$942.7 million	$612.0 million
Payback Period (from start of production)	–	5.4 years

Table 1-4 shows the steady-state average annual OPEX for the Project. Steady state has been defined as the operating years 2022–2065. These years generally exclude major one-time OPEX that are included in years 2016 through 2021 such as equipment leasing, initial receding face expenditures, and inventory adjustments, as well as the costs associated with the Project's startup and closure.

Sustaining CAPEX for the life of the Project has been estimated at $1.407 billion USD. Three additional magnesium sulfate ponds with a cost of $9.3 million will be added in the first 3 years of operation. The Tailings Storage Facility (TSF) will be expanded every two years until reaching final capacity for a total cost of $115.2 million.

Table 1-4. Steady-State Average Annual OPEX (USD)

Operating Cost	2022 to 2065 Cost (millions)	Average Annual Cost (millions)	Cost/Ton of Ore	Cost/Ton of Product
Mining	$2,475.8	$56.27	$15.12	$78.76
Processing	$3,389.0	$77.02	$20.70	$107.82
General and Administrative	$267.4	$6.08	$1.63	$8.51
Total OPEX	$6,132.3	$139.37	$37.46	$195.09

Sensitivity analysis (±20%) was performed on the economic analysis taking into account variations in CAPEX, sustaining CAPEX, OPEX, and revenues. The Project remains economic throughout the range of sensitivities.

Project risks include permitting, economics, mining, ore grade dilution, processing, and project implementation. Opportunities include higher resource recovery, conversion of Resources to Reserves, and longer Project life.

The Project's implementation schedule will begin with receiving final permits and sufficient financing to complete bridge engineering while completing full financing. Site development includes sinking the mine shaft and driving the mine slope, and constructing the processing plant and ancillary facilities (roads, power lines, water pipelines, ponds, and similar items). Site development is estimated to take approximately 3 years from initial site preparation to full plant production.

1.9 Recommendations

1.9.1 General

Based on the results of the FS, it is recommended that ICP proceed immediately with partial funding for bridge engineering and early works while seeking full-time funding of the Project. Additional specific recommendations for mining and processing are listed in the sections below.

1.9.2 Mining

- Complete a continuous miner cutter head modeling study. Estimated cost is less than $25,000.
- Proceed with detailed mine design, mine substation design, detailed slope belt conveyor design, and the selection of the slope and shaft contractor(s), and the final design of the slope and shaft. Estimated cost is $2.2 million.
- Meet with district representatives of the Mine Safety and Health Administration (MSHA) to discuss the project in detail and confirm that the proposed mine plans are acceptable to MSHA. Estimated cost is minimal.
- Conduct additional geotechnical modeling to determine whether the production panel extraction ratio can be increased above 60%. Estimated cost is $50,000.
- Design a monitoring program for surface subsidence and an underground geotechnical monitoring program of data collection and analysis. Estimated cost is $22,500.

- Conduct FLAC3D modeling of gas and oil well casing and various well protective barrier pillar sizes, using updated extraction ratios. Estimated cost is $30,000.
- Include Priorities 3 and 4 of the mine geotechnical testing program as part of any exploration drilling that may be conducted in the future. Estimated cost of the geotechnical portion is $130,000.

1.9.3 Processing

1.9.3.1 Process Design Finalization—During the bridge engineering phase, several process activities will be carried out to advance the design of the Project to a level where implementation can successfully be initiated. Estimated cost is $65,000.

1.9.3.2 Process Optimization—Optimization activities would be focused on revising the design to lower the CAPEX of the Project while improving the technical capability of the design put forward. Estimated cost of these activities is $445,000.

1.9.3.3 TFS—Recommendations for future work required to complete the detailed design of the TSF are as follows:

- The design of the TSF is based on assumed materials properties for gypsum tailings. Laboratory tests should be completed to adequately characterize the tailings material, and the design and recommendations should be refined accordingly. The estimate of laboratory testing costs for tailings materials is $20,000.
- The design of the TSF is based on assumed material properties for the foundation soils. Laboratory tests should be completed to adequately characterize the foundation soils, especially strength properties, and the design and recommendation should be refined accordingly. The estimate of laboratory testing costs for foundation materials is $40,000.
- The number of boreholes located in the proposed TSF area, especially in the south of the TSF is limited. Additional boreholes are required to adequately define the stratigraphy and hydrogeology. Additionally, the engineering properties of the foundation materials should be further investigated with sampling and testing programs (*in situ* and laboratory). For drilling investigation planning, field supervision and reporting costs are estimated at $30,000. This does not include expenses (travel, accommodations, sustenance, etc.) or disbursements (drilling subcontractor, materials, courier/shipping, etc.).

1.10 Forward Looking Information

Certain information set forth in this TR may contain forward-looking statements that involve substantial known and unknown risks, uncertainties, and other factors which may cause the actual results, performance, or achievements of ICP to be materially different from any future results, performance, or achievements expressed or implied by such forward-looking statements. Forward-looking statements include statements that use forward-looking terminology such as “may,” “will,” “expect,” “anticipate,” “believe,” “continue,” “potential” or the negative thereof or other variations thereof or comparable terminology. Such forward-looking statements include, without limitation, reserve estimates, ICP’s expected position as one of the lowest cost producers of SOP in the world, the timing of receipt and publication of ICP’s environmental permits, the sufficiency of ICP’s cash balances, the timing of production, and other statements that are not historical facts. These forward-looking statements are subject to numerous risks and uncertainties, certain of which are beyond the control of ICP, including, but not limited to, risks associated with mineral exploration and mining activities, the impact of general economic conditions, industry conditions, dependence upon regulatory approvals, the uncertainty of obtaining additional financing, and risks associated with turning reserves into product. Readers are cautioned that the assumptions used in the preparation of such information, although considered reasonable at the time of preparation, may prove to be imprecise and, as such, undue reliance should not be placed on forward-looking statements.

ITEM 2: INTRODUCTION

2.1 Terms of Reference

The subject of this report is the approximately 89,787-acre Ochoa Mine Property (Property) located in Lea and Eddy Counties, New Mexico, approximately 60 miles east of Carlsbad, New Mexico, USA. The Property is controlled by IC Potash Corp. (Company), a Canadian federally incorporated public company traded on the Toronto Stock Exchange (TSX). The Company owns 100% of Intercontinental Potash Corp., a Canadian federally incorporated corporation, which owns 100% of Intercontinental Potash Corp. (USA), a Colorado incorporated company. ICP refers to the Company and/or all affiliates, and is used interchangeably. Through its indirect wholly owned subsidiary, Intercontinental Potash Corp. (USA), the Company holds a 100% interest in the Project. Development of the Company's Ochoa Project (the Project) is being carried out through Intercontinental Potash Corp. (USA).

The Company was incorporated under the Canada Business Corporations Act on November 8, 2002. The Company's head office is located at First Canadian Place, 100 King Street West, Suite 5600, Toronto, Ontario, M5X 1C9. Its registered office is located at 36 Toronto Street, Suite 1000, Toronto, Ontario, M5C 2C5.

The Company is a reporting issuer under applicable securities legislation in the provinces and territories of Alberta, British Columbia, Ontario, Saskatchewan, Manitoba, New Brunswick, Nova Scotia, Prince Edward Island, Newfoundland, and the Northwest Territories. Its outstanding Common Shares are listed on the TSX under the symbol "ICP" and traded on the OTC Markets Group Inc. (OTCQX) under the symbol "ICPTF." For the purpose of this report, ICP refers to the Company and all affiliates.

ICP acquired rights to the Property via BLM prospecting permits covering approximately 61,983 acres and convertible to Preference Rights Leases (PRL), and NMSLO mining leases covering approximately 27,804 acres. ICP has applied for and been offered seven new BLM prospecting permits covering approximately 12,484 acres. ICP has acquired options for surface leases and ROWs.

ICP's objective is to become a primary producer of high-quality SOP by mining and processing polyhalite from its Property to supply regional and international markets. The Project is considered to be at the feasibility stage.

Agapito Associates, Inc. (AAI) was commissioned by ICP to compile an independent QP authored NI 43-101 TR. This report incorporates relevant information from five previous TR's—Micon (2008), Gustavson (2009); Gustavson (2011a), Gustavson (2011c), and Gustavson (2011d), the maiden reserve NI 43-101 TR.

ICP originally developed plans for a room-and-pillar mine plan in the Ochoa polyhalite bed as part of a PFS TR published in December 2011 (Gustavson 2011d). In January 2014, SNC-Lavalin, Inc. (SNCL) compiled an FS (2014) for ICP.

The purpose of this TR is to update the polyhalite Mineral Resources and Mineral Reserves based on (1) exploration information through May 25, 2013 and (2) the results of the January 2014 FS.

The FS encompasses the exploration, geologic modeling, Resource and Reserve estimation, mine planning and design, mining methodology and equipment, mineral processing and metallurgical testing, surface infrastructure, labor, environmental and permitting, marketing, project economics, project development schedule, and risk analysis in support of a project to mine and process polyhalite to produce SOP.

ICP retained the following consulting companies to assist with the development of the FS:

- SNCL—lead consultant charged with compiling the FS, including developing and/or reviewing the processing plant design, site surface infrastructure design, Jal loadout design, three-dimensional (3D) model development (plant), project execution plan, project schedule, operations readiness plan, capital and operating cost estimates (CAPEX and OPEX), economic and financial analysis, risk assessment, and conclusions and recommendations
- AAI—reviewing and auditing the exploration program, developing the resource geologic model, and the resources and reserves estimates, mine geotechnical design and mine engineering, developing the mine operating and capital cost estimates, mine risk assessment, and mining related recommendations and conclusions. AAI subcontracted Bruno Engineering for the mine electrical distribution system and communication and monitoring systems designs and cost.
- Veolia Water Solutions and Technologies (Veolia)—specialized process design for evaporation and crystallization circuits and pilot plant testing.
- NovoPro Projects Inc.—owner's engineer (processing), process development, testing, and contract development services.
- Resource Development Inc. (RDi)—provided overall technical reviews of processing technology and surface facilities.
- Upstream Resources—carried out the substantial portion of the exploration programs and subsequent data analysis and interpretation and geological modeling.
- Hazen Research Inc. (Hazen)—research and development services in the adaptation of known technology to new situations, pilot plant testing, preliminary engineering and cost analysis.
- INTERA Incorporated (INTERA)—provided coordination of air and groundwater permitting and hydrogeological modeling.
- Walsh Environmental Scientists and Engineers (Walsh)—contributed to environmental permitting and related activities.

Other consultants contributing to the FS were:

- AB Engineering Inc.
- Chastain Consulting
- Chemfelt Engineers
- FEECO International
- Fakatselis Consulting Inc.
- Gundlach Equipment Corporation (Gundlach)
- Harrison Western Construction Corp. (HWCC)

- Hard Rock Consulting LLC
- Metso Minerals Industries Inc. (Metso)
- SGS Lakefield Research Limited (SGS)
- Sage Earth Sciences
- Western Technologies

Technical personnel from AAI and SNCL are the QP authors of this TR, as summarized below, and identified in the Certificates of Qualified Person included at the end of this report.

The authors obtained information and data during multiple meetings beginning in early 2012 through January 2014 at ICP's offices located in Golden, Colorado, SNCL's offices in Saskatoon, Saskatchewan, Canada, and at various other offices of the FS participates and at site visits in 2012 and 2013. A rail operation site visit to Hobbs, New Mexico, occurred on August 30, 2012. ICP provided the authors with the following information:

- Overall project scope
- Property ownership and location
- Mineral tenure status
- Boundary information
- Land tenure title opinions
- Gas and oil well database
- Regional and local geology
- 2009 through May 25, 2013 exploration drilling and chemical analysis data (Additional chemical analysis data not available for resource modeling was available for geochemistry analysis prior to the completion of the FS.)
- 2009–2013 exploration program QA/QC protocol documents
- Permit status, draft Environmental Impact Statement (EIS)
- SGS Lakefield Roll Crusher Testing Report (FS Appendix A6-1)
- Rod Mill Grinding and Centrifuge Tests with Gray Polyhalite (FS Appendix A6-2)
- Polyhalite Laboratory Program (FS Appendix A6-3)
- HPD[1] Technical Report Pilot Scale Crystallization of Langbeinite (FS Appendix A6-4)
- HPD Bench Scale Crystallization of Langbeinite and Conversion of Langbeinite (FS Appendix A6-5)
- Calcining Gray Polyhalite Ore in an Indirectly Heated Fluid Bed (FS Appendix A6-6)
- Calcining Gray Polyhalite Ore in a Direct-Heated Fluid Bed (FS Appendix A6-7)
- Ontario Stock Commission Demonstration Test Plan (FS Appendix A6-8)
- Bench Scale Dissolution of Leonite and Crystallization of SOP with Investigation of Calcium Precipitation (FS Appendix A6-9)
- Recovery of K_2SO_4 from Ochoa Polyhalite Ore Pilot Plant Demonstration (FS Appendix A6-10)
- Gray Ore Pilot Plant Report (FS Appendix A6-11)
- Feasibility Pilot Plant Demonstration for the Recovery of K_2SO_4 and $MgSO_4$ from Ochoa Polyhalite Ore (FS Appendix A6-12)
- Pilot Scale Crystallization of Potassium Sulfate and Leonite from Calcined Ore Leach Brine (FS Appendix A6-13)
- Well Packer Test Water Quality Results (FS Appendix A6-14)

[1] HPD and Veolia Water Solutions and Technologies are used interchangeability throughout this report.

- Settling and Filtration of Polyhalite Pilot Leach Residue Slurries (FS Appendix A6-15)
- Thickening, Settling, and Filtration of Unwashed Gray Polyhalite Slurries (FS Appendix A6-16)
- Process Alternative Trade-off Investigation Report (FS Appendix A6-17)
- Assessment of Polyhalite Gypsum Dominant Waste Solubility (FS Appendix A6-18)
- Water-Cooled Versus Air-Cooled Surface Condensers Trade-off Study (FS Appendix A6-19)
- Gundlach Sieve Scale Screen Test Data (FS Appendix A6-20)

Key references are included in Item 27 of this TR.

Relevant data were reviewed in sufficient detail for preparation of the FS and this TR. The following personnel are independent QP's for this TR:

<u>Agapito Associates, Inc.</u>

- Gary L. Skaggs, P.E., P.Eng., acted as project manager for AAI's role portion of the FS and as AAI's project manager for the compilation of this TR (Items 1.1, 1.2, 1.3.2, 1.4, 1.7.2, 1.8.1, 1.9.1, 1.9. 2,, 3.1 – 3.7, 4, 5, 6, 7.6, 15, 16, 19, 20, 211..2, 21.1.7.3, , 23, 24, 25.1, 25.3, 25.6.2, 26.2, and 27), reviewed technical data, oversaw the development of the room-and-pillar mine plan, cuttability analysis, mine access, and underground mining engineering design, oversaw the estimation of Mineral Reserves and the mine operating and capital cost, and participated in the risk analysis for the FS. Mr. Skaggs conducted a site visit September 20–21, 2012.

- Leo J. Gilbride, P.E., reviewed technical data, conducted geologic resource modeling and developed the Mineral Resource estimate (Items 1.7.1 and 14). Mr. Gilbride conducted a site visit September 26–28, 2012.

- Susan B. Patton, Ph.D., P.E., developed the productivity analysis, ventilation analysis, mine safety and health regulation listing, and the mine operating and capital cost (Items 16.2.4, 16.7, 21.1.7.3, , and 21.2.2).

- Thomas L. Vandergrift, P.E., oversaw the mine geotechnical analysis, pillar design, roof anhydrite modeling, shaft lining convergence, and subsidence (part of Item 16).

- Vanessa Santos, P.G., reviewed and audited the geology, exploration, drilling, sample preparation, analysis, and security programs and the QA/QC documentation (Items 1.3.1, 1.5, and 7–12). Ms. Santos conducted a site visit September 25–28, 2012.

<u>SNC-Lavalin Inc.</u>

- Lawrence Berthelet, P.Eng., SNCL project manager for the development and compilation of the FS. He oversaw all aspects of the FS coordination with the various consultants and SNCL in-house analysis and engineering designs, including process flow diagrams, process plant design, infrastructure, project planning and scheduling for construction, risk assessment, CAPEX and OPEX and economic analysis (Items 1.8.3, 1.9.1, 3.4, 18, 21.1 (except for 21.2.1 and 21.1.7.3), 21.2 (except for 21.2.2), 22, 25.2, 25.4, 25.5, 25.6.1, 25.6.3, 26.1, 26.3, and 27). Mr. Berthelet visited the site on November 28, 2012 and February 12, 2013.

- Jack Nagy, P.Eng., monitored the development of the mineral process testing and plant design during the FS (Items 1.6, 1.8.2, 1.9.3, 3.8, 13, and 17).

2.2 Units

Units used in this report are expressed in imperial (USA) unless otherwise noted. As the project is located in the USA, currencies are expressed in September 15, 2013 USD.

2.3 List of Acronyms and Abbreviations

A J Roth and Associates	Roth
Adobe Portable Document Format	PDF
Agapito Associates, Inc.	AAI
Air Quality Bureau	AQB
ambient temperature saturated SOP brine	ATSB
anhydrite	$CaSO_4$
Analysis of Roof Bolting Systems program	ARBS
animal unit month	AUM
Association of American Plant Food Control Officials	AAPFCO
atmospheric monitoring system	AMS
below ground surface	bgs
British Thermal Unit	BTU
Bureau of Labor and Statistics	BLS
Bureau of Land Management	BLM
Calcium	Ca
calcium oxide	CaO
calcium sulfate	$CaSO_4$
calories per mole	cal/mol
Canada Center for Mineral and Energy Technology	CANMET
Canadian Institute of Mining, Metallurgy and Petroleum	CIM
capital cost estimate	CAPEX
cascading pillar failure	CPF
CIM Definition Standards on Mineral Resources and Mineral Reserves	CIMDS
carnallite	$KMgCl_3 \bullet 6(H_2O)$
Carlson Mining 2013's Underground Mining Module	Carlson
Central Basin Platform	CBP
Certified Reference Material	CRM
chlorine	Cl
Class One Technical Services Inc.	COTS
Colorado School of Mines	CSM
Comprehensive Integrated Subsidence Prediction Model	CISPM
degrees	°
Department of the Interior	DOI
diesel particulate matter	DPM
differential thermal analysis	DTA
Draft Environmental Impact Statement	DEIS
dual-split super section	DSSS
Earth Mechanics Institute	EMI
engineer, procure, and construct	EPC
engineering, procurement and construction management	EPCM
Environmental Impact Statement	EIS
Epsom salt	$MgSO_4 \bullet 6H_2O$
equation	Eqn.

Fahrenheit	F
Feasibility Study	FS
feet/foot	ft
feet per day	fpd
Final Environmental Impact Statement	FEIS
freight on board	FOB
frequency per minute	fpm
heating, ventilation, and air conditioning	HVAC
horsepower	hp
Harrison Western Construction Corp.	HWCC
gallons per minute	gpm
gram	g
Great Salt Lake Minerals Corporation	GSL
gross profit royalty	GPR
Gundlach Equipment Corporation	Gundlach
Gustavson Associates	Gustavson
gypsum	$CaSO_4 \cdot 2H_2O$
halite (sodium chloride)	NaCl
Hazen Research Inc.	Hazen
H&M Analytical Services	H&M
High Recovery Membrane	HRM
hydrogen chloride	HCl
hydrogen sulfide	H_2S
IC Potash Corp.	the Company
IC Potash Corp. and/or affiliates	ICP
inductively-coupled plasma-optical emission spectroscopy	ICP-OES
instantaneous cutting rate	ICR
Internal rate of return	IRR
INTERA Incorporated	INTERA
Intercontinental Potash Corp. (USA)	ICP
International Centre for Diffraction Data	ICDD
International Fertilizer Industry Association	IFA
Interstage Precipitation Reactor	IPR
Inorganic Crystal Structure Database	ICSD
inverse distance squared	ID^2
Joy Global	Joy
K+S KALI GmbH	K+S KALI
kilovolt	kV
kilowatt	kW
kilowatt-hour	kWh
kilowatt-hour per ton	kWh/t
langbeinite	$K_2Mg_2(SO_4)_3$
Layne Heavy Civil Inc.	Layne
leonite	$K_2SO_4 \bullet MgSO_4 \bullet 4H_2O$
linear cutting machine	LCM
licensed professional surveyor	LPS
liquid to solid	L/S
load haul dump	LHD
magnesite	$MgCO_3$
magnesium	Mg
magnesium oxide	MgO

magnesium sulfate	$MgSO_4$
maintenance and repair	M&R
mean sea level	MSL
Measured and Indicated	M&I
mechanical vapor recompression	MVR
megapascals	MPa
megavolt amperes	MVA
megawatt	MW
megawatt-hour	MWh
Memorandum of Understanding	MOU
Metso Minerals Industries Inc.	Metso
microdarcy	µD
million British Thermal Units	MMBTU
Mine Safety and Health Administration	MSHA
milligrams per liter	mg/l
million ton	Mt
million tons per year	Mtpy
million years before present	MYBP
Mine Plan of Operations	MPO
Motor Control Center	MCC
muriate of potash	MOP
National Environmental Protection Act	NEPA
National Institute of Occupational Safety and Health	NIOSH
National Institute of Standards and Technology	NIST
National Instrument	NI
Net Present Value	NPV
New Mexico Administrative Code	NMAC
New Mexico Environment Department	NMED
New Mexico Environmental Department Air Quality Board	NMAQB
New Mexico Office of the State Engineer	NMOSE
New Mexico Principal Meridian	NMPM
New Mexico State Land Office	NMSLO
nitrogen, phosphorous, and potassium	NPK
North American Datum of 1983	NAD83
Ochoa Mine Project (means the mine project)	the Project
Ochoa Property (means the ICP land holdings)	the Property
operating cost estimate	OPEX
optical emission spectrometry	OES
original equipment manufacturer	OEM
OTC Markets Group Inc.	OTCQX
out-of-seam dilution	OSD
particle size distribution	psd
percent	%
polyhalite	$K_2Ca_2Mg(SO_4)_4 \cdot 2H_2O$
Potash Corporation of America	PCA
potassium	K
potassium oxide	K_2O
potassium sulfate	K_2SO_4
potassium magnesium sulfate	$K_2Mg_2SO_4$
calcined polyhalite	$K_2CaMg(SO_4)_3$
pounds	lbs

pounds per cubic foot	pcf
pounds per hour	lb/h
pounds per square inch	psi
pounds per ton	lb/t
Preference Rights Lease	PRL
Preliminary Feasibility Study	PFS
Prevention of Significant Deterioration	PSD
programmable logic controllers	PLCs
public switched telephone network	PSTN
Record of Decision	ROD
Rights-of-Way	ROW
Qualified Person	QP
quality assurance and quality control	QA/QC
quarter	Q
radius of influence	ROI
Resource Development Inc	RDi
reverse osmosis	RO
rock quality designation	RQD
run-of-mine	ROM
SGS Lakefield Research Limited	SGS
silica dioxide	SiO_2
SNC-Lavalin, Inc.	SNCL
Sociedad Quimica y Minera	SQM
sodium	Na
sodium chloride (halite)	NaCl
solid-liquid separation	SLS
South African Council for Scientific and Industrial Research	CSIR
splitting tensile strength	Brazilian
square feet	ft^2
square feet per day	ft^2/day
stability factor	SF
standard reference material	SRM
State Highway	SH
State Historic Preservation Office	SHPO
strontium	Sr
strontium oxide	SrO
Sulfate of Potash	SOP
sulfate of potash magnesia	SOPM
sulfur	S
sulfur trioxide	SO_3
Tailings Storage Facility	TSF
The Fertilizer Institute	TFI
Technical Report	TR
Texas-New Mexico Railroad	TNMR
thermogravimetric analysis	TGA
thousand cubic feet per minute	kcfm
thousand pounds per square inch	ksi
three-dimensional	3D
tons (2,000 lbs, short)	t
tons per cubic foot	t/ft^3
tons per hour	tph

tons per minute	tpm
tons per year	tpy
Toronto Stock Exchange	TSX
total depth	TD
total dissolved solids	TDS
total suspended solids	TSS
triaxial compressive strength	TCS
uniaxial compressive strength	UCS
uniaxial compressive strength with elastic properties	UCS-E
ultra-high frequency	UHF
Union Pacific/Southern Pacific Railroad	UPSP
United States Army Corps of Engineers	USACE
United States Bureau of Mines	USBM
United States Department of Energy	USDOE
United States dollars	USD
United States Geological Survey	USGS
United States of America	US or USA
Veolia Water Solutions and Technologies	Veolia
volt	V
Walsh Environmental Scientists and Engineers	Walsh
Waste Isolation Pilot Plant	WIPP
water	H_2O
water treatment plant	WTP
weight percent	wt-%
x-ray diffraction	XRD
x-ray fluorescence	XRF
Yara International ASA	Yara

ITEM 3: RELIANCE ON OTHER EXPERTS

3.1 General

The authors of this TR state they are QPs for those areas as identified in the appropriate QP "Certificate of Qualified Persons" attached to this report. The authors have relied upon the following expert reports described below pertaining to mineral tenure, surface rights, access, seismic interpretations, marketing, hydrology, geochemistry, environment, and permitting as allowed under Item 3 of Form 43-101F1.

This TR carries forward the general body of information reported in the following documents:

- NI 43-101 TR titled *Independent Technical Report on the Ochoa Polyhalite Project, New Mexico*, dated November 2008, prepared by Micon (2008)
- NI 43-101 TR titled *Independent Technical Report on the Ochoa Polyhalite Project of Intercontinental Potash Corp., New Mexico*, dated November 2008, revised January 2009, prepared by Micon (2009)
- NI 43-101 TR titled *Polyhalite Resources and a Preliminary Economic Assessment of the Ochoa Project, Lea County, Southeast New Mexico*, dated 19 August 2009, prepared by Chemrox Technologies Corp. (Chemrox) and Gustavson (2009)
- NI 43-101 TR titled *NI 43-101 Technical Report on the Polyhalite Resources and Updated Preliminary Economic Assessment of the Ochoa Project, Lea County, Southeast New Mexico*, dated January 14, 2011, prepared by Gustavson (2011a)
- Report for BLM titled *Ochoa Project, Mine Plan of Operations, Lea County, New Mexico*, dated September 30, 2011, prepared by Gustavson (2011b)
- NI 43-101 TR titled *NI 43-101 Technical Report on the Polyhalite Resources and Updated Mineral Resources Estimate for the Ochoa Project, Lea County, Southeast New Mexico*, dated November 25, 2011, prepared by Gustavson (2011c)
- NI 43-101 TR titled *NI 43-101 Technical Report, Prefeasibility Study for the Ochoa Project, Lea County, New Mexico*, dated December 30, 2011, prepared by Gustavson (2011d)
- Gustavson, *Prefeasibility Study of the Ochoa Project, Lea County, New Mexico*, prepared for Intercontinental Potash Corp (USA), effective date April 15, 2012

3.2 Mineral Tenure

The QP's have not reviewed mineral tenure, nor independently verified the legal status or ownership of the mineral title and underlying property agreements. The QP's have relied upon and disclaim responsibility for information supplied by ICP and independent experts retained by ICP with respect to mineral tenure, which is represented in Item 4 of this TR, including information derived from the following documents:

- Eighteen (18) letters from Holland & Hart LLC, legal counsel, dated beginning March 4, 2012 and ending April 8, 2013, offering title opinions on various NMSLO mining leases

- Thirty-three (33) letters from Holland & Hart LLC, legal counsel, dated beginning June 15, 2012 and ending September 7, 2012, offering title opinions on various Federal potassium prospecting permits and leases (see Item 4 for listing of permits and leases)

3.3 Surface Rights and Access

The QP's have not reviewed surface rights and access agreements, nor independently verified the legal status or ownership of the surface title and underlying property agreements. The QP's have relied upon and disclaim responsibility for information supplied by ICP and independent experts retained by ICP with respect to surface rights and access, which is represented in Item 4 of this TR, including information derived from the following documents:

- Jal Loadout Site, including access road to highway SH 128
  - Trustees of the Jal Public Library Fund: portion of Loadout site, Option to Lease, July 22, 2013, 5 years land is under Commitment for Title Insurance No. F13-312W1, First American Title Insurance Company, March 6, 2013
  - RRR Land & Cattle Company LLC: portion of Loadout site, Option to Lease, June 5, 2013, 5 years
  - Christeen Pruett: portion of Loadout site, Option to Lease, June 6, 2013, 5 years
  - Johnny Chapman: haul road segment, Option for Easement Agreement, June 25, 2013, 3 years with option to extend 1 additional year
  - BLM, segment of haul road, ROW approval concurrent with EIS ROD
  - NMSLO, segment of pipeline, ROW application and approval following EIS ROD and detailed design
- Wellfield
  - State of New Mexico Commissioner of Public Lands, application and approval following EIS ROD and detailed design
- Water Pipeline, Well Field to Process Plant
  - Bert Madera, segment of pipeline, Option for Easement Agreement, October 30, 2013, 3 years with two successive 1-year options to extend
  - New Mexico Department of Transportation, segment of pipeline in SH 128 ROW, Letter of Intent, June 5, 2012
  - Trustees of the Jal Public Library, segment of pipeline, in process, pending resolution of Jal Public Library title issues
  - BLM, segment of pipeline, ROW approval concurrent with EIS ROD
  - NMSLO, segment of pipeline, ROW application and approval following EIS ROD and detailed design
- Mine and Process Plant Site
  - NMSLO, approval implicit in mining leases (Lease HP-0045)
  - BLM, plant site approval concurrent with EIS ROD

3.4 Surface Geotechnical

ICP completed surface geotechnical exploration for surface structure foundation design with Western Technologies Inc. of Albuquerque, New Mexico, an independent industry recognized surface geotechnical specialist. The QP's have relied on this independent expert retained by ICP for Items 16 and 18 of this TR through the document titled "Geotechnical

Evaluation, ICP Ochoa Project, Lea County, New Mexico, Job No. 3222JJ042, Revision No. 1" Albuquerque, New Mexico, 26 April 2015.

3.5 Marketing

An independent marketing analysis was completed for the PFS and associated TR (Gustavson 2012, 2011d) by CRU Strategies (CRU), an independent, industry-recognized marketing expert retained by ICP. Arthur Roth, ICP Director of Marketing and Chief Executive Office of A J Roth and Associates (Roth), an independent industry consulting firm that specialized in agribusiness, fertilizer, minerals and energy business, provided an updated marketing summary for the FS. The QP has reviewed these analyses and relied upon the results and conclusions produced by CRU and those results as updated by Roth in Item 19 of this TR through the document titled "Potassium Sulphates and Potassium Nitrate Market Outlook (2012 Edition)," (CRU 2012) and *Chapter 22 – Marketing* published in the FS (SNCL 2014). The results support the assumptions in this TR.

3.6 Geochemistry and Hydrology

The geochemistry analysis of the polyhalite bed was overseen by ICP based on x-ray diffraction (XRD) and x-ray fluorescence (XRF) analysis and by elemental measurements as a check. H&M Laboratories in Allentown, New Jersey, conducted the XRD and XRF analysis. The QPs have reviewed the procedures and have relied upon these geochemical analyses presented in Item 7 of this TR through Chapter 8.2 of the FS (SNCL 2014).

Independent hydrology studies at the at the plant water supply well site were completed for the FS by INTERA Incorporated (INTERA 2012, 2013a, 2013b) an independent environmental, water resources, and waste isolation design firm retained by ICP. The QPs have relied upon the results and conclusions produced by INTERA in Item 7 of this TR through Chapters 5 and 8 published in the FS (SNCL 2014). In addition, INTERA (2013b) conducted an independent groundwater study at the shaft site and the QPs have relied on this independent expert retained by ICP for Item 16 of this TR through the document titled "Estimated Shaft Groundwater Inflows Based on Formation Testing Conducted in ICP-092" dated May 31, 2013.

3.7 Environmental and Permitting

An independent environmental and socio-economic assessment, permitting schedule, and reclamation evaluation was completed for the FS by INTERA (2013a) and Walsh (2013a). In addition, INTERA acted as an ICP consultant for the EIS. The QPs have relied upon the results and conclusions produced by INTERA in Item 20 of this TR through Chapter 5 published in the FS (SNCL 2014).

3.8 Mineral Processing and Metallurgical Test Work

SNCL was responsible for the design of the process based on information received from AAI, Layne Heavy Civil Inc. (Layne), Veolia, and Hazen. While SNCL provided overview and due diligence in the review of the results of testing, they are relying on the originators and supervisors of the test work to have applied best industry practice in the setup and execution of the sampling and test work.

ITEM 4: PROPERTY DESCRIPTION AND LOCATION

4.1 Mineral and Surface Land Tenure

ICP's Property for the Ochoa Mine Project is located in Lea and Eddy Counties, New Mexico, and consists of 89,787 acres, more or less, with 28 US DOI, BLM prospecting permits (61,983 acres), and 18 NMSLO State Trust Lands potash mining leases (27,804 acres), based on legal descriptions. ICP's permits and leases are shown on Figure 4-1 and listed in Tables 4-1 and 4-2. ICP recently relinquished five BLM prospecting permits and previously relinquished one permit because exploration drilling indicated that resources in these leases were not adequate. However, ICP applied for, and has been offered seven new BLM prospecting permits for 12,484 additional acres listed in Table 4-3. These new BLM permits will be subject to the royalties pursuant to BLM PRLs, once the Ochoa Project comes into production.

The BLM prospecting permits have a term of 2 years with one 2-year extension. The BLM will convert a prospecting permit to a PRL, which allows mining, when it has been demonstrated that a valuable mineral resource has been discovered on any portion of the prospecting permit and that the lands are "chiefly valuable" for the potassium mineral.

ICP has applied to convert 26 BLM prospecting permits (58,226 acres) into PRLs, based on the results of ICP's exploration program. These 26 BLM prospecting permits, along with 10 of the 18 NMSLO potash mining leases contain almost the entire ICP Ochoa Project 50-Year Mine Plan. Additional PRLs will be applied for as exploration results warrant. All indications are BLM will agree that the ICP exploration program results indicate a valuable mineral resource is present on ICP's prospecting permits, the lands are chiefly valuable for the potassium mineral, and the permits will be converted after the EIS is finalized and a Record of Decision (ROD) issued.

The BLM PRLs do not expire, but are subject to readjustment every 20 years. By accepting ICP's application to convert these prospecting permits to PRLs, these prospecting permits will not lapse during the period required to obtain permits for development.

The NMSLO mineral leases have a term of 10 years. The leases are automatically extended as long as the average annual production, over any three consecutive years, is enough to generate the minimum required royalty.

ICP had intended to apply for PRLs for Permits NMNM121100 through NMNM121104 and NMNM121106; however, geophysical data and core analysis indicated results from exploratory drill-hole ICP-095, drilled in support of the application, did not justify proceeding with the PRL application and the permits were relinquished in November 2012 and January 2013.

ICP has negotiated options for surface leases and ROW.

Title Opinion

The legal firm Holland & Hart, LLP has provided title opinions on the Ochoa Project's BLM prospecting permits and NMSLO leases in 2012 and 2013. The title opinions evaluated the following:

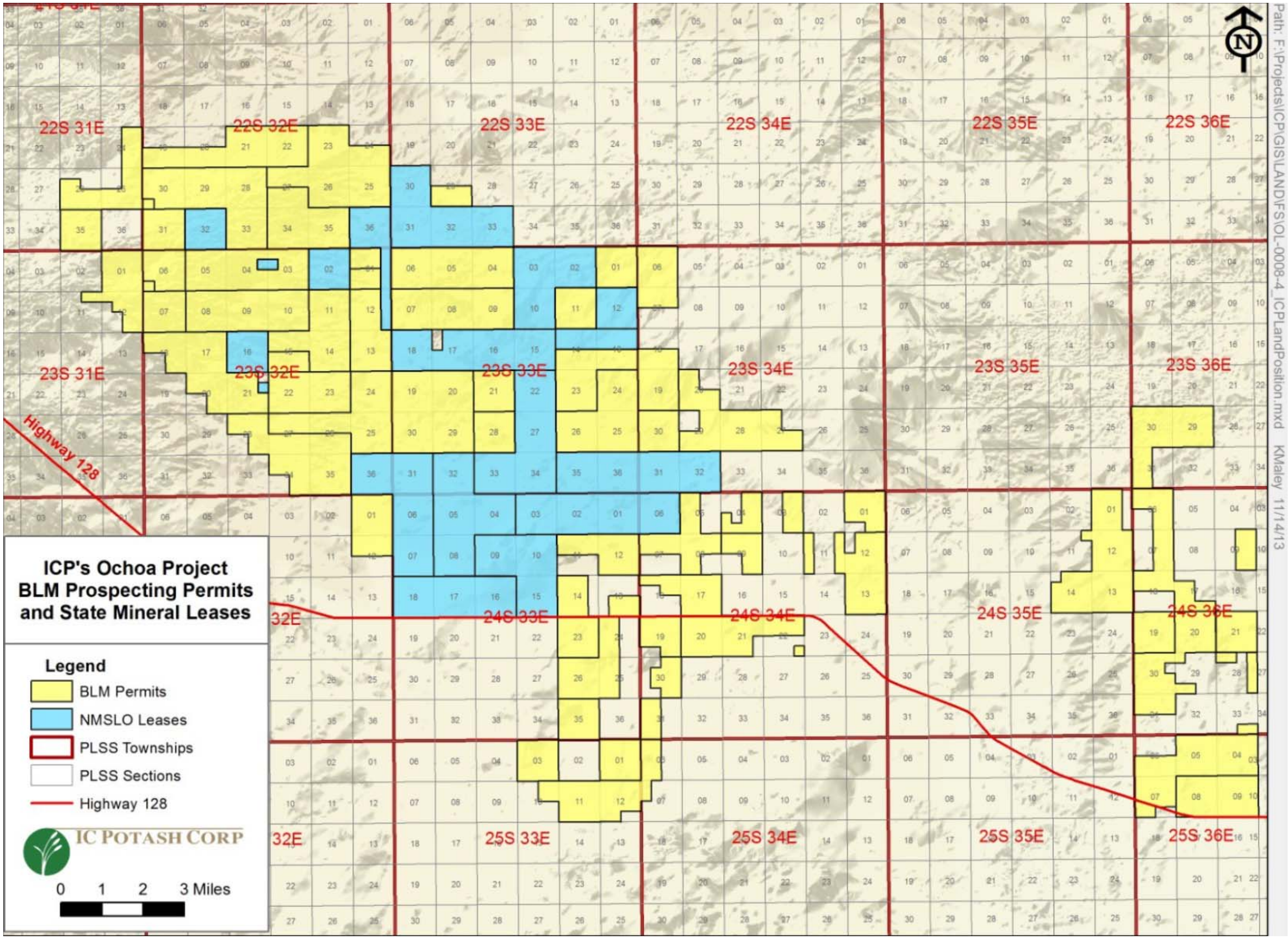


Figure 4-1. Ochoa Project BLM Prospecting Permits and State Mineral Leases

Table 4-1. Ochoa Project BLM Prospecting Permits

Serial Number	Township and Range	Sections and Descriptions		BLM Approval Date (dd-mm-yyyy)	Acreage*
121105	Township 24 South, Range 34 East, NMPM	Section 9:	N2, Southeast(SE)4	12/1/2008	2,560
		Section 11:	West(W)2W2, E2E2		
		Section 12	E2, Southwest (SW4), E2NW		
		Section 13:	All lands		
		Section 19:	N2, SE4, N2SW4		
121107	Township 23 South, Range 34 East, NMPM	Section 6:	Lots 1–7, SE4Northwest(NW)4, E2SW4, S2Northeast(NE)4, SE4	12/1/2008	1,892
		Section 7:	Lots 1–2, E2NW4, NE4		
		Section 18:	Lots 3–4, E2SW4, SE4		
		Section 19:	Lots 1–4, E2W2, E2		
121108	Township 24 South, Range 34 East, NMPM	Section 1:	Lots 1–4, S2N2, N2SW4, SE4	12/1/2008	2,439
		Section 3:	Lots 1–2, S2NE4, SE4		
		Section 4:	Lots 1–2, S2NE4, SE4, S2SW4, NW4SW4		
		Section 5:	Lots 3–4, S2NW4, SW4		
		Section 7:	Lots 1–2, E2NW4, NE4		
		Section 8:	N2, SW4		
121109	Township 24 South, Range 33 East, NMPM	Section 11:	N2	12/1/2008	2,320
		Section 12:	All lands		
		Section 13:	SE4, E2SW4		
		Section 14:	W2, W2E2		
		Section 23:	All lands		
121110	Township 24 South, Range 33 East, NMPM	Section 24:	W2	12/1/2008	1,280
		Section 25:	W2		
		Section 26:	All lands		
121111	Township 23 South, Range 33 East, NMPM	Section 24:	All lands	12/1/2008	2,560
		Section 25:	All lands		
		Section 26:	All lands		
		Section 28:	All lands		
121112	Township 24 South, Range 34 East, NMPM	Section 17:	All lands	12/1/2008	2,440
		Section 18:	Lot 1, NE4NW4, NE4		
		Section 20:	All lands		
		Section 21:	N2, SW4, W2SE4		
		Section 22:	N2, S4ESE4		
121113	Township 23 South, Range 33 East, NMPM	Section 13:	S2	12/1/2008	1,920
		Section 14:	S2		
		Section 21:	All lands		
		Section 23:	All lands		
121114	Township 23 South, Range 33 East, NMPM	Section 1:	Lots 1–4, S2N2, S2	12/1/2008	2,547
		Section 4:	Lots 1–4, S2N2, S2		
		Section 5:	Lots 1–4, S2N2, S2		
		Section 6:	Lots 1–7, E2SW4, SE4NW4, S2NE4, SE4		
121115	Township 23 South, Range 33 East, NMPM	Section 7:	Lots 1–4, E2W2, E2	12/1/2008	2,551
		Section 8:	All lands		
		Section 9:	All lands		
		Section 11:	All lands		

Table 4-1. Ochoa Project BLM Prospecting Permits *(continued)*

Serial Number	Township and Range	Sections and Descriptions		BLM Approval Date (dd-mm-yyyy)	Acreage*
123690	Township 23 South, Range 32 East, NMPM	Section 24: Section 25: Section 26: Section 27:	All lands All lands N2 N2	3/1/2010	1,920
123691	Township 23 South, Range 32 East, NMPM	Section 1: Section 3: Section 4: Section 5: Section 6:	SW4, W2SE4 Lots 1–4, SE4NW4, S2NE4, S2 Lots 1–4, S2NW4, SW4NE4, S2 Lots 1–4, S2N2, S2 Lot 7	3/1/2010	2,165
	Township 22 South, Range 32 East, NMPM	Section 30:	Lot 4		
123692	Township 23 South, Range 32 East, NMPM	Section 6: Section 8: Section 9: Section 10:	Lots 1–6, SE4NW4, S2NE4, E2SW4, SE4 All lands All lands All lands	3/1/2010	2,536
123693	Township 23 South, Range 32 East, NMPM	Section 12: Section 13: Section 22: Section 23:	W2, W2E2 All lands All lands All lands	3/1/2010	2,400
123694	Township 22 South, Range 32 East, NMPM	Section 28: Section 29: Section 30: Section 33:	All lands All lands Lots 1–3, E2W2, E2 All lands	3/1/2010	2,535
124371	Township 22 South, Range 33 East, NMPM	Section 29:	S2	4/1/2011	320
124371	Township 22 South, Range 32 East, NMPM	Section 19: Section 20: Section 21: Section 22:	Lots 3–4, E2SW4, SE4 S2 All lands All lands	4/1/2011	1,930
124372	Township 22 South, Range 32 East, NMPM	Section 23: Section 24: Section 25: Section 26: Section 27:	All lands S2 All lands All lands N2	4/1/2011	2,560
124373	Township 22 South, Range 32 East, NMPM	Section 27: Section 31: Section 34: Section 35:	S2 Lots 1–4, E2W2, E2 All lands All lands	4/1/2011	2,261
124374	Township 22 South, Range 31 East, NMPM	Section 24: Section 25: Section 26:	E2 SW4, E2 S2NW4, S2	4/1/2011	1,200
124375	Township 22 South, Range 31 East, NMPM	Section 35:	All lands	4/1/2011	640
124375	Township 23 South, Range 31 East, NMPM	Section 1: Section 11: Section 12:	Lots 1–4, S2N2, S2 N2NE4 N2NW4, SE4NW4, E2	4/1/2011	1,159
124375	Township 23 South, Range 32 East, NMPM	Section 1:	Lots 2–4, SW4NE4, S2NW4	4/1/2011	240

Table 4-1. Ochoa Project BLM Prospecting Permits *(concluded)*

Serial Number	Township and Range	Sections and Descriptions		BLM Approval Date (dd-mm-yyyy)	Acreage*
124376	Township 23 South, Range 32 East, NMPM	Section 7: Section 11: Section 14: Section 15:	Lots 1–4, E2W2, E2 All lands All lands N2	4/1/2011	2,264
124377	Township 23 South, Range 32 East, NMPM	Section 15: Section 17: Section 18: Section 20: Section 21:	S2 All lands Lots 1–2, E2NW4, E2 N2, SE4 S2, NW4, W2NE4	4/1/2011	2,492
124378	Township 23 South, Range 32 East, NMPM	Section 26: Section 27: Section 28: Section 34: Section 35:	S2 S2 N2, SE4 N2, SE4 All lands	4/1/2011	2,240
124379	Township 23 South, Range 33 East, NMPM	Section 19: Section 20: Section 29: Section 30:	Lots 1–4, E2W2, E2 All lands All lands Lots 1–4, E2W2, E2	4/1/2011	2,543
124380	Township 23 South, Range 34 East, NMPM	Section 20: Section 27: Section 28: Section 29:	S2, NW4 S2, NW4 All lands S2, NE4	4/1/2011	2,080
124381	Township 23 South, Range 34 East, NMPM	Section 30:	Lots 1–4, E2W2, E2	4/1/2011	640
124381	Township 24 South, Range 32 East, NMPM	Section 1: Section 12:	Lots 1–4, S2N2, S2 N2	4/1/2011	960
124381	Township 24 South, Range 33 East, NMPM	Section 35:	All lands	4/1/2011	633
124382	Township 24 South, Range 34 East, NMPM	Section 30: Section 31:	E2, NE4SW4, SE4NW4 Lots 1–4, E2W2	4/1/2011	719
124382	Township 25 South, Range 33 East, NMPM	Section 1: Section 3:	Lots 1–4, S2N2,S2 Lots 1–4, S2N2, S2	4/1/2011	1,280
124383	Township 25 South, Range 33 East, NMPM	Section 10: Section 11: Section 12:	NE4 All lands W2, NE4, N2SE4	4/1/2011	1,361
124383	Township 25 South, Range 34 East, NMPM	Section 6: Section 7:	Lots 3–7, SE4NW4,E2SW4 Lot 1 and 2	4/1/2011	397
TOTALS					61,983

NMPM = New Mexico Principal Meridian

*Acreage has been rounded to the nearest acre, discrepancies may occur due to rounding.

Table 4-2. Ochoa Project State of New Mexico Leases

Serial Number	Township and Range	Sections and Descriptions		New Mexico Approval Date (dd-mm-yyyy)	Acreage*
HP-0030	Township 22 South, Range 32 East, NMPM	Section 32		5/24/2010	640
HP-0031	Township 22 South, Range 32 East, NMPM	Section 36		5/24/2010	640
HP-0031	Township 23 South, Range 32 East, NMPM	Section 1: Section 12:	Lot 1, E2SE4, SE4NE4 E2E2	5/24/2010	320
HP-0032	Township 23 South, Range 32 East, NMPM	Section 3: Section 4:	SW4NW4 SE4NE4	5/24/2010	80
HP-0033	Township 23 South, Range 32 East, NMPM	Section 2:	Lots 1–4, S2, S2N2	5/24/2010	639
HP-0034	Township 23 South, Range 32 East, NMPM	Section 16:	All lands	5/24/2010	640
HP-0035	Township 23 South, Range 32 East, NMPM	Section 21:	SE4NE4	5/24/2010	40
HP-0036	Township 22 South, Range 33 East, NMPM	Section 30: Section 31: Section 32: Section 33:	Lots 1-4, E2, E2W2 Lots 1-4, E2, E2W2 All lands All lands	5/24/2010	2,533
HP-0037	Township 23 South, Range 33 East, NMPM	Section 2: Section 3: Section 10:	Lots 1–4, S2, S2N2 Lots 1–4, S2, S2N2 All lands	5/24/2010	1,918
HP-0038	Township 23 South, Range 33 East, NMPM	Section 12:	All lands	5/24/2010	640
HP-0039	Township 23 South, Range 33 East, NMPM	Section 15: Section 16: Section 17: Section 18:	All lands All lands E2, E2NW4, SW4 Lots 1–4, E2, E2W2	5/24/2010	2,471
HP-0040	Township 23 South, Range 33 East, NMPM	Section 22: Section 27: Section 33: Section 34:	All lands All lands All lands All lands	5/24/2010	2,560
HP-0041	Township 23 South, Range 33 East, NMPM	Section 35: Section 36:	All lands All lands	5/24/2010	1,280
HP-0041	Township 23 South, Range 34 East, NMPM	Section 31: Section 32:	Lots 1–4, E2, E2W2 All	5/24/2010	1,275
HP-0042	Township 24 South, Range 33 East, NMPM	Section 1: Section 2: Section 3:	Lots 1–4, S2, S2N2 Lots 1–4, S2, S2N2 Lots 1–4, S2, S2N2	5/24/2010	1,919
HP-0042	Township 24 South, Range 34 East, NMPM	Section 6:	Lots 1–7, SE4, S2NE4, E2SW4, SE4NW4	5/24/2010	636
HP-0043	Township 23 South, Range 33 East, NMPM	Section 32:	All lands	5/24/2010	640
HP-0043	Township 24 South, Range 33 East, NMPM	Section 4: Section 5: Section 8:	Lots 1–4, S2, S2N2 Lots 1–4, S2, S2N2 All lands	5/24/2010	1,919
HP-0044	Township 23 South, Range 32 East, NMPM	Section 36:	All lands	5/24/2010	640

Table 4-2. Ochoa Project State of New Mexico Leases *(concluded)*

Serial Number	Township and Range	Sections and Descriptions		New Mexico Approval Date (dd-mm-yyyy)	Acreage*
HP-0044	Township 23 South, Range 33 East, NMPM	Section 31:	Lots 1–4, E2, E2W2	5/24/2010	632
HP-0044	Township 24 South, Range 33 East, NMPM	Section 6: Section 7:	Lots 1–7, SE4, S2NE4, E2SW4, SE4NW4 Lots 1–4, E2, E2W2	5/24/2010	1268
HP-0045	Township 24 South, Range 33 East, NMPM	Section 9: Section 10: Section 15:	All lands All lands All lands	5/24/2010	1,920
HP-0046	Township 23 South, Range 33 East, NMPM	Section 13: Section 14:	N2 N2	5/24/2010	640
HP-0047	Township 24 South, Range 33 East, NMPM	Section 16: Section 17: Section 18:	All All Lots 1–4, E2, E2W2	1/15/2013	1,914
TOTALS:					27,804

*Acreage has been rounded to the nearest acre, discrepancies may occur due to rounding.

Table 4-3. Ochoa Project Pending BLM Permits

Serial Number	Township and Range	Sections and Descriptions		Acreage*
122278	Township 23 South, Range 36 East, NMPM	Section 29: Section 30: Section 31:	All lands Lots 1–4, E2, E2W2 Lots 1–4, E2W2	1,591
122279	Township 24 South, Range 36 East, NMPM	Section 6: Section 7: Section 17: Section 18: Section 19:	Lots 1–5, S2NE4, SE4NW4, SE4 E2 S2SE4, S2NW4, SW4 Lots 1–2, E2NW4, NE4 Lots 1–4, E2W2, E2	2,081
122280	Township 24 South, Range 36 East, NMPM	Section 20: Section 28: Section 29: Section 30: Section 31: Section 33:	All lands N2NW4,E2NE4,E2SE4 NW4NW4,S2SW4 Lots 1–4, E2W2, SE, W2NE4, NE4NE4 Lots 1, 2, E2NW4, NE4 S2SE4	2,006
122281	Township 25 South, Range 36 East, NMPM	Section 4: Section 5: Section 6: Section 7:	Lots 1–4, S2N2, S2 Lots 1–4, S2N2, S2 Lots 6–7, E2SW4, SE4 Lots 1–4, E2W2, NE4, N2SE4	2,165
122282	Township 25 South, Range 36 East, NMPM	Section 8: Section 9:	All lands All lands	1,280
129927	Township 24 South, Range35 East, NMPM	Section 1: Section 11: Section 12: Section 13: Section 14:	Lots 2–4, S2NW4, SW4NE4, W2SE4, SW4 NE4NE4 All All E2, SW4, S2NW4, NE4NW4	2,400
129928	Township 24 South, Range 36 East, NMPM	Section 9: Section 21:	E2 All	960
TOTALS:				12,484

*Acreage has been rounded to the nearest acre, discrepancies may occur due to rounding.

- Land location description
- Status of the state lease or federal prospecting permit
- Mineral rights
- Related gas and oil leases
- Surface rights and patents
- Mortgages, liens, and other surface encumbrances
- Surface easements and ROW
- Agricultural and business leases
- Royalties

Completed title opinions are listed in Table 4-4.

Table 4-4. Title Opinions

Agency	Number of Title Opinions	Date
New Mexico State Land Office (NMSLO)	10	March 2012
	7	May 2012
	1	April 2013
US Bureau of Land Management (BLM)	13	March 2012
	26	September 2012

All BLM prospecting permits and NMSLO leases have been recorded at the Lea or Eddy County Clerk's Office and certified as on file at the BLM for the BLM permits and on file at the NMSLO for the State Leases.

4.2 Agreements and Royalties

ICP will be required to pay royalties on production from NMSLO leases and BLM permits. Royalty rates are detailed in Table 4-5.

Table 4-5. Anticipated Royalty Schedule

Agency	Royalty
New Mexico State Land Office (NMSLO)	$8.00 per acre, or 2.5% of the gross value of production, whichever is greater
US Bureau of Land Management (BLM)	$3.00 per acre, or 2% of the gross value at the point of shipment to market, whichever is greater

A minimum advance royalty payment of $8.00 per acre is payable to the state of New Mexico Commissioner of Public Lands on the 18 state leases. Once the Ochoa Project comes into production, minimum royalties of $8.00 per acre or 2.5% of the gross value of production after processing, whichever is greater, will be owed on the state leases. In addition, once the Ochoa Project comes into production, and no later than 6 years after obtaining the federal BLM PRLs, minimum royalty payments of $3.00 per acre or 2% of the gross value at the point of shipment to market, whichever is greater, are expected to be imposed on the federal BLM PRLs.

Production during the mining is expected to be 45% from NMSLO leases and 55% from BLM permits. ICP has provided a weighted average royalty rate of 2.225% to all production.

Gross profit royalties (GPRs) totaling 3% are payable for a term of 25 years after production first reaches 50%. ICP may acquire, at its option, up to one-half of the GPRs at a price of $3,000,000 per 0.5% royalty interest. Payments due may be deferred under certain conditions until any initial project financing for the Project has been repaid or other terms of the project financing have been satisfied.

An additional private royalty of $1.00/t of polyhalite mine applies to the first 1,000,000 t of $0.50/t thereafter is also payable on the Project pursuant to an agreement with a third party.

4.3 Existing Land Use

The combined population of Lea and Eddy Counties is 119,575, according to the US Census Bureau's 2011 report. The town of Jal, with a population of 2,074, is the nearest community to the Ochoa Project site and is located 22 miles southeast of ICP's land holdings on SH 128. While food, fuel, and limited services are available in Jal, heavy equipment, industrial supplies, and mining-support services are available in Carlsbad, Hobbs, and Albuquerque, New Mexico. Experienced labor for construction, mining, and processing operations is available in nearby communities.

There are active and plugged gas and oil wells within the limits of the Project area, along with roads, power lines, and pipelines associated with oilfield development. Existing infrastructure includes SH 128 and a number of dirt roads for vehicle access to the oil wells. A high-voltage power line is located near the southern edge of the Ochoa Project property, and Xcel supplies electric power. Natural gas transmission pipelines cross the project property.

The Ochoa Project is located in the Pecos Valley section of the southern Great Plains Physiographic Province. The climate of the area is characterized as a high-plains desert environment. The surface consists of relatively flat terrain with minor arroyos and low-quality, semi-arid rangeland. Vegetation is primarily mesquite, shinnery oak, and coarse grasses. Topsoil is caliche rubble and wind-blown sand. The project area is sparsely vegetated, and no cultivation is present, as shown on Figure 4-2.

Figure 4-2. Typical Terrain and Vegetation of the Ochoa Project Land

Cattle grazing occurs throughout most of the BLM permitted and NMSLO leased areas in the project vicinity. The Ochoa Project process plant and shaft are within the Diamond and Half Inc. grazing lease, which has 3,685 permitted animal unit months (AUMs).

4.4 Adjacent Owners and Tenants

Adjacent land owners consist of BLM, the state of New Mexico, and private entities. Landowners adjacent to the ICP processing facilities are shown on Figure 4-3. Grazing leases on and adjacent to ICP processing facilities are shown on Figure 4-4.

4.5 Gas and Oil Wells

Lea County is an active gas and oil exploration and production area. The BLM and state of New Mexico minerals may be divided among development companies. This can result in the fluid minerals being leased to one company and the solid minerals being leased to a different company. This is the case with the ICP BLM and NMSLO leases. ICP holds potash leases while gas and oil companies hold fluid minerals leases. Permitted, active, and plugged gas and oil wells on the Property are shown on Figure 4-5.

ICP is working with gas and oil operators to develop a Memorandum of Understanding (MOU) with each operator to formalize relationships so that operating and planned gas and oil wells can be accommodated within the project mine plan. ICP has obtained MOUs with a number of gas and oil companies.

4.6 Other Land Agreements

The loadout facility will be located on a combination of private, state, and federal lands. The bulk of the loadout facility will be on private land, and ICP has negotiated lease options with land owners. The proposed haul road alignment from SH 128 into the loadout facility is owned by New Mexico, the BLM, and private entities. ICP has negotiated ROW options for the haul road with private entities. ROWs from the BLM are pending results of the EIS. New Mexico state ROWs will be applied for when detailed road design is complete. Table 4-6 lists the other land agreements.

The water supply pipeline, which will be approximately 12 miles, will be located on private, state, and BLM lands. ICP is negotiating ROW options with the private land owners. ROWs from the BLM and NMSLO are pending the issuance of the ROD.

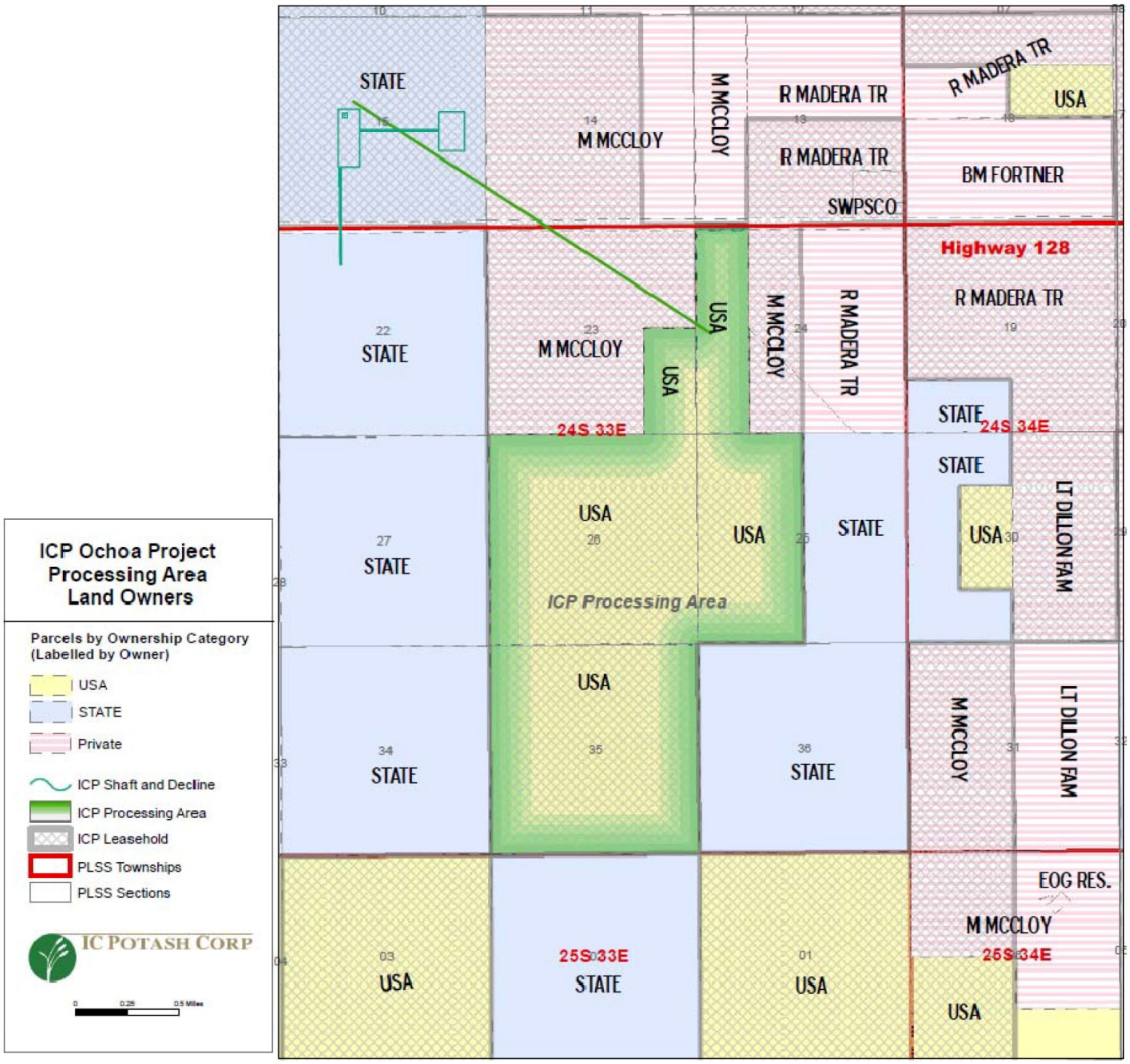


Figure 4-3. ICP Ochoa Project Land Owners

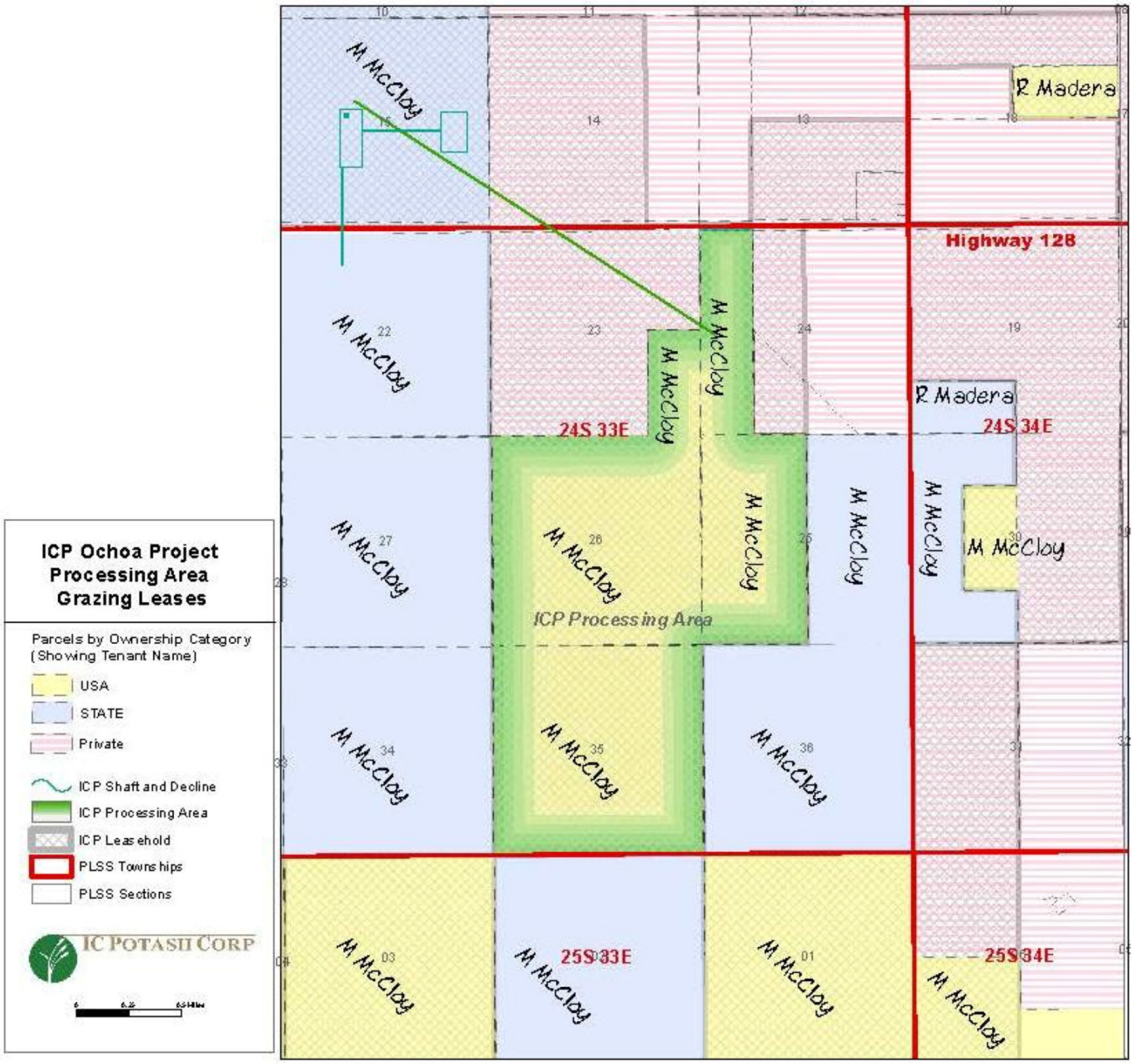


Figure 4-4. Grazing Leases

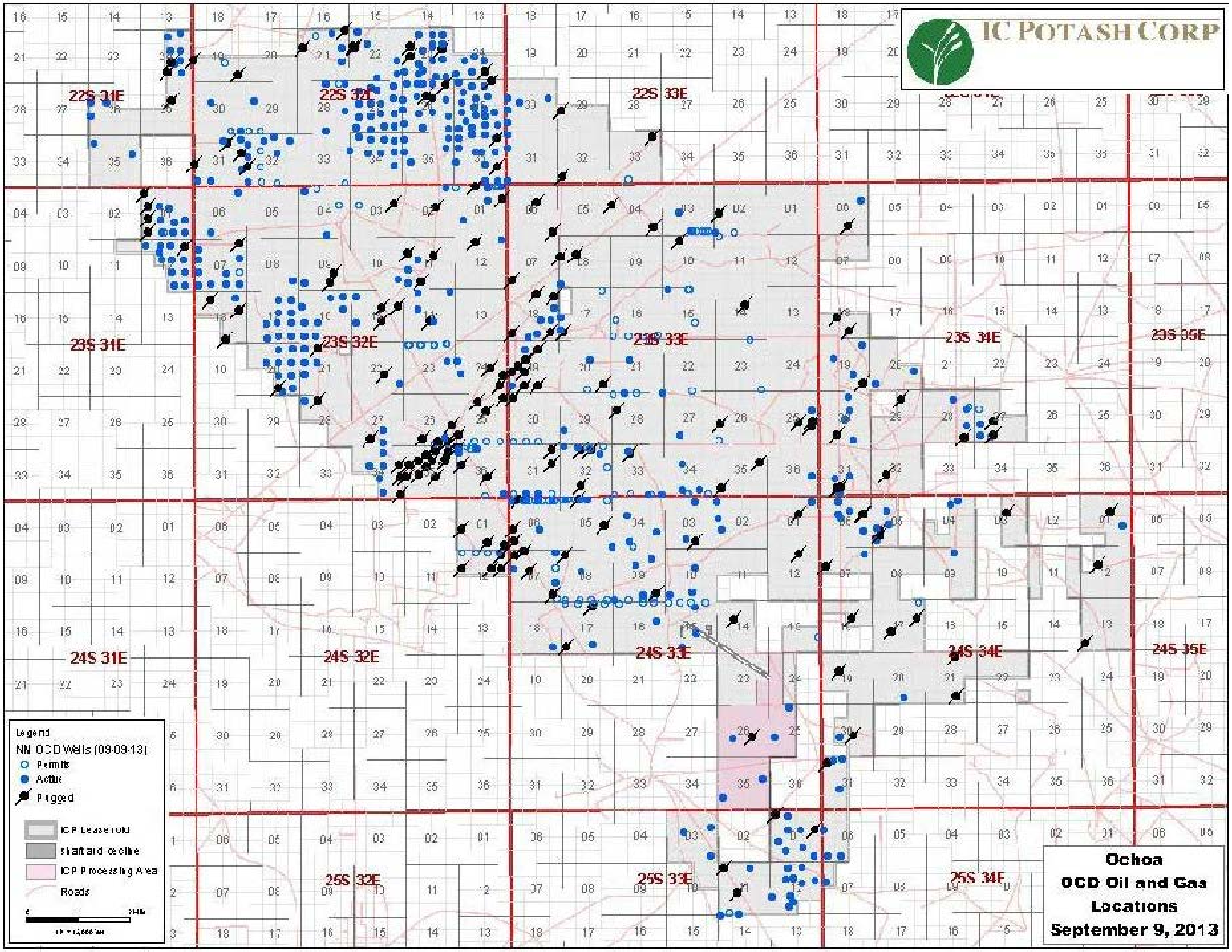


Figure 4-5. Gas and Oil Wells at the Ochoa Project 50-Year Mine Plan

Table 4-6. Landowner and ROW Agreements

Land Owner	Type of Agreement	Date	Terms
Shaft Site			
State of New Mexico	Surface use	May 24, 2010	Surface use allowed by Mining lease Sections 9, 10, and 15 of T24S, R33E 10-year initial term
Processing Site			
BLM	ROW	Pending	Will be granted as part of the EIS ROD Sections 26 and 30 and parts of Sections 24 and 25 of T24S, R33E
Jal Loadout			
Johnny Chapman	Option for Easement	June 25, 2013	Option for road easement through: Section 10, to the easterly most 100 ft for a distance of 500 north of the south line in T25S, R36E Section 15, all that portion lying northeasterly of the ROW for SH128 3-year extendable term
Jal Public Library Fund	Option to Lease	July 22, 2013	Option to lease the following lands for the Jal loadout: Section 25, T24S, R36E Section 1, N2N2, T25S, R36E Section 31, SW4 and all that portion of the SE4 lying west of the TNMR ROW in T24S, R36E 5-year term
Christine Pruett	Option to Lease	June 6, 2013	Option to lease the following lands for the Jal loadout: Section 19, that portion of the SE4 lying west of the TNMR ROW in T24S, R37E Section 30, and that portion of the NE4 lying west of the Texas-New Mexico ROW in T24S, R37E 5-year term
RRR Land & Cattle Company LLC	Option to Lease	June 5, 2013	Option to lease the following lands for the Jal loadout: Section 19, SW4, T24S, R37E Section 30, SW4 and that portion of the SE4 lying west of the TNMR ROW in T24S, R37E Section 31, NW4 and that portion of the NE4 lying west of the TNMR ROW in T24S, R37E Section 24, SE4SE4 in T24S, R36E 5-year term
State of New Mexico	ROW	Pending	Will apply for ROWs after detailed engineering
BLM	ROW	Pending	Will be granted as part of the EIS ROD
Water Well Field and Pipeline			
Bert Madera	Option for Easement	October 30, 2013	Option for pipeline easement for the following lands: Section 13, S2S2 of T24S, R34E Section 24, NW4NW4 of T24S, R34E Section 18, S2S2 of T24S, R35E Section 17, S2SW4, S2SE4SE4 of T24S, R35E 3-year extendable term
Jal Public Library Fund	Option for Lease/ Intention to	Pending	Intention to Grant option to lease: Sections 11, 12, 13, and 14 in T24S, R35E
State of New Mexico	ROW	Pending	Will apply for ROWs after detailed engineering Section 2 of T24S, R35E

T = Township, R = Range, SH = State Highway

ITEM 5: ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The material presented herein references and is extracted in part from Gustavson's (2011d) December 30, 2011, NI 43-101 TR and its April 2012 PFS (2012), with additions and updates from ICP's *Draft Valuable Deposit Determination Report*, dated July 2013 and SNCL's FS (2014).

5.1 Access

The Property is readily accessible from New Mexico SH128 and an extensive network of gravel roads. The Property is traversed by Lea County Road 2 and numerous two-track trails and primitive roads. The site's administrative facilities and processing plant site will be accessed directly from SH128 via approximately 2,170 ft of two-lane, chip-sealed roadway, with an acceleration lane and possibly a turn lane constructed on SH128. The main shaft site will be accessed from SH128 via Brininstool Road by a two-lane gravel access roadway approximately 760 ft long.

The Property is located in Lea County, New Mexico, approximately 8 miles east of the Eddy County line. Airports are located near Carlsbad (Eddy County), approximately 60 miles west via SH128, and at Hobbs, New Mexico (Lea County), via SH128 and SH18, located about 70 highway miles north-northeast of the plant site. Both airports provide commercial and general aviation services.

The Jal loadout site is approximately 22 miles east of the plant site and north of the community of Jal, New Mexico. The loadout will be located near the existing TNMR line running north-south through Jal and connecting to the Union Pacific Railroad near Monahans, Texas. Highway access will be via Phillips Hill Road off SH18, which connects to SH128 in Jal. An industrial spur track connection will be made with the TNMR to handle train shipments of SOP.

The mine's main shaft site is located in Sections 14 and 15, T24S/R33E. The processing plant, administrative facilities, dry stack tailings, evaporation ponds, and slope portal are located in parts of Sections 23, 24, 25, 26, and 35, R24S/R33E. The Jal loadout site will be sited in Sections 24, 25, and 36, T24S/R36E and Sections 19 and 31, T24S/R37E.

5.2 Climate

The climate in southeastern New Mexico is typical of a high plains semi-arid desert environment, with generally mild temperatures and low precipitation and humidity. The prevailing winds are from the southeast in the summer and from the west in winter. Winter temperatures range from lows of –20 degrees Fahrenheit (°F) to highs of 50°F. Summer daytime temperatures are typically above 90°F with nighttime lows in the 70°F range. The average precipitation is about 13 inches per year, with about half of which comes from thunderstorms from June through September. Climate should not affect year-round operations.

5.3 Local Resources and Infrastructure

According to the 2010 Federal Census, the population of Lea County is 64,727 and the population of Eddy County is 53,829. Jal's population is about 2,000 and it is the nearest community to the Project site. Food, fuel, and limited services are available in Jal. Heavy equipment, industrial supplies, and mining support services are available in Carlsbad and Hobbs, New Mexico, and in Midland, Texas. Experienced labor for construction, mining, and processing is available from most all of the southeastern New Mexico and nearby West Texas communities. Many local residents have worked in the underground potash mines and processing plants located between Carlsbad and Hobbs.

A 115-kilovolt (kV) Xcel Energy power line is located near the southern boundary of the Property. Several natural gas transmission pipelines cross the Property. Figure 5-1 illustrates the utilities in the vicinity of the Property.

The area encompassing the Property and surrounding lands has long been an active gas and oil production area with numerous permitted, active, and abandoned gas and oil well sites serviced with a network of interconnection small dirt roads, power lines, and pipelines.

Adequate surface rights have been obtained or are under option to support mining and processing operations on the Property in the form of leases or prospecting permits from the BLM and the NMSLO and private parties.

5.4 Physiography

The Property is located in the Pecos Valley section of the southern Great Plains physiographic province. The surface consists of relatively flat terrain with minor arroyos and low-quality semi-arid rangeland. Top soil is caliche rubble and wind-blown sand with mesquite, shinnery oak, and course grasses as the dominant vegetation. The project area is sparsely vegetated and no cultivation is present. Elevation ranges from 3,100 to 3,750 ft mean sea level (MSL).

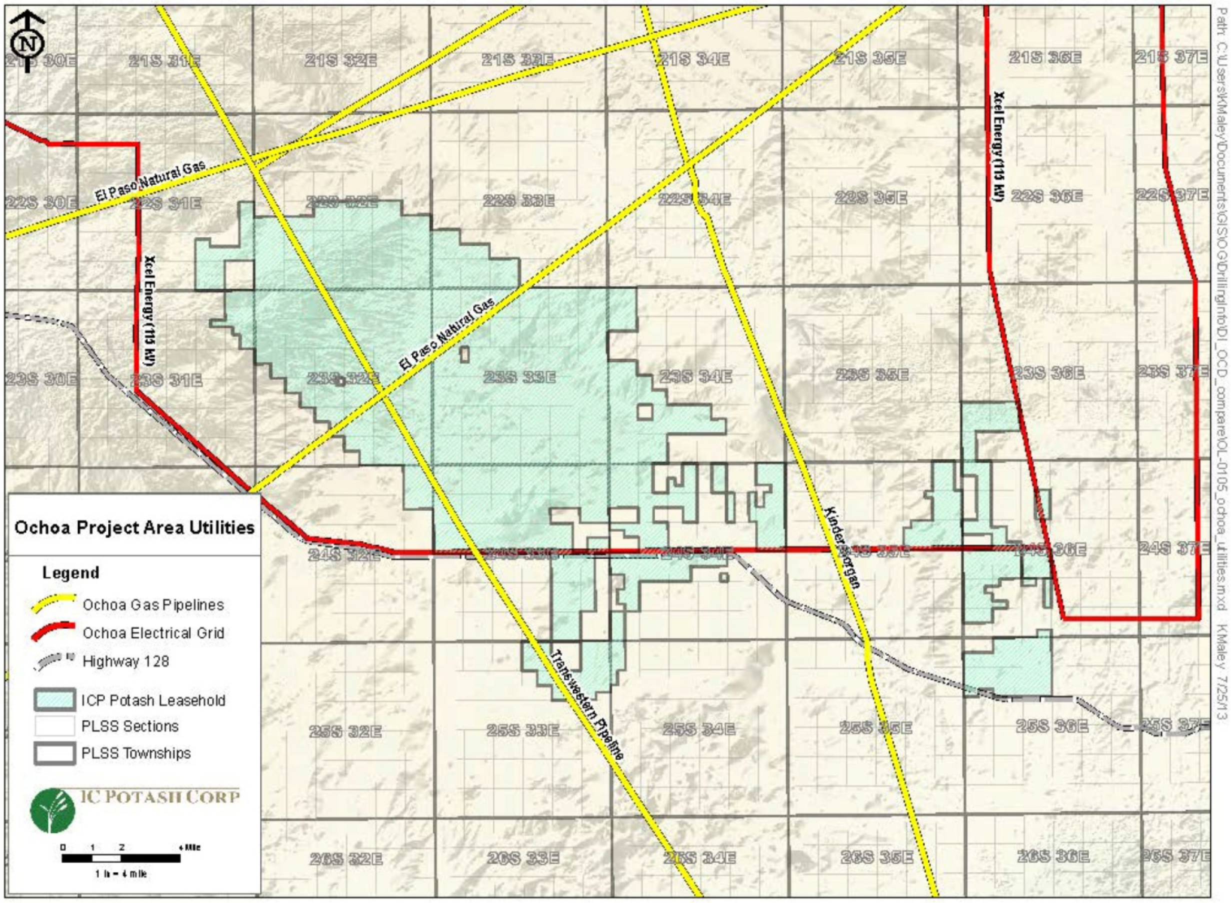


Figure 5-1. Utilities in the Vicinity of the Ochoa Project

ITEM 6: HISTORY

The history section presented herein references and is extracted in part from Gustavson's (2011d) December 30, 2011 NI 43-101 TR and its April 2012 PFS (2012), with additions and updates based on the SNCL January 2014 FS (2014).

The Property does not have any mining history. The Delaware Basin has been explored for hydrocarbons since the early 20th century, but it has not been previously explored for polyhalite. ICP's planned commercial mining and processing operation to produce SOP and potentially other potassium/magnesium fertilizers is based on work that was performed in the 1920s and 1930s by the USBM and PCA. The large-scale development of economic production of potash from potassium chloride and langbeinite in the Carlsbad, New Mexico area significantly reduced interest in the use of polyhalite to produce potassium-based fertilizers. ICP began preliminary polyhalite exploration in 2008 when they applied for exploration permits and initiated a scoping study. That study was prepared by Micon (2008, 2009) and it indicated that the Property had good potential for a sizeable polyhalite deposit.

The Carlsbad, New Mexico, potash deposits that were amenable to economic extraction and processing were identified in 1925 through cuttings from an oil well being drilled near Carlsbad that was being drilled by Snowden & McSweeney Company. After additional exploration activities, the deposits in southeastern New Mexico were established as the only ones in the USA that could be mined by conventional underground mining techniques. At the peak of Permian Basin potash production, there were seven mining companies in operation. Today, only two companies remain in operation in the area: Intrepid Potash, Inc. and Mosaic Potash Carlsbad, Inc.

In the 1930s and 1940s, the USBM (1930a and b, 1933, 1944) was tasked with performing scientific and engineering research regarding polyhalite processing to produce SOP. PCA conducted pilot plant testing in the 1950s. This work formed the basis of the process that ICP has developed for commercialization.

ICP validated the USBM and PCA results during the Ochoa Project PFS (Gustavson 2009d) and FS (SNCL 2014) via process testing, verifying, and validating the earlier work, while collecting data regarding equipment design for processing the Ochoa polyhalite.

The following list outlines the major events in the history of the development of the Property to date. For this listing below, ICP refers to Intercontinental Potash Corp. (USA).

- November 2002—IC Potash Corp. ("Company") was incorporated under the Canada Business Corporations Act (formerly Trigon Uranium Corp. and Trigon Exploration Canada Ltd.).
- November 2004—The Company was listed on the TSX Venture Exchange.
- January 2008—Intercontinental Potash Corp. (USA) was incorporated under the laws of Colorado (formerly U.S. Potash Corp.).
- March 2008—Intercontinental Potash Corp, a Canadian company, was incorporated and owns 100% of ICP.
- May 2008—The Company acquired a 50% interest in Intercontinental Potash Corp. (USA) (subsequently diluted to 38%).
- November 2008—The Company posted its first NI 43-101 TR for the Ochoa Property (Micon 2008).

- December 2008—ICP was awarded 16 BLM prospecting permits.
- January 2009—The Company completed a restated NI 43-101 TR for the Property (Micon 2009).
- August 2009—The Company completed an NI 43-101 TR and Preliminary Economic Assessment for the Property (Chemrox and Gustavson 2009).
- November 2009—The Company acquired 100% of Intercontinental Potash Corp. as part of a reverse takeover.
- December 2009—ICP began Phase 1 exploration drilling.
- February 2010— The Company announced results of the Phase 1 exploration drilling.
- March 2010—ICP was awarded five BLM prospecting permits.
- April 2010—Phase 2 exploration drilling began.
- May 2010—ICP was awarded 17 state mining leases.
- September 2010—The Company announced results of the Phase 2 exploration drilling program.
- January 2011—The Company began trading on the OTCQX.
- January 2011—The Company posted an NI 43-101 TR and updated Preliminary Economic Assessment for the Property (Gustavson 2011a).
- January 2011—Phase 2B drilling program began.
- April 2011—ICP was awarded 13 BLM prospecting permits.
- May 2011—The Company announced results of Hazen process test work.
- June 2011—The Company moved its listing to the TSX.
- September 2011—The Company announced ICP had signed an MOU with the BLM for an EIS study on the Property.
- October 2011—The Company announced results of the Phase 2B drilling program.
- October 2011—ICP submitted a Mine Plan of Operations (MPO) to the BLM (Gustavson 2011b).
- November 2011—The Company posted an NI 43-101 TR and updated resource estimate for the Property (Gustavson 2011c).
- December 2011—The Company posted an NI 43-101 TR summarizing the results of the PFS (Gustavson 2011d).
- June 2012—ICP signed its first MOU with a petroleum company.
- August 2012—Phase 3A drilling program began.
- March 2011—ICP filed its PRL application.
- November 2012 and January 2013—Six BLM prospecting permits were relinquished and allowed to expire.
- January 2013—ICP was awarded one state mining lease.
- Summer 2013—ICP conducted pilot plant test work at Hazen and Veolia.
- June 2013—The Company announced results of the Phase 3A drilling program.
- August 2013—The Draft EIS was published and public comment period was completed in September 2013.
- November 2013—ICP updated the MPO to incorporate Phase 3A drilling results.
- January 2014—The Company announced the results of the FS (SNCL 2014).

ITEM 7: GEOLOGIC SETTING AND MINERALIZATION

The geology section presented herein references and is extracted in part from Gustavson's (2011d) December 30, 2011 NI 43-101 TR and its April 2012 PFS, with additions and updates. The geochemistry of the polyhalite ore zone section was extracted in part from ICP's work and the hydrogeology section was extracted in part from INTERA's (2012, 2013a) work that was presented in the January 2014 SNCL Ochoa Project FS.

7.1 Regional Stratigraphy

The Ochoa Project lies at the northeastern margin of the Delaware Basin (Figure 7-1). The Delaware Basin is a structural sub-basin of the larger Permian Basin that dominated the region of southeast New Mexico, west Texas, and northern Mexico from 265 to 230 MYBP. The Permian Basin is an asymmetrical depression formed on top of Precambrian basement rocks. Marine sediments accumulated in the basin throughout the Paleozoic era. The slow collision of the North American and South American crustal plates resulted in tectonic subdivision of the Permian Basin into numerous sub-basins, of which the Delaware and Midland Basins are the largest (Ward, Kendall and Harris 1986). The Delaware Basin has been extensively studied, in part because of extensive gas and oil exploration, but also because of the WIPP in the northern part of the basin. WIPP is a geologic repository to permanently dispose of radioactive waste. The geology of the WIPP site was studied extensively prior to it being certified in 1998 by the US Department of Energy (USDOE 2013).

7.2 Local Geology

The sedimentary sequence of the Delaware Basin is composed of deep water siliciclastics, shelf carbonates, marginal marine evaporites, and terrestrial red beds. The deep water siliciclastics and shelf carbonates occur well below the horizon of interest and are not discussed further. Extensive and thick evaporite deposits occur throughout the late-Permian period (Ochoan-age) rocks within the basin. The Upper Permian Series consists of Ochoan-age sedimentary deposits, specifically the Castile, Salado, and Rustler Formations (Figure 7-2). Collectively, the Castile, Salado, and Rustler evaporite-bearing formations are more than 4,000 ft thick in the Ochoa Project area (after Jones 1972).

7.3 Lithology

7.3.1 Castile Formation

The Castile Formation is the oldest evaporite cycle of the Ochoan series in the Delaware Basin, and is composed largely of anhydrite, light and dark laminae with halite (NaCl), and limestone. The calcareous component increases with depth. In outcrop, anhydrite alters to gypsum (King 1948).

7.3.2 Salado Formation

The Salado Formation consists of cyclic anhydrite ($CaSO_4$), halite, and clay deposits. The Salado Formation is divided into three units—the upper, lower, and middle—in the northern portion of the Delaware Basin. Potassium minerals in the McNutt Member of the Salado

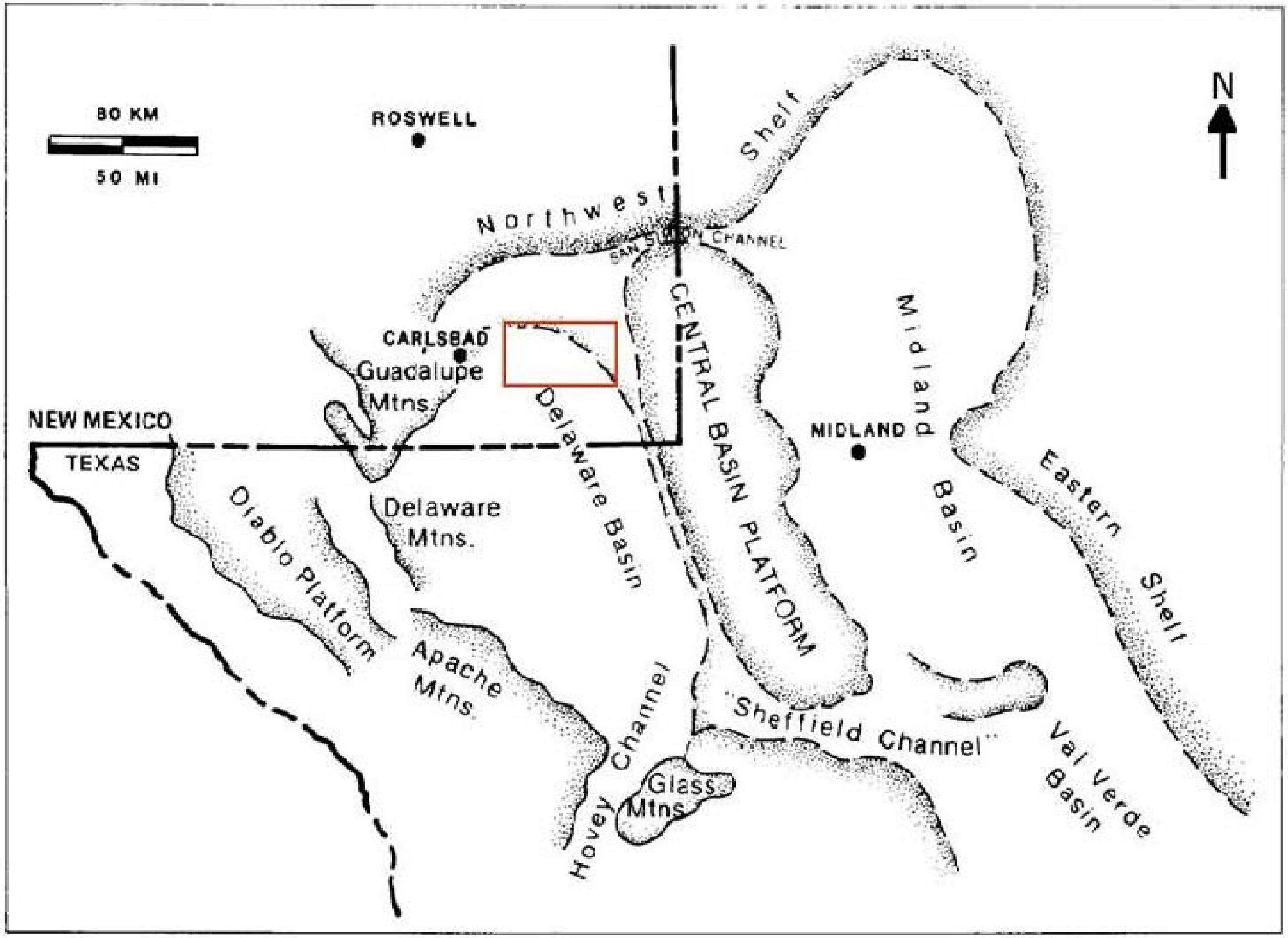


Figure 7-1. Delaware Basin, Ochoa Project Boundary in Red (modified from Ward, Kendall and Harrs 1986)

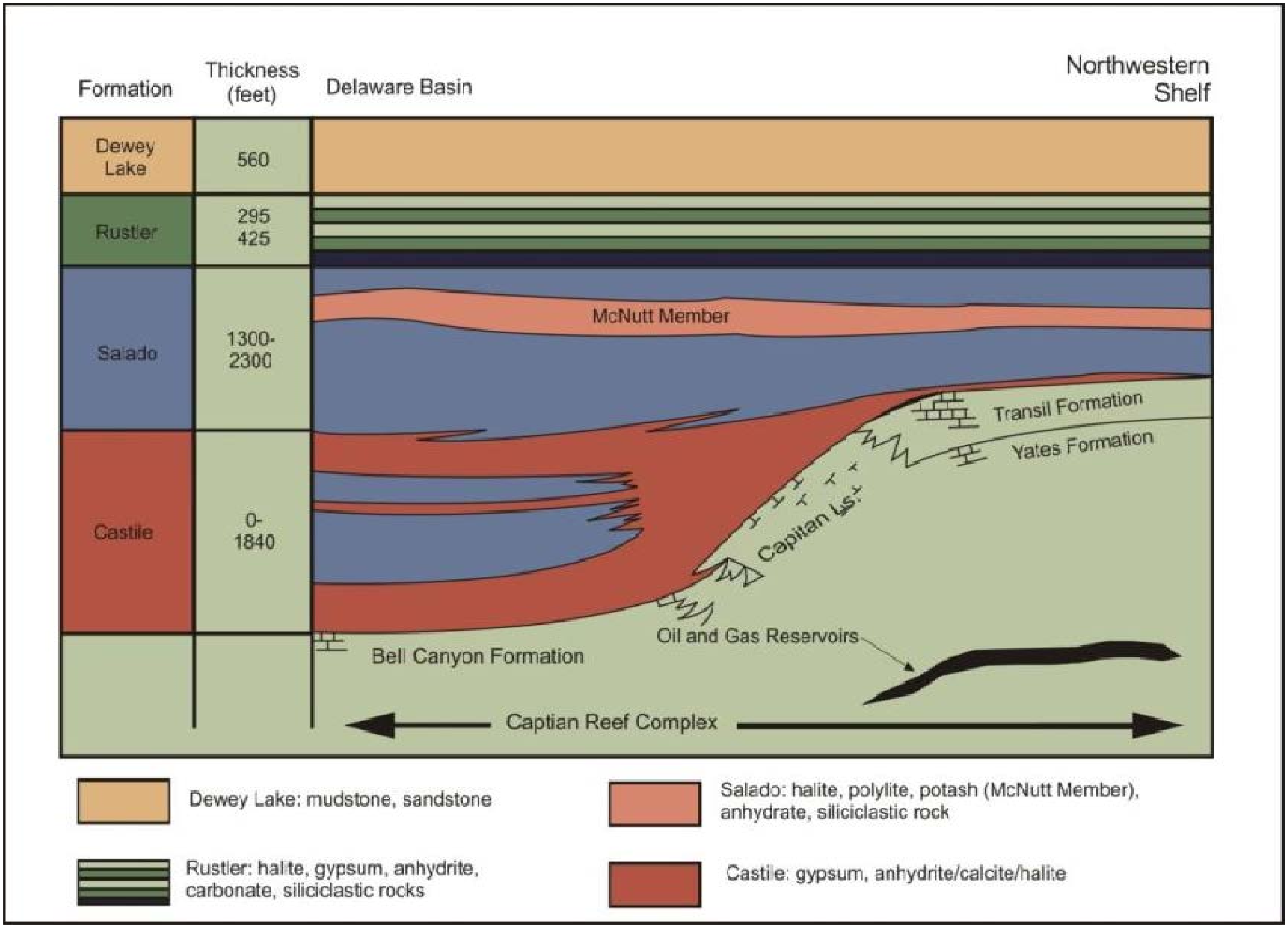


Figure 7-2. Diagrammatic North-South Cross Section and Stratigraphic Relationships of the Northern Edge of the Delaware Basin, Southeastern New Mexico (after Austin 1980 and Jones 1972)

Formation occur as interbeds within the anhydrite and halite stratigraphic units and are mined commercially in and around Carlsbad, New Mexico. Potash occurs in the form of polyhalite (K_2SO_4•$MgSO_4$•$2CaSO_4$•$2H_2O$) in anhydrite, and as sylvite (KCl), langbeinite [($K_2Mg_2(SO_4)_3$], or carnallite ($KMgCl_3 \cdot 6(H_2O)$ in halite.

7.3.3 Rustler Formation

The target horizon of ICP's Ochoa Project is the polyhalite found within the Tamarisk Member of the Rustler Formation (Figure 7-3). The late-Permian Rustler is found in both the Delaware and Midland Basins and on the Central Basin Platform (CBP) that divides them. The Rustler Formation is composed of anhydrite, halite, dolomite, sandy siltstone, and polyhalite (Jones 1972), representing a transitional phase between end-stage marine evaporative and the onset of terrestrial depositional regimes.

There are five recognized members of the Rustler Formation (Powers and Holt 1999), which are, from oldest to youngest, the Los Medaños (Lowenstein 1987), Culebra, Tamarisk, Magenta, and Forty-niner. Polyhalite occurs in the Tamarisk Member of the Rustler Formation.

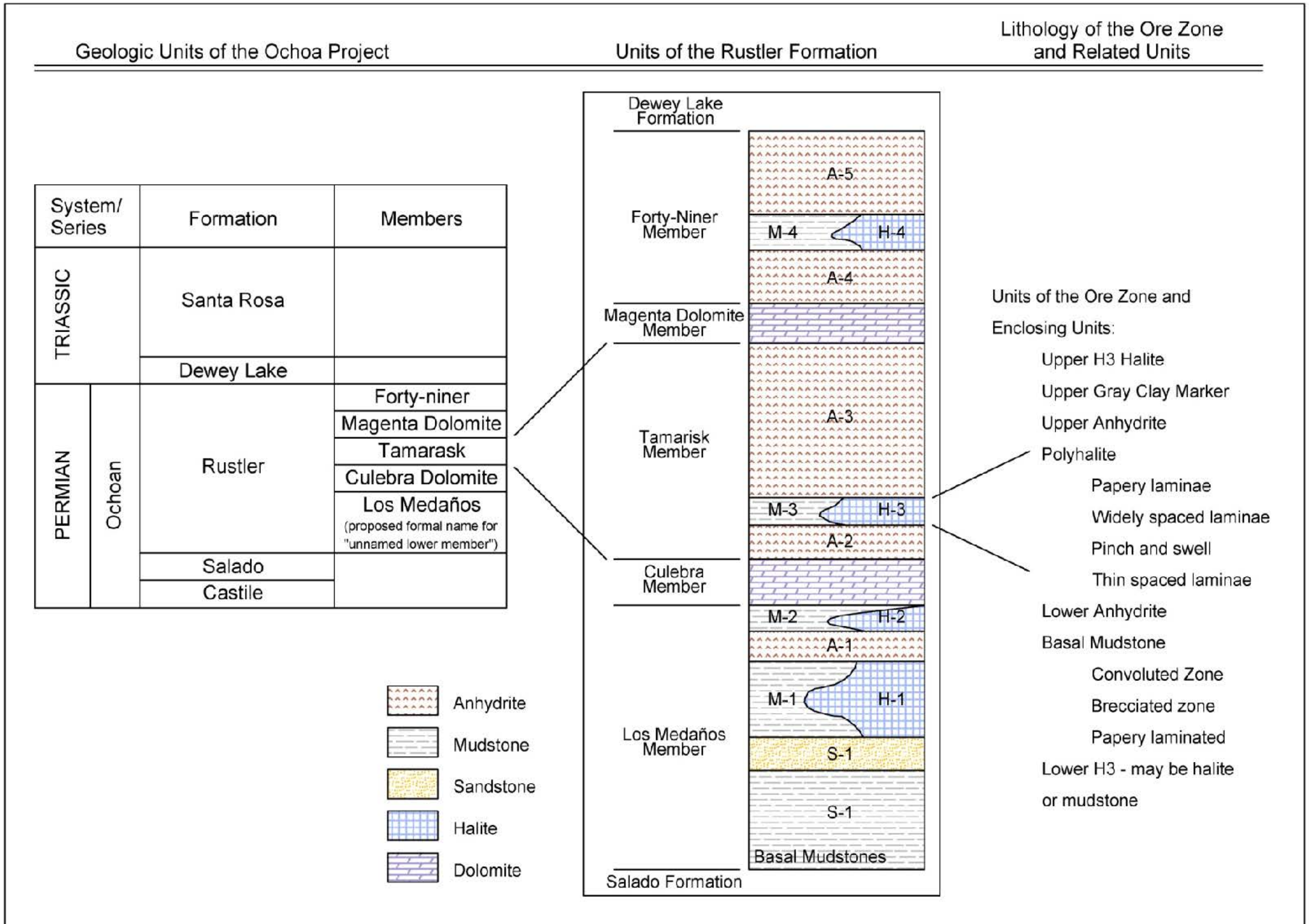
- The Los Medaños Member consists of siliclastics, halitic mudstones and muddy halite, and sulfate minerals, principally anhydrite (Powers and Holt 1999).
- The Culebra Member consists of pinkish gray dolomite.
- The Tamarisk Member is composed of three sub-units: a lower basal anhydrite, a middle halitic mudstone, and an upper anhydrite. Polyhalite occurs within the basal anhydrite. The thickness of the Tamarisk varies principally as a function of the thickness of the middle halite unit.
- The Magenta Member is predominantly dolomite with minor amounts of gypsum.
- The Forty-niner Member has a similar general stratigraphy to the Tamarisk. It is made up of a lower and an upper anhydrite with a middle siltstone.

7.3.4 Dewey Lake Formation

The Dewey Lake Formation is composed of mudstone, siltstone, claystone, and interbedded sandstones consistent with terrestrial red beds. The formation is divided into upper and lower members. The lower Dewey Lake is characterized by gypsum-filled fractures, and the upper Dewey Lake is cemented by carbonate (Beauheim and Holt 1990). It is unconformable over the Rustler.

7.4 Structure

The geology of the Ochoa Project is characterized by a simple structural setting within the Delaware Basin. The stratigraphic section of interest, the Rustler Formation, is present in its entirety throughout the project area. In general, the Ochoa Project overlies a gentle, northwest-southeast oriented downwarped basin that originated in the late-Proterozoic and persisted through the end of the Permian. The early Paleozoic was dominated by shallow water deposition of limestones and clay contrasting with periods of emergence and subaerial erosion. By the Mississippian, the basin was bound to the east by the CBP that separates it from the Midland Basin, perhaps representing reactivation of Precambrian lateral faulting (West Platform Fault Zone) (Keller, Hills, and Djeddit 1980). The basins are ringed by broad limestone shelves



after Powers, D.W. and R.M. Holt (1999). "The Los Medeños Member of the Permian (Ochoan) Rustler Formation," New Mexico Geology, November.

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Figure 7-3. Geologic Units of the Ochoa Project

followed by clastic fill. Beginning in the mid-Permian, the slight lowering of the eustatic sea level and continued restriction within the basin resulted in the formation of back-reef evaporites. By the end of the Permian, with downwarping slowing against the CBP, the basin began filling with fine clastics, then with continental red beds.

The Laramide orogeny uplifted the western edge of the basin on the carbonate shelf. Downfaulting resulted in salt dissolution forming the Salt Basin Graben in the late-Cenozoic (Anderson 1981) defined at the western edge of the basin as Nash Draw and in the east, the San Simon Sink and Swale at the margin of the reef. A Bouguer gravity study confirmed the positive anomaly at the CPB with corresponding steep gradient at the West Platform Fault Zone and negative anomalies at the deepest portions of the Delaware Basin (Keller, Hills, and Djedditt 1980).

7.5 Geochemistry of the Polyhalite Ore Zone

The geochemistry of the Ochoa Project ore zone is laterally consistent throughout the ore zone (see Item 14). Mineralogy was determined using quantitative XRD analysis and consists of a narrow range of minerals—polyhalite, halite, magnesite ($MgCO_3$), and anhydrite. Approximately 5% of the mineral constituents were not specifically identified. The polyhalite zone was determined based on minimum percentage of polyhalite (65%).

Quantitative and semi-quantitative XRF analyses were conducted on the polyhalite horizon from 32 drill cores. The quantitative elemental measurements were used to calculate the mineral compositions by normative mineral calculation as a double check of the XRD analysis.

The semi-quantitative XRF data were used to check for silicates (e.g., quartz, clay) that might be present at levels too low for XRD detection. Silicates can be important to mining and processing because they can be abrasive, influence fluid rheology, and contribute to the fines fraction. Additionally, some elements have toxicity, radiation, oxidation, or other process related characteristics that could affect mining, processing, or waste disposal decisions. Minimal silicates were present and no heavy metals or radioactive elements were found.

Full quantitative analyses were performed for sodium (Na), chlorine (Cl), magnesium (Mg), sulfur (S), potassium (K), and calcium (Ca). The remaining trace elements were analyzed by a semi-quantitative analysis. The results are a hybrid of fully quantitative analysis for the major elements (with error ≈ 1%), and semi-quantitative analysis for the trace elements (with errors ≈ 10%).

XRD and XRF analyses were conducted at H&M Analytical Services, Inc. (H&M) in Allentown, New Jersey.

Polyhalite zone composites were calculated for all bore holes except ICP-095, located approximately 10 miles east of the main mining area, because it did not have a polyhalite zone of sufficient thickness to be included in the resource estimate. The mineralogy results are shown in Table 7-1. Figure 7-4 illustrates these results. As shown in the table and figure, there is little variation in the mineralogy. Results for quantitative analyses of major ions are listed in Table 7-2 and shown on Figure 7-5. Minor element results, as oxides, from Phase 1 and Phase 2 exploration activities are shown in Table 7-3, reported in weight percent (wt-%), and on Figure 7-6. Both silica dioxide (SiO_2) and strontium oxide (SrO) show some variation over a limited range.

Table 7-1. Drill-Hole Assay Composite Mineralogy (wt-%) (from SNCL FS 2014, Table 8.5)

Drill-Hole Number	Anhydrite	Gypsum	Polyhalite	Halite	Magnesite
ICP-001	3.63	0.01	91.7	3.15	8.01
ICP-002	3.60	0.01	83.7	1.98	9.38
ICP-003	4.72	0.01	80.0	3.93	11.31
ICP-005	2.23	0.01	91.7	1.74	4.33
ICP-042	5.57	0.01	85.8	1.67	5.69
ICP-043	3.63	0.01	81.7	5.81	8.25
ICP-045	3.97	0.01	79.4	6.40	8.97
ICP-046	3.57	0.01	86.0	2.91	7.49
ICP-047	7.12	0.01	80.7	2.32	8.02
ICP-048	5.37	0.01	77.8	3.44	12.41
ICP-051	2.72	0.01	79.8	5.02	11.58
ICP-053	2.10	0.01	88.3	1.69	7.96
ICP-056	2.99	0.01	89.5	2.77	4.75
ICP-058	2.79	0.01	80.9-	4.91	12.04
ICP-059	2.05	0.01	84.7	3.58	9.71
ICP-061	1.97	0.01	92.4	1.76	3.88
ICP-062	8.24	0.01	81.6	1.49	7.06
ICP-063	5.40	0.02	80.0	5.60	9.01
ICP-076	7.63	0.01	83.0	0.72	8.69
ICP-078	1.74	0.01	80.7	6.75	10.79
ICP-083	2.48	—	89.0	—	5.67
ICP-084	2.35	—	89.3	—	5.44
ICP-085	3.02	—	85.7	—	8.15
ICP-086	3.83	—	90.1	—	4.45
ICP-087	2.72	—	90.3	—	4.77
ICP-088	3.98	—	88.5	—	—
ICP-089	2.70	—	85.2	—	6.28
ICP-090	1.53	—	91.2	—	4.53
ICP-092	2.52	—	91.1	—	5.10
ICP-093	2.29	—	90.0	—	5.56
ICP-095	—	—	—	—	—
ICP-097	4.67	—	79.0	—	9.25

Note: Table data based on completion of final Phase 3A assays made available after completion of resource geologic modeling (Item 14).

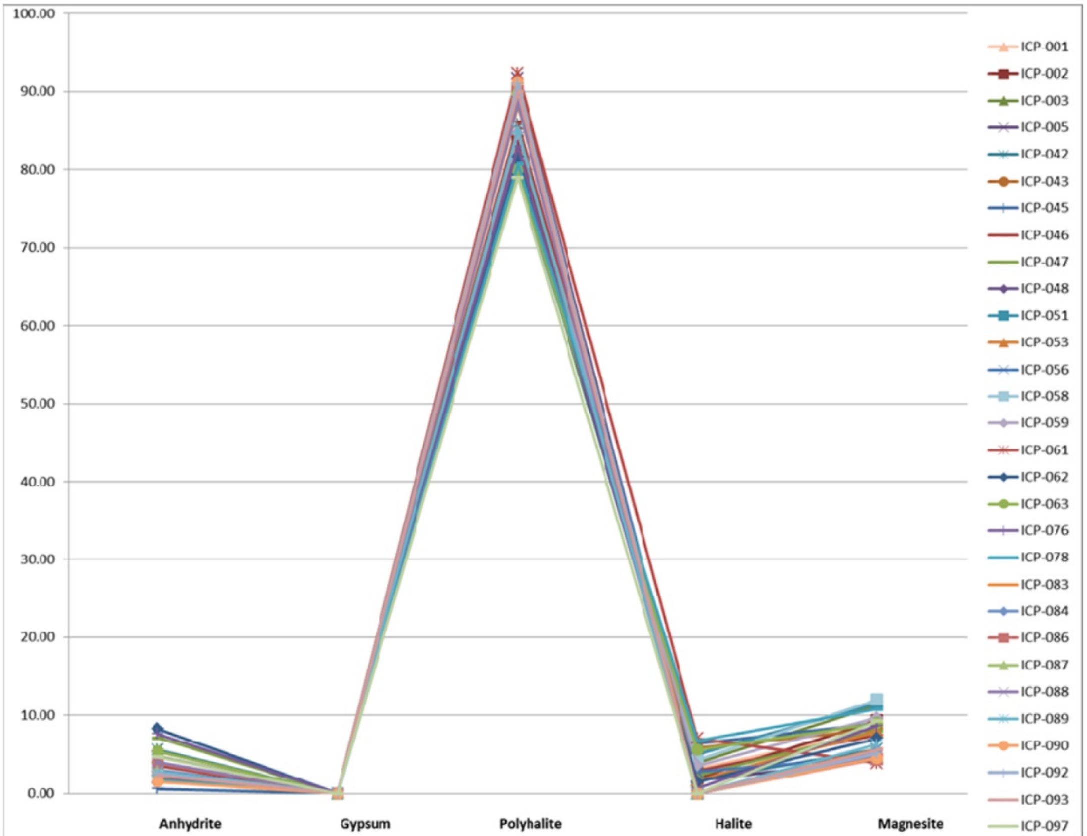


Figure 7-4. Mineralogy of the Polyhalite Zone (from SNCL FS 2013, Figure 8.12)

Table 7-2. Quantitative Composite Results—Polyhalite Target Zone (wt-%) (from SNCL FS 2013, Table 8.6)

Drill-Hole Number	NaCl	MgO	SO_3	K_2O	CaO
ICP-001	3.18	9.46	44.85	13.12	17.12
ICP-002	2.02	10.08	46.45	13.09	17.11
ICP-003	3.98	10.74	45.30	12.45	16.91
ICP-005	1.72	8.19	50.03	14.38	17.92
ICP-042	1.92	8.64	49.26	13.50	18.44
ICP-043	5.94	9.38	45.50	12.73	16.68
ICP-045	6.43	9.66	44.85	12.46	16.52
ICP-046	2.83	9.34	47.85	13.38	17.48
ICP-047	2.36	8.83	47.51	12.63	18.35
ICP-048	5.12	10.86	43.95	12.44	15.96
ICP-051	5.12	10.86	43.95	12.44	15.96
ICP-053	1.71	9.71	48.03	13.80	17.20
ICP-056	4.35	11.15	44.63	12.64	16.19
ICP-058	1.89	8.04	50.22	12.73	18.61
ICP-059	5.80	9.67	45.68	12.50	17.08
ICP-061	0.93	9.72	48.51	12.89	18.61
ICP-062	6.32	10.68	44.01	12.64	15.80
ICP-063	3.72	9.02	45.42	14.53	13.99
ICP-076	4.74	9.16	45.81	14.58	14.03
ICP-078	5.03	10.11	45.20	14.30	13.79
ICP-083	1.93	8.20	47.40	14.40	14.76
ICP-084	3.77	8.75	46.36	14.74	14.17
ICP-085	3.36	9.32	46.28	14.42	14.55
ICP-086	8.18	8.65	44.29	14.07	13.70
ICP-087	4.19	0.00	46.47	14.80	14.21
ICP-088	2.12	9.20	47.12	14.82	14.61
ICP-089	3.27	9.60	46.36	14.59	14.19
ICP-090	8.42	8.61	38.21	12.58	12.90
ICP-092	3.18	9.46	44.85	13.12	17.12
ICP-093	2.02	10.08	46.45	13.09	17.11
ICP-095	3.98	10.74	45.30	12.45	16.91
ICP-097	1.72	8.19	50.03	14.38	17.92

Table 7-3. Semi-Quantitative Minor Element Composite Results—Polyhalite Target Zone (wt-%) (from SNCL FS 2013, Table 8.7)

Drill-Hole Number	Al_2O_3	SiO_2	TiO_2	MnO	Fe_2O_3	CoO	CuO	ZnO	SrO
ICP-001	0.17	1.11	0.01	0.00	0.19	0.00	0.01	0.00	1.04
ICP-002	0.24	1.80	0.01	0.01	0.14	0.01	0.01	0.00	1.15
ICP-003	0.17	1.29	0.01	0.01	0.15	0.01	0.01	0.00	1.53
ICP-005	0.11	0.77	0.00	0.00	0.08	0.00	0.01	0.00	1.26
ICP-042	0.14	0.93	0.00	0.00	0.14	0.00	0.01	0.00	0.96
ICP-043	0.24	1.75	0.01	0.01	0.14	0.01	0.01	0.00	0.90
ICP-045	0.22	1.57	0.01	0.01	0.36	0.01	0.01	0.00	0.94
ICP-046	0.12	0.97	0.00	0.00	0.10	0.01	0.01	0.00	1.29
ICP-047	0.21	1.60	0.02	0.01	0.15	0.01	0.01	0.00	1.09
ICP-048	0.27	2.00	0.02	0.01	0.21	0.00	0.01	0.00	1.16
ICP-051	0.25	1.88	0.02	0.01	0.15	0.00	0.01	0.00	1.08
ICP-053	0.15	1.09	0.00	0.02	0.17	0.00	0.01	0.00	1.28
ICP-056	0.16	0.97	0.00	0.00	0.09	0.00	0.02	0.01	1.11
ICP-058	0.16	1.19	0.00	0.01	0.11	0.00	0.01	0.00	1.35
ICP-059	0.14	1.03	0.00	0.00	0.11	0.01	0.01	0.00	1.33
ICP-061	0.11	0.90	0.00	0.00	0.08	0.00	0.01	0.00	1.17
ICP-062	0.16	1.08	0.00	0.01	0.12	0.00	0.01	0.00	1.05
ICP-063	0.19	1.37	0.01	0.01	0.13	0.00	0.01	0.01	1.30
ICP-076	0.17	1.46	0.00	0.00	0.10	0.01	0.01	0.00	1.19
ICP-078	0.17	1.37	0.01	0.01	0.13	0.01	0.01	0.00	1.13

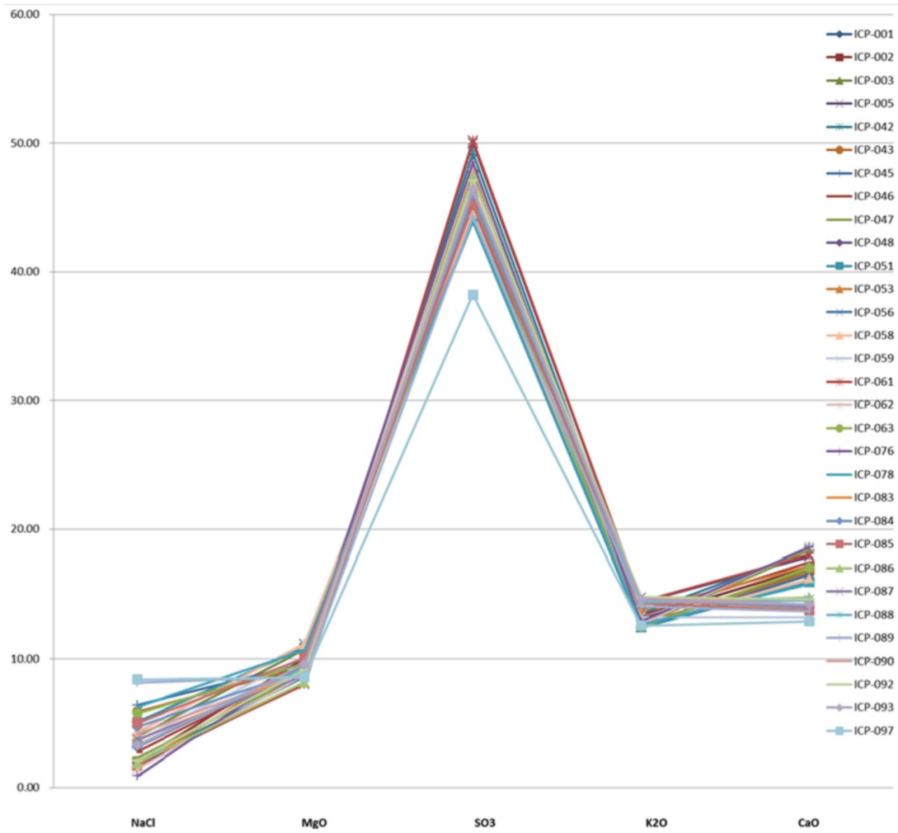


Figure 7-5. Quantitative Elemental Results for the Polyhalite Zone (from SNCL FS 2013, Figure 8.13)

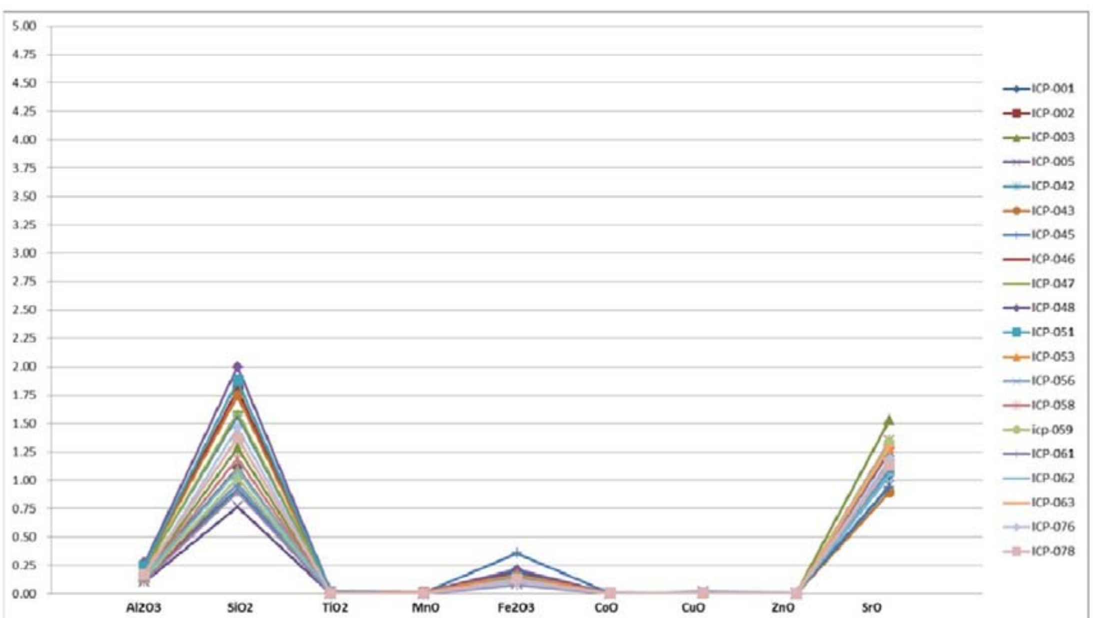


Figure 7-6. Semi-Quantitative Minor Elemental Results for the Polyhalite Zone (from SNCL FS 2013, Figure 8.14)

Although, little variation is shown, these results suggest slight changes in the amount of clay and evaporite minerals. Strontium (Sr) could substitute for Ca in polyhalite, while the SiO_2 could be associated with clay minerals.

7.6 Hydrogeology

Hydrogeologic analysis for the Property is focused on two areas: (1) the water supply for the mining and processing operations, and (2) groundwater inflows at the shaft and slope site. Sources of water for the mine and plant operations were evaluated and the use of highly saline groundwater from the Capitan aquifer was selected as the most viable source. The water supply wells are located approximately 12 miles east of the Project's plant site. Approximately 2,700 gpm of saline water will be needed to support the Project requirements for raw water, RO water, and potable water. Shaft site analysis indicated there are relatively minimal groundwater flows above the halite/anhydrite zones. The following discussion is summarized from INTERA's (2012, 2013a, 2013b) contribution to the FS, Chapters 8.3 and 11.

7.6.1 Regional Hydrogeologic Framework

This section presents information on water availability from the Capitan aquifer including the hydrogeologic framework. The Ochoa Project lies at the northeastern margin of the Delaware Basin as shown in Figure 7-7. The Delaware Basin was an area of subsidence, resulting in deposition of a thick sequence of marine rocks; the study area is underlain by almost 12,000 ft of Permian-age deposits (Figures 7-7 and 7-8) (Bjorklund and Motts 1959). The basin is bounded on the west by the Diablo Plateau, on the north by the Northwestern Shelf, and on the east by the CBP. The Capitan aquifer was formed by the youngest of the Permian shelf-margin complexes developed around the Delaware Basin (Harris and Saller 1999). Hiss (1975) describes the Capitan aquifer as a "lithosome that includes the Capitan and Goat Seep Formations and most or all of the Carlsbad (carbonate) facies of the Artesia Group" from northern Pecos County, Texas, around to the Guadalupe Mountains and the Gilliam and Word Formations in southern Pecos County (Glass Mountains) (Armstrong and McMillion 1961; Hill 1996). Some of the upper San Andres Limestone, which is equivalent to the Vidrio Limestone member of the Word Formation in the Glass Mountains (Armstrong and McMillion 1961; Hill 1996), is included in the Capitan aquifer when it cannot be distinguished from the Goat Seep Limestone or the Carlsbad facies of the Artesia Group (Hiss 1975).

The Capitan Reef Complex is a horseshoe-shaped limestone, dolomite, and sandstone deposit surrounding the Delaware Basin. The complex extends over approximately 200 miles in southeastern New Mexico and western Texas. Geologic formation names vary somewhat from west to east, but the overall structure of the reef complex (i.e., the basin, slope, reef, back-reef, and shelf facies) is consistent throughout. The full depositional history and the development of the structure of the Delaware Basin have been discussed in great detail elsewhere (e.g., Bjorklund and Motts 1959; Ward, Kendall and Harris 1986; Hill 1996). Figure 7-9 illustrates the thickness of the Capitan aquifer. Figure 7-10 shows the depth from ground surface to the top of the Capitan aquifer.

The Capitan aquifer is confined above by the Salado Formation. The Salado Formation is characterized by extremely low hydraulic conductivity that has been measured at between 4×10^{-2} and 4×10^{-3} ft/day using compressed-air injection in a test hole assuming a porosity of 0.001 (reported as 12 and 21 microdarcies [μD]) (Mercer 1983). Rocks of this type are considered to be essentially impermeable (Bear 1972) and were one of the main reasons for

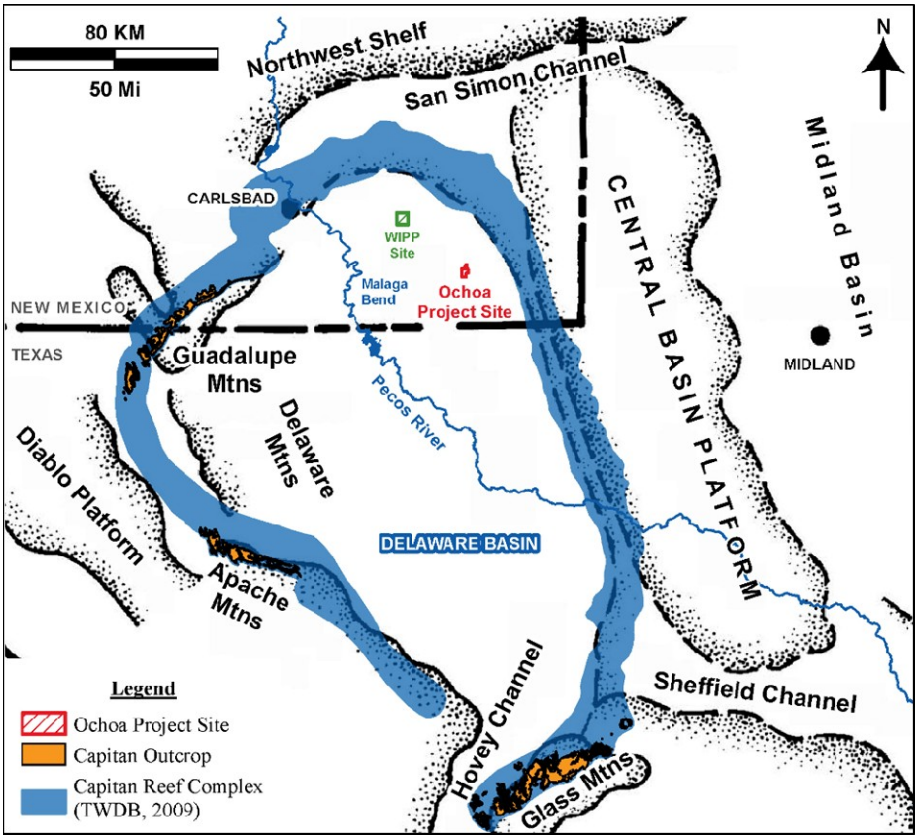


Figure 7-7. The Delaware Basin, Capitan Reef, and Associated Structures

constructing the WIPP in the Salado Formation. Other studies, based on water levels from many existing wells, revealed no hydraulic connection between rocks overlying and underlying the Salado Formation (Hunter 1985). Thus, no vertical communication is expected between the Capitan aquifer and any overlying aquifers, which may occur within the Rustler, Dewey Lake, and Chinle Formations. Alluvial aquifers within the basin that lie above the Salado Formation are also not in communication with the Capitan aquifer, except where the Salado has been eroded by the Pecos River in the vicinity of Carlsbad and the alluvial aquifers are in contact with the Capitan aquifer. As a result, groundwater flow within the Capitan aquifer over nearly its entire extent between Carlsbad and the Glass Mountains is constrained to the associated Capitan Reef Complex formations and, to a limited extent, adjacent formations such as the San Andres.

Relying on analysis of the response in the observation well (ICP-WS-01) during the 7-day pumping test of ICP-WS-02 (INTERA 2012), transmissivity of the Capitan aquifer was estimated at 7,000 square feet per day (ft^2/day). The open-hole portion of the well within the aquifer was approximately 1,000 ft, resulting in a hydraulic conductivity of 7 ft/day.

ICP drilled two exploratory groundwater wells in the southern half of Section 2, Township 24 South, Range 35 East in Lea County on New Mexico State Trust Land approximately 12 miles northwest of Jal. Well ICP-WS-01 is located approximately 1,500 ft south of ICP-WS-02. Both wells are owned by ICP.

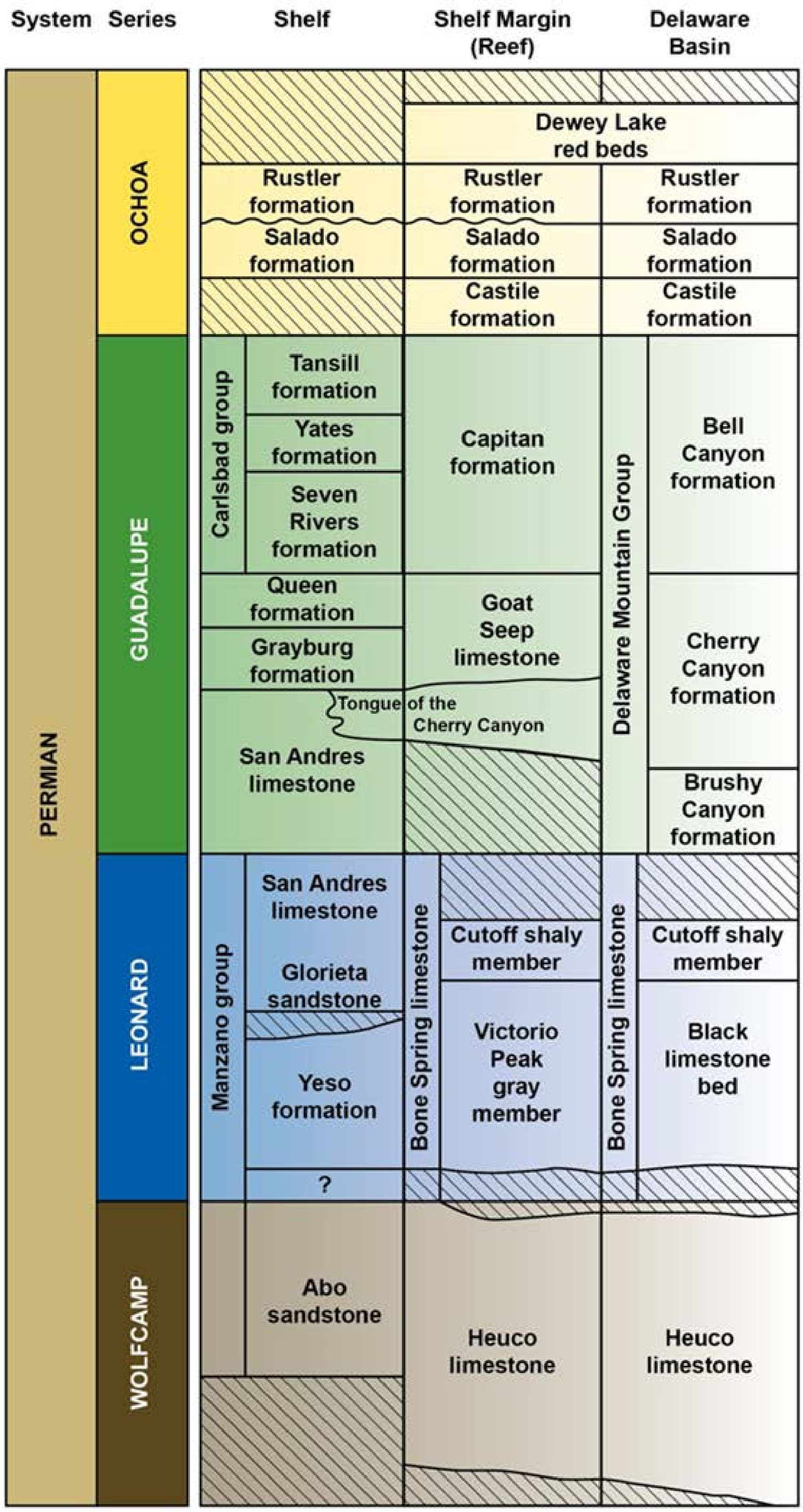


Figure 7-8. Permian Age Stratigraphy in Southeastern New Mexico

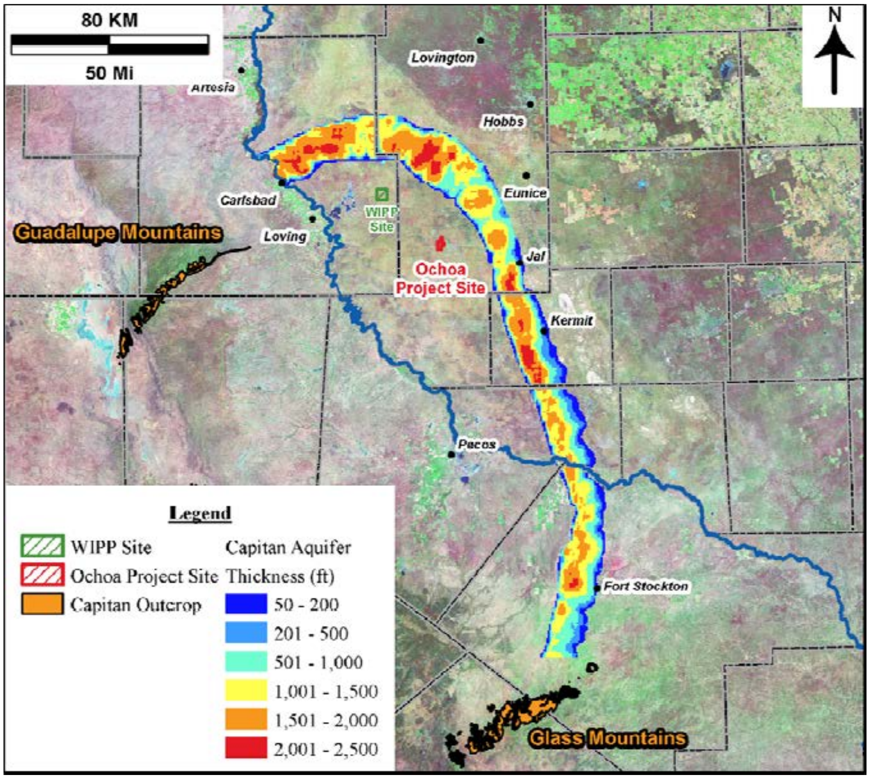


Figure 7-9. Thickness of the Capitan Aquifer

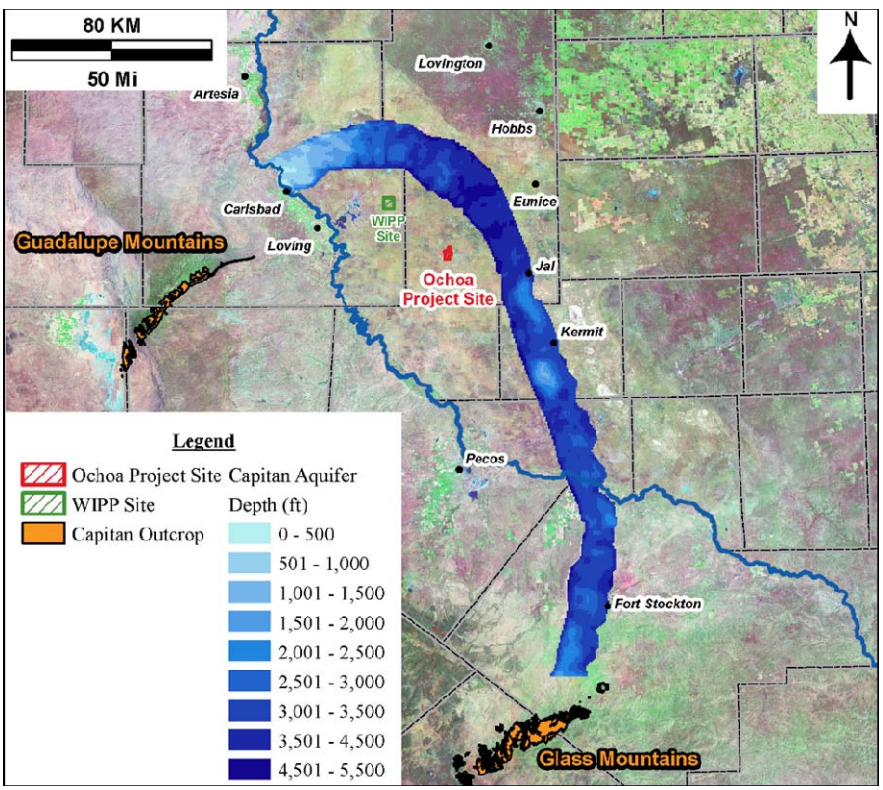


Figure 7-10. Depth to the Top of the Capitan Aquifer

Well ICP-WS-01 was drilled in February 2012, and was completed with on open-hole configuration through the Capitan aquifer from 4,384 to 5,381 ft below ground surface (bgs). ICP-WS-01 was drilled with fresh water through the Capitan aquifer, with significant volumes of water lost to the Capitan aquifer due to loss of return circulation. To maintain circulation, ICP-WS-02 was drilled using fresh water with high-pressure entrained air. Well ICP-WS-02 was drilled in May and June of 2012 and was completed as an open-hole configuration within the Capitan aquifer from 4,396 to 5,375 ft bgs. Well construction information and other characteristics for both wells are summarized in Table 7-4.

Table 7-4. Exploration Well Specifications

Specifications	ICP-WS-01	ICP-WS-02
Location Latitude	32° 14' 25.827" N	32° 14' 40.688" N
Location Longitude	103° 20' 21.319" W	103° 20' 21.079" W
Section Township Range	T24S, R35E, Section 2, SW, SE	T24S, R35E, Section 2, SW, NE
Drilling Dates	January 20 to February 8, 2012	May 11 to June 9, 2012
Well Construction Completion Date	February 8, 2012	June 9, 2012
Total Depth (ft bgs)	5,381	5,375
Casing Depth (ft bgs)	0 to 4,384	0 to 4,396
Open-Hole Depth Interval (ft bgs)	4,384 to 5,381	4,396 to 5,375
Producing Zone Length (ft)	997	979
Depth to Top of Capitan Aquifer (ft bgs)	4,351	4,341
Depth to Water, Below Measuring Point (measured July 8, 2012) (ft bgs)	~715	~720

In July 2012, a step drawdown test and a 7-day aquifer test were conducted by INTERA on behalf of ICP to assess the suitability of the proposed water supply for the Ochoa Project. During testing, the aquifer was pumped at a constant rate of 500 gpm for 7 days.

The Capitan aquifer has a saturated thickness of approximately 1,000 ft, which is composed of heterogeneous layers. Based on specific-capacity and aquifer testing, the aquifer can sustain pumping rates of 500 gpm or greater for extended periods of time. Drawdown was observed at the observation well located south of the pumping well during the step-drawdown and 7-day constant rate tests. Water levels in the observation well, ICP-WS-01, decreased approximately 6 ft during the 7-day constant rate test and recovered 90% after 34 hours of recovery.

During the 7-day constant rate test, the pumping-well (ICP-WS-02) water levels decreased approximately 206 ft, and drawdown stabilized after 4 days of pumping. The pumping well recovered 90% within 3 days and was 94% recovered after 24 days.

The pressure response analysis revealed characteristic behaviors indicative of a dual-permeability and/or multi-layered system. The aquifer appears to have producing intervals with differing pressures and permeabilities; for example, a low-permeability interval with higher pressure, and a high-permeability interval with lower pressure. Aquifer transmissivity is estimated to be 7,000 ft^2/day, yielding an estimated horizontal hydraulic conductivity value of 7 ft/day, which is on the lower end of the expected range for karst limestone, but higher than the expected range for limestone and dolomite aquifers (Freeze and Cherry 1979). The estimated storativity value is $5x10^{-5}$.

7.6.2 Hydrogeologic Characteristics at the Main Shaft Site

INTERA (2013b) conducted groundwater inflow testing on drill-hole ICP-092 for estimating groundwater pressure and inflows for shaft and slope design and construction. Hydrologic (drill stem) packer testing in drill-hole ICP-092 by INTERA (2013b) indicates that most of the strata have low hydraulic conductivity (1.0×10^{-3} feet per day [fpd]); however, the sandstone beds between 458 ft and 771 ft are highly fractured and had hydraulic conductivity on the order of 7.0×10^{-2} to 7.9×10^{-3} fpd and heads-above interval of 100 to 200 ft. These strata are generally considered to be "readily groutable" (Powers et al. 2007, p. 413). Due to the packer spacing, the definition of the inflow zones and permeability values were limited to the interval averages and were not defined for specific strata types. Exploration drilling, however, experienced lost circulation at times, indicating the existence of a fracture network in certain strata. Early time inflow rates (non-grouted) as high as 112 gpm were determined for a strata interval between 510 and 644 ft bgs, with steady-state inflow rates as high as 27 gpm for a strata interval from 458 to 508 feet bgs. The predominant water-bearing zone is expected to be between 458 and 699 ft bgs, with early time flow of 178.5 gpm and a steady-state rate of 42 gpm. Table 7-5 shows the results of the inflow testing.

Table 7-5. Summary of Estimated Steady-State and Early-Time Interval-Specific Inflow Rates (INTERA 2013b)

Interval Top (ft bgs)	Interval Bottom (ft bgs)	Interval Thickness (ft)	Lithological Category	Interval-Specific Steady-State Inflow Rate (gpm)	Interval-Specific Early-Time Inflow Rate (gpm)
40	53	13	Sandstone	0.02	0.04
165	203	38	Sandstone	0.2	0.3
267	303	36	Sandstone	0.3	0.6
318	326	8	Sandstone	0.08	0.1
349	361	12	Sandstone	0.1	0.2
382	411	29	Sandstone	0.3	0.4
458	508	50	Sandstone	27	21
510	644	134	Sandstone	11	112
655	699	44	Sandstone	4.4	45.5
735	771	36	Sandstone	0.4	0.6
1,348	1,366	18	Dolomite	0.001	1.6
1,585	1,611	26	Dolomite	1.4	1.8

ITEM 8: DEPOSIT TYPES

Potash is a general term for a potassium-bearing, chemical sedimentary mineral deposit that is the result of low-temperature chemical processes governed by evaporative concentration of a fluid such as seawater or freshwater. Bedded potash deposits commonly occur in sedimentary basins that have restricted connection to more dilute fluid. Diagenetic processes play an important role in evaporite mineral alteration and the production of specific potash ore minerals.

Potash mineralization occurs as assemblages of predominantly potassium chloride or predominantly potassium sulfate minerals. These assemblages may be interbedded or adjacent to one another, but rarely occur as a mixed assemblage in a single sedimentary bed. Individual potash mineral deposits can be correlated with geophysical logs and mapped over large areas.

Polyhalite is a hydrated sulfate of potassium (K), calcium (Ca), and magnesium (Mg) [$K_2SO_4 \cdot MgSO_4 \cdot 2CaSO_4 \cdot 2H_2O$] (Conley and Partridge 1944). Polyhalite may be white, light or medium gray, or salmon colored to orange to brown, or reddish. When pure it has 15.6% K_2O, 6.6% MgO, 18.6% CaO, 53.2% SO_3 with 6.0% H_2O. It is usually finely to medium crystalline, massive, and compact. The hardness is only 2.5 to 3 Moh's scale with a specific gravity of 2.8, but has a conchoidal fracture due to its compact, massive structure that gives an apparent hardness that is much greater. The polyhalite beds in the project area have exhibited finely crystalline laminae and "pinch and swell" structure (Figure 8-1).

Polyhalite is weakly soluble in cold and hot water and more so with a weak solution of hydrogen chloride (HCl). The weakly soluble nature of it has been the subject of study for a slow release fertilizer. Potassium sulfate is the preferred fertilizer for citrus, tobacco, sugar beet, and potatoes and for use in soils that would be intolerant to the additional salts found in muriate of potash (MOP) (Conley and Partridge 1944).

Polyhalite mineralization within the Ochoa Project area occurs within the lower half of the Tamarisk Member of the Rustler Formation. The polyhalite is interpreted to have formed in a paleolagoon of Ochoan age or alternatively a result of early or late diagenesis (Lowenstein 1987). The evaporites of the Rustler Formation were deposited in a shallow marine basin. Alteration is of gypsum to anhydrite (at burial) to polyhalite. The latter theory is supported by identification of gypsum pseudomorphs, and brecciation identified in the anhydrites. Gypsum pseudomorphs are composed of anhydrite.

Gypsum seen in the present day Rustler has been interpreted to have formed by late-stage replacement by rehydration, void filling cement, or fracture filling. Age dating of similar Salado Formation polyhalites has reported potassium-argon dates in the Delaware Basin from 198 to 216 MYBP which would suggest an early Jurassic to Triassic age of diagenesis (Brookins 1981).

Within the project area, the principal polyhalite resource occurs as a synform approximately 20 miles in length (northwest-southeast) having a width of approximately 9 miles. The polyhalite is typically light gray, massive, and is defined by a basal zone of about 1ft thickness with parallel to sub-parallel dark gray laminations (approximately 0.4 inches) with a sharp contact with the lower anhydrite unit. The middle zone of approximately 3 ft is defined by

Laminae Structure

"Pinch and Swell" Structure

Figure 8-1. Polyhalite Core Photos (from ICP) finer laminations at approximately 0.02-inch spacing. The upper approximate 1 ft is laminar and gradational to the upper anhydrite. The mineralized area is characterized by a bed thickness greater than 4 ft across the majority of the area, and a narrow peripheral zone that contains bed thickness from 0 to 4 ft thick. The upper anhydrite consists of parallel and crenulated laminations capped by a small clay parting and sharp contact with the upper halite.

ITEM 9: EXPLORATION

Drilling, gamma logging, geotechnical logging, and geochemical logging were utilized in exploration and investigation of the mineral deposit. Evaluation of the Ochoa Project (the Property) has been accomplished by:

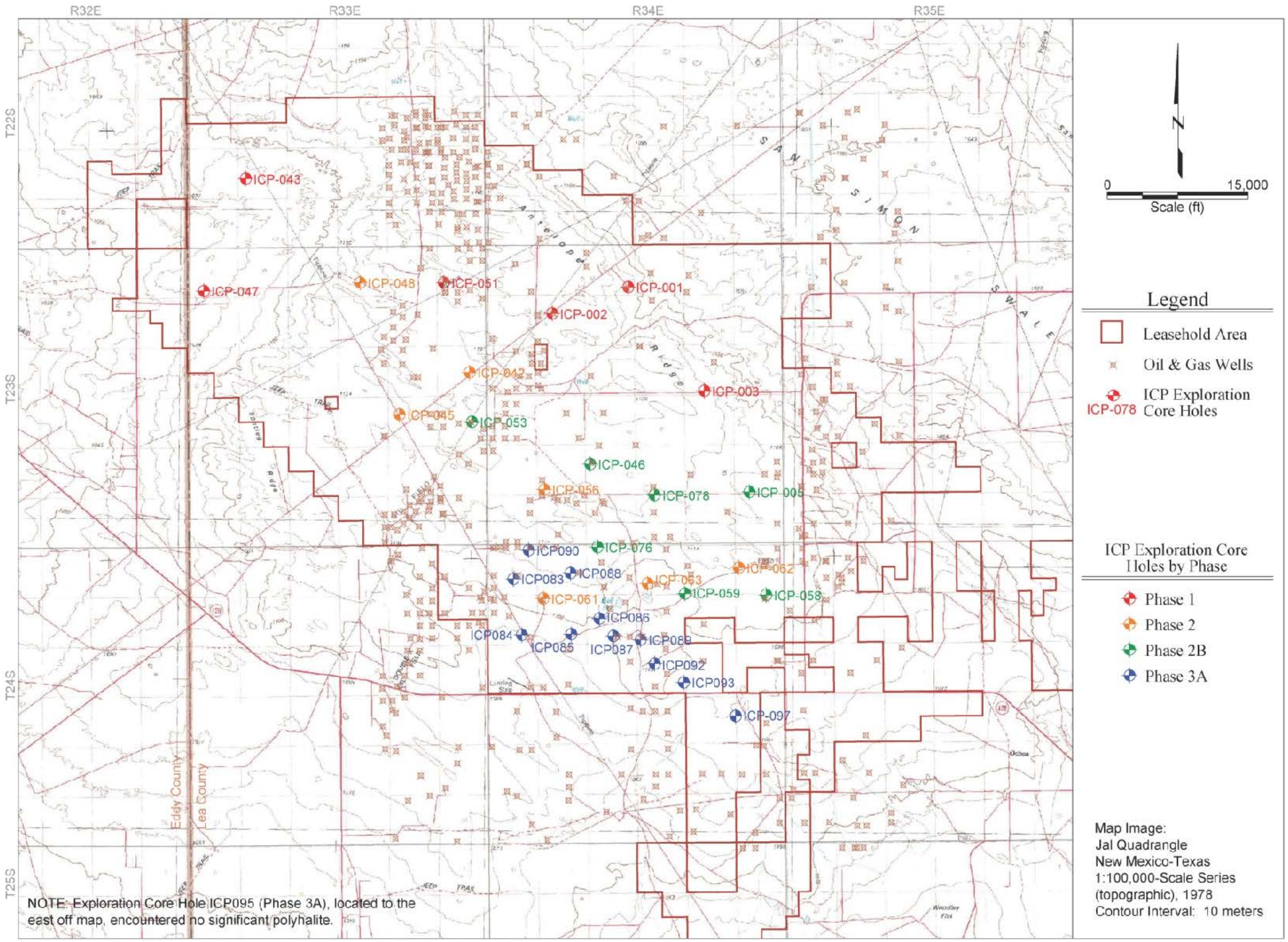
- Review of published literature
- Review of Micon (November 2008) NI 43-101 TR on ICP's Ochoa Project, Lea County, New Mexico, USA and data used in that report
- Review of Micon (January 2009) NI 43-101 TR on ICP's Ochoa Project, New Mexico, USA, and data used in that report
- Review of Chemrox and Gustavson (19 August 2009) NI 43-101 TR on ICP's Ochoa Project, Lea County, New Mexico, USA and data used in that report
- Review of Gustavson (November 25, 2011c) NI 43-101 TR on ICP's Ochoa Project, Lea County, New Mexico, USA and data used in that report
- Review of Gustavson (December 30, 2011d) NI 43-101 TR on ICP's Ochoa Project, Lea County, New Mexico, USA and data used in that report
- Review of Gustavson (April 15, 2012) PFS of the Ochoa Project on ICP's Ochoa Project, Lea County, New Mexico, USA and data used in that report
- Exploration drilling with downhole gamma surveying conducted by ICP between February 2009 and March 2013
- XRD, XRF, and chemical analysis on core samples by commercial laboratories
- Analysis and check of assay results
- Drilling and assay work of the bulk of contemporary exploration work are described in Items 10 and 11

9.1 Exploration Drill-Hole Data

ICP successfully drilled, cored, logged, plugged and abandoned 32 vertical exploration holes throughout the permit area during a three-phase exploration drilling campaign (Figure 9-1). Data from an additional 855 petroleum wells were used to establish regional correlations. Phase 1 consisted of 6 holes, Phase 2, 7 holes and Phase 2B, 7 holes. Phase 3A began in August of 2012 and completed 12 holes, 11 of which were in the main resource area. Early phases of drilling recovered smaller diameter core (3 inches). The need for bulk samples for metallurgical testing drove the acquisition of 6-inch-diameter core for most of Phase 3A. This TR includes drilling and assay data to 25 May 2013, except for a few additional assay analyses provided later that were included in the geochemistry analysis only (Item 7.5).

ICP applied for approval to explore for potassium minerals on federal exploration permits and was granted permission in December 2008. ICP applied for and received permission to explore for potassium minerals in May 2010. ICP does not have any private mineral leases. See *Item 4.1 Mineral Surface and Land Tenure* for details.

To estimate a Measured and Indicated (M&I) Resource, ICP has drilled to delineate the polyhalite mineralization within the Property boundary. Agapito Associates, Inc. (AAI) accepts the drilling spacing of within 0.75 mile for Measured and 0.75 to 1.5 miles for Indicated Resources that was used in the December 30, 2011 TR (Gustavson 2011d).



800-01 IC Potash [800-01 ICP_Drill Hole Map by Phase.dwg; Layout: DH Map]:smvf/rjl (7-23-2013)

Figure 9-1. Ochoa Drill-Hole Locations (ICP-095 not shown, off map to east)

As part of the above, ICP drilled three exploration drill holes for mine geotechnical sampling and physical properties analysis along the projected path of the mine slope and ventilation shaft location.

9.2 Seismic

No deep (ore bed elevation) seismic surveys were conducted by ICP on the Property. Surface seismic velocity surveys were conducted by Sage Earth Sciences, Inc. (2013) in the processing plant and tailings site areas for surface facility design purposes, and along the line of the mine slope.

ITEM 10: DRILLING

Stewart Brothers Drilling Company of Milan, New Mexico, drilled all 32 exploration holes. Each drill hole was drilled in two sections. The upper portion of each hole, from ground surface to within 50 to 75 ft of the top of the polyhalite, was drilled using rotary drilling techniques. The lower portion of each hole was cored in order to obtain samples for grade and engineering analyses. In the Phase 3A drilling, one to four sidetracks were drilled in addition to the vertical hole to obtain additional samples for metallurgical testing. In those cases, drilling tools were pulled back up the hole, a whipstock was set, and additional 6-inch diameter core samples were drilled through the ore zone.

10.1 Rotary Drilling

Rotary drilling was used to advance each hole through the Dewey Lake Formation and into the upper portion of the Rustler Formation. This portion of the drill hole was advanced using water-based gel chemical drilling fluid, and was cased to maintain borehole integrity and protect groundwater. Rock chips collected at 5-ft intervals were washed in water, logged for lithologic description, placed in chip trays, and were transported to and stored at ICP's core lab in Hobbs, New Mexico. The rotary drilling program in general was as follows:

1. Set a 10⅞-inch conductor pipe at the surface and drill ahead using water-based gel at 9⅞ inches to approximately 25 ft below the poorly consolidated and muddy Dewey Lake Redbed Formation (approximately 125 ft). For Phase 3A, drilling was to ~5 to 10 ft below the Dewey Lake.

2. Set 7-inch casing and cement to surface. Drill a 6-inch hole to approximately 15 ft below the Magenta Dolomite Member of the Rustler Formation. Displace gel mud with salt-saturated brine and drill ahead to the core point just below the Magenta Dolomite. For Phase 3A, 9⅝-inch casing was set to obtain 6-inch core in some cases.

A Pason Live Rig View was used on the Phase 3A drilling. Pason is a proprietary well monitoring system that allows one to monitor and record drilling parameters to optimize drilling activities and provide a permanent record of the drill. This also allows for monitoring of the well depth and conditions on the rig remotely.

The rig geologist determines the depth at which to begin coring based on cuttings, hole condition, and well correlations. In exploration Phases 1 and 2, this depth (the "core point") was typically about 20 ft above the polyhalite seam and was demarcated by an anhydrite marker bed (i.e. APH05 and APH06). During Phase 2B drilling, the core point was moved to roughly 50 to 75 ft above the polyhalite seam to allow coring of additional strata to recover roof rock core samples for geotechnical analysis. In Phase 3A, the core point was as little as a few feet above the ore zone.

10.2 Diamond Core Drilling

For coring in the target evaporite intervals, a sodium chloride (NaCl) salt-saturated drilling fluid was used to minimize dissolution and alteration of water-soluble minerals, predominantly halite and polyhalite. Use of salt-saturated drilling fluid was initiated prior to drilling to core point in order to provide sufficient time to establish stable chemical and rheological properties in the drilling fluid of both the active and reserve drilling fluid systems.

Phase 2 and 2B coring was 3-inch core; Phase 3A was generally 6-inch core. The geotechnical sections of holes were cored with a 3-inch barrel, the drill holes were then reamed out, and the target stratigraphy was cored with a 6-inch barrel in the vertical and sidetracks. At the core point, the rotary drilling assembly was removed from the hole and replaced with a 40-ft core barrel and bottom-hole assembly. A 40-ft core run was completed, and the core barrel and drill string were then tripped out and the core recovered. This process was repeated if a second or third core run was desired. The large-diameter core in Phase 3A was recovered in 10-ft core runs and also supplemented by sidetracked holes to core additional samples. In that case, the bit and string were pulled back up and a whipstock set to obtain as many as four sidetracked cores through the ore zone. ICP-092 and ICP-093, the location of the shaft and slope, respectively, were cored from near surface for geotechnical logging. ICP-097 was cored from near surface to below the expected slope horizon at that location (approximately 131 ft bgs), and then at the polyhalite bed zone.

10.3 Wireline Logs

The completed drill holes were logged with wireline geophysical tools. Logs collected during Phase 1 work include total gamma, caliper, and standard electric logs. No density or neutron logs were acquired during Phase 1 exploration. A variety of tools were used in Phase 1 drilling and presentation of the data recorded was not standardized. Phase 2, 2B and 3A holes were logged using a consistent suite of tools. Logs collected include spectral gamma, laterolog and induction electrical, formation density, sonic, and neutron density logs (Table 10-1).

10.4 Collar Surveys

ICP commissioned commercial surveying companies to survey the location of each of the 32 drill holes completed during Phases 1, 2, 2B, and 3A. Drill-hole collar location information is presented in Table 10-2 and Figure 9-1 shows a drill-hole location map.

10.5 Core Recovery

Core recovery in the polyhalite and anhydrite zones was excellent in terms of length and minimal alteration of the rock by the salt-based drilling fluid. In early Phase 1 drilling, halite zones above and below the polyhalite reacted with the drilling fluid and partially dissolved. In most cases, the core was under gauge by less than 0.04 to 0.08 inches. Severe reduction in gauge (e.g., 0.4-inch radial reduction) occurred when the drilling fluid was not properly conditioned or maintained near salt saturation or when there was a prolonged coring time caused by a slow penetration rate through the anhydrite and polyhalite horizons.

Other than dissolution, the surface of the core showed little to no evidence of chemical reaction, such as pitting or efflorescence, between the drilling fluid and minerals. The core was not washed or scrubbed to remove drilling fluid.

After drilling and logging operations were complete, all holes were plugged with cement from total depth (TD) to ground surface. Drill-hole summary reports were compiled for drilling completed during Phases 1, 2, 2B, and 3A. These reports contain core descriptions, photographic records, and assay data. The reports are on file in the Golden and Hobbs business offices; digital copies are on the server at ICP’s Golden office.

Table 10-1. Summary of Wireline Logs Collected

	Hole ID(Drill Order)	Caliper*	Gamma	Spectral Gamma	Sonic	Density	Neutron	Resistivity*	Directional Survey
Phase 1 Drilling Program	ICP-021(001)	P	P	N	P	N	N	N	P
	ICP-022(002)	P	P	N	P	N	N	N	P
	ICP-026(003)	P	P	N	N	N	N	N	P
	ICP-047(004)	P	P	N	P	N	N	N	P
	ICP-043(005)	P	P	N	N	N	N	P	P
	ICP-051(006)	P	P	N	P	N	N	P	P
Phase 2 Drilling Program	ICP-042(007)	P	P	P	P	P	P	P	P
	ICP-045(008)	P	P	P	P	P	P	P	P
	ICP-048(009)	P	P	P	P	P	P	P	P
	ICP-062(010)	P	P	P	P	P	P	P	P
	ICP-063(011)	P	P	P	P	P	P	P	P
	ICP-061(012)	P	P	P	P	P	P	P	P
	ICP-056(013)	P	P	P	P	P	P	P	P
Phase 2B Drilling Program	ICP-046(014)	P	P	P	P	P	P	P	P
	ICP-053(015)	P	P	P	P	P	P	P	P
	ICP-005(016)	P	P	P	P	P	P	P	P
	ICP-078(017)	P	P	P	P	P	P	P	P
	ICP-076(018)	P	P	P	P	P	P	P	P
	ICP-058(019)	P	P	P	P	P	P	P	P
	ICP-059(020)	P	P	P	P	P	P	P	P
Phase 3A Drilling Program	ICP-083(028)	P	P	P	P	P	P	P	P
	ICP-084(027)	P	P	P	P	P	P	P	P
	ICP-085(024)	P	P	P	P	P	P	P	P
	ICP-086(023)	P	P	P	P	P	P	P	P
	ICP-087(022)	P	P	P	P	P	P	P	P
	ICP-088(030)	P	P	P	P	P	P	P	P
	ICP-089(021)	P	P	P	P	P	P	P	P
	ICP-090(029)	P	P	P	P	P	P	P	P
	ICP-092(032)	P	P	P	P	P	P	P	P
	ICP-093(031)	P	P	P	P	P	P	P	P
	ICP-095(025)	P	P	P	P	P	P	P	P
	ICP-097(026)	P	P	P	P	P	P	P	P

N = not run P = partial run

*1-arm caliper run in all holes, 3-arm caliper run in Phase 2, 2B and 3A holes; resistivity logs variously included guard, induction, and normal.

Table 10-2. Drill-Hole Collar Location Information

Hole Identifier	Location (x)	Location (y)	Elevation (ft)	Depth (ft)	Alias	Phase	Drill Order
ICP-001	776952.02	484139.85	3632.97	1433.00	ICP-021	1	1
ICP-002	768852.72	481330.50	3700.71	1547.80	ICP-022	1	2
ICP-003	785071.00	473170.56	3690.11	1582.50	ICP-026	1	3
ICP-005	789877.11	462426.84	3627.10	1573.84	ICP-016 ICP-027	2B	16
ICP-042	760044.08	475098.62	3726.82	1505.90	ICP-007	2	7
ICP-043	736170.90	495629.41	3561.29	1023.60	ICP-005	1	5
ICP-045	752561.30	470629.69	3692.37	1451.80	ICP-008	2	8
ICP-046	772896.70	465395.34	3673.47	1567.80	ICP-014	2B	14
ICP-047	731687.82	483716.59	3519.44	1022.19	ICP-004	1	4
ICP-048	748357.44	484608.98	3677.52	1350.00	ICP-009	2	9
ICP-051	757275.09	484597.78	3747.04	1521.50	ICP-006	1	6
ICP-053	760294.62	469862.41	3693.52	1478.30	ICP-015	2B	15
ICP-056	768016.43	462690.21	3666.93	1521.70	ICP-013	2	13
ICP-058	791673.52	451511.77	3624.10	1568.60	ICP-019	2B	19
ICP-059	783012.40	451667.58	3607.76	1562.90	ICP-020	2B	20
ICP-061	767904.05	451133.20	3627.05	1481.50	ICP-012	2	12
ICP-062	788750.85	454398.88	3631.85	1602.40	ICP-010	2	10
ICP-063	779035.25	452735.26	3587.33	1535.10	ICP-011	2	11
ICP-076	773682.68	456577.64	3657.16	1563.50	ICP-018	2B	18
ICP-078	779760.57	462099.88	3664.45	1595.24	ICP-017	2B	17
ICP-083	764713.75	453206.82	3634.86	1195.30	ICP-028	3A	28
ICP-084	765606.30	447289.65	3576.78	1215.60	ICP-027	3A	27
ICP-085	770882.90	447366.82	3595.55	1241.40	ICP-024	3A	24
ICP-086	773873.80	449021.32	3609.40	1514.50	ICP-023	3A	23
ICP-087	775388.27	447169.85	3610.32	1234.60	ICP-022	3A	22
ICP-088	770818.65	453836.69	3651.74	1251.60	ICP-030	3A	30
ICP-089	778291.12	446749.47	3614.55	1549.70	ICP-021	3A	21
ICP-090	766339.09	456255.89	3648.01	1183.00	ICP-029	3A	29
ICP-092	779789.57	444238.60	3624.10	1654.40	ICP-032	3A	32
ICP-093	782886.59	442230.97	3599.93	1565.20	ICP-031	3A	31
ICP-095	851048.06	438539.34	3364.02	1524.70	ICP-025	3A	25
ICP-097	788417.77	438679.02	3581.90	1553.70	ICP-026	3A	26

Note: Coordinates are UTM Zone14, datum WGS84. With the exception of holes ICP-001, ICP-002, and ICP-003, the Alias for each drill hole represents the drill hole ID submitted on associated permit documents, which also corresponds directly to that hole's position (number) in the drilling sequence. For holes ICP-001, ICP-002, and ICP-003, the reverse is true, and the Alias for these holes represents the pre-drilled, assigned hole identification, and the hole ID reflects the permit ID and number in the drilling sequence.

10.6 Gas and Oil Geophysical Evaluation

ICP acquired 855 geophysical borehole logs from gas and oil wells within the project area. Wireline log readings from these boreholes were used to interpret subsurface lithology. Logs from additional wells exist, but were not used for this study because closely spaced holes did not require logs from more than one hole to map the location (i.e., one hole within a nine hole production spot).

Initially, the polyhalite bed was identified based on the total gamma curve. The base and top contacts of the deposit were selected at the inflection points of the gamma curve. Precisely determining the inflection point was challenging given the ratio of the amplitude of the gamma peak to the thickness of the polyhalite bed. In many cases, this method was found to overestimate the thickness of the polyhalite bed by as much as 25%.

Mineral and chemical analyses from the ICP core holes were used to precisely identify the upper and lower limits of the polyhalite deposit, resulting in measurable true thickness. ICP geologists used these core data and wireline logs from the core holes to refine the polyhalite bed thickness method for the gas and oil logs. The polyhalite bed is defined or “picked” by combining gamma data with the resistivity curve, noting the doublet on the resistivity curve (Figure 10-1). The base is picked at the minimum of the lower part of the doublet. Similarly, the top is picked at the minimum of the upper part of the doublet.

The wireline logs from the gas and oil wells were reevaluated for the individual marker beds and polyhalite contacts were adjusted accordingly. A total of 1,385 wireline logs were evaluated, and 855 from within a 6-mile perimeter of the Property were used for correlation in and around the project area, including wireline logs from the 32 ICP core holes. In the case of closely spaced (i.e., clustered) drilling, only one or a few logs were used. Data from the ICP holes were used to anchor all correlation efforts. Gas and oil well wireline logs were correlated with ICP core-hole logs, working outward from the ICP core holes to create interpreted subsurface maps of each marker bed.

10.7 Subsurface Mapping

Fifteen geophysical wireline log markers were defined within the target geologic framework. Six of these are formal lithostratigraphic units that are encountered throughout the study area. The remaining nine markers are associated with individual sedimentary beds within the formal lithostratigraphic units which exhibit unique geophysical responses (Table 10-3).

The effective use of marker correlation and mapping was limited to establishing structural framework, estimating lithostratigraphic volumes, and evaluating physical trends such as changes in elevation and thickness. Figure 10-1 is an example of wireline borehole logs correlated using the 15 markers.

Some of the markers were not present throughout the entire reconnaissance area (e.g., Halite_U, APH_05, APH_06, Top Polyhalite, and Base Polyhalite), indicating a limit to the mineralization and presumed delineation of the paleoshoreline. Structural maps with contoured surfaces of the marker bed horizons were created based on the correlated wireline logs.

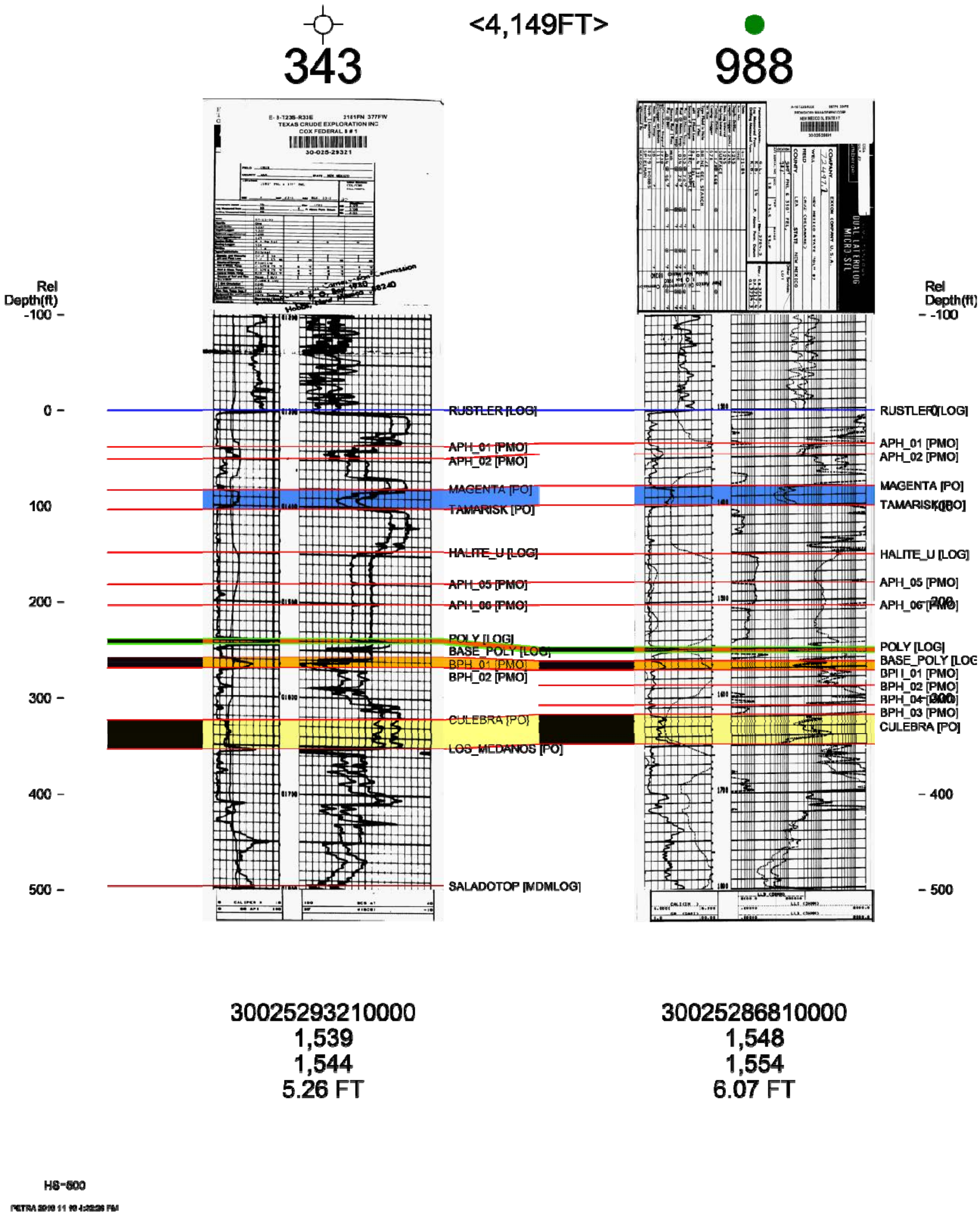


Figure 10-1. Typical Wireline Logs with Marker Horizons (from Gustavson 2011c)

Table 10-3. Summary of Geophysical Markers Defined for Correlation

	Marker	Type of Marker	Lithology
1	Top Rustler	Stratigraphic – formation	Anhydrite
2	APH_01	Geophysical	Siltstone-shale within Forty-niner Member
3	APH_02	Geophysical	
4	Top Magenta	Stratigraphic - member	Dolomite
5	Top Tamarisk	Stratigraphic – member	Anhydrite
6	Halite_U	Geophysical – unknown origin; appear to be the base of the upper half of the Tamarisk anhydrite and marks the change to a lower zone of anhydritic halite and siltstone	None – reflects division between upper and lower anhydrite zones
7	APH_05	Geophysical	None – may be a bedding plane feature
8	APH_06	Geophysical	None – may be a bedding plane feature
9	Top Poly	Geophysical	Polyhalite, depth of gamma high may occur below depth of density log because anhydrite density is similar to polyhalite density
10	Base Poly	Geophysical	Transition to underlying anhydrite
11	BPH_01	Geophysical	Top shale or anhydritic shale
12	BPH_02	Geophysical	Base of shale zone, transition to anhydrite
13	Top Culebra	Stratigraphic - member	Silty dolomite
14	Top Los Medaños	Stratigraphic - member	Siltstone, top of thick siltstone sequence, include 1st anhydrite as part of upper portion of sequence and immediately below siltstone that forms the spike
15	Top Salado	Stratigraphic – formation	Halite

Previous studies by Keller, Hills, and Djeddit (1980) concluded the reconnaissance area is a depocenter within the Delaware Basin. The results of correlating and mapping the subsurface markers of the Rustler Formation support that hypothesis and suggest the following with regard to the structure of the basin:

- Elongate depression oriented northwest-southeast
- Closed in the northwest and open, but restricted in the southeast
- Bound on the east by a well-defined ridge (50 to 200 ft relief, 2 to 3 miles wide)
- Bound on the west and north by broad sloping ramp
- No disruptions identified (e.g., sharp elevation changes, sharp isopach variations, or sharp slope changes from marker to marker)
- No significant migration of basin depocenter axis or other framework features including highs, lows, and edges
- Variation in thickness between markers is very consistent, but clearly thin or truncate toward and at the edges of the sub-basin
- No clear evidence of significant faults

The geology of the project area is representative of a depositional basin that has experienced uplift and minor structural deformation. The interpretation of a structurally quiescent basin is supported by strong marker correlation, consistent thicknesses between markers, consistent slope of surfaces within the sub-basin, and the thinning trend and truncation of markers near areas where underlying markers begin to shallow in depth. The present shape and slope of the basin is probably enhanced by post-lithification events in the region, the most predominant being salt dissolution and subsidence in the Nash Draw to the west and the San Simon Swale to the east.

In addition, smaller, localized subsidence features are present in the general project area, but have not been investigated at this time. The most prominent of these features that has been identified is located just east-southeast of the processing plant site.

ITEM 11: SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Sample Handling and Security

Sodium chloride-saturated drilling fluids were used during coring in order to minimize drilling-induced alteration of the recovered core. The rate of penetration, revolutions per minute, weight on bit, pump pressure, and strokes per minute were monitored by the driller and documented by the Pason system. Following each core run, the drill string and core barrel were brought to the surface, and the core was removed from the vertically suspended core barrel. ICP has prepared Resource Assessment Team Protocols for core handling, sample preparation and processing (Protocols 4a, 4b, and 5).

The core was laid out on a core logging table and the broken sections were fitted together to reconstruct the continuous core recovered. If core loss was suspected, a spacer was placed in the layout until the core could be matched to the geophysical logs. Core length was measured and percent recovery calculated based on the actual length of core cut, and lost core and broken core intervals were documented. The core was cleaned with dry rags and marked to show vertical orientation and drilled depth in 1-ft increments. The marked core was video recorded and digitally photographed, then boxed with desiccant packs and foam spacers to impede shifting during transport. Broken and fragmented core was bagged and labeled prior to boxing. The top and bottom of each core box were labeled with the drill-hole name, core run number, box number, and depth interval of core contained in the box. The boxes were sealed with security tape and a chain of custody form was completed documenting the date of transport from the field. All core was transported from the field to ICP's core lab in Hobbs by ICP personnel using company vehicles.

Upon arrival at the core lab, the chain of custody form was checked against the shipment to verify all materials were present and in secure condition. Ore zone thicknesses were corrected to match the spectral gamma ray geophysical log; however, the geophysical log depth was corrected to the drillers reported depth. The standard industry procedure regarding depth correction between geophysical logs and core (which typically rely on Pason or driller-provided depths) is to adjust the generally less accurate field log depths to match the depths indicated on the geophysical logs. ICP took an alternate approach, relying on the driller-provided rod count depth to adjust the depth of the geophysical logs and to establish the depth of the ore zone for geologic modeling and mine planning (Okita 2013a).

Corrected depths were marked on the core in red permanent marker. As part of improved sample handling protocols which were implemented during Phase 2B of the project, the full length of each core run was photographed with a Canon EOS Rebel T1i camera mounted on a stationary tripod. The core was passed by the camera on a rolling table, and each photograph contained an engineer's scale, color scale, and a gray scale. The individual photographs were archived and stitched together using computer software to create a single photograph of the full length of core.

After the full length of core was photographed, it was sawn in half (dry), and one half was then cut into two quarters. One quarter was canted (the outer curved portion of the quarter core was cut off) to limit the possibility of sending core altered by the drilling fluid to the lab for analysis. The canted quarters were used as the analytical samples and were cut into 3-inch to 6-inch interval lengths. These samples were assigned a blind number from a sample book which associated the drill-hole identifier, depth interval, and sample description to the blind number. The samples were individually vacuum sealed in 6-inch by 10-inch, 3-mil poly bags, labeled with their respective blind numbers, and sent to the lab. Multiple core runs may be sent to the lab as a batch, but a single core run was never split between two batches. A chain of custody document listed the sample numbers, shipment date, and mode of transfer and was completed for each batch of samples sent to the lab. A signed copy of the chain of custody was returned to ICP upon delivery to the lab. The designated primary lab was H&M of Allentown, New Jersey.

All retained core was individually vacuum sealed in less than 2-ft intervals in 6-mil poly tubing with a desiccant pack, a humidity indicator, and an index card marked with the drill-hole identifier and sample depth. All vacuum sealed cores were placed back in the appropriate box, with adjusted depths labeled on the outside and a maximum temperature indicator placed on the inside of the box. Core boxes were stacked five boxes high on shelves for long-term storage after the core was processed.

11.2 Analytical Procedures and Sample Preparation

During exploration Phases 1 and 2, samples were shipped to two independent contract labs, The Mineral Lab of Golden, Colorado and H&M, for preparation and XRD and XRF analysis, and to one independent lab, ALS Chemex of Reno, Nevada, for inductively coupled plasma optical emission spectrometry (OES) and supporting analysis. The results of the different methods of analyses were evaluated, and ICP determined that quantitative XRF and XRD analyses were the most useful in establishing polyhalite grade. The XRD and XRF methods provide the added benefit of quantitatively determining the mineralogy and distribution of the elements of interest without the need to dissolve the sample.

Beginning in Phase 2B exploration, ICP standardized the sampling process and began using only XRD and XRF analyses from H&M. Samples from Phases 1 and 2 were reanalyzed according to this process in order to standardize the analytical data. The entire amount of each sample was crushed with a jaw crusher to less than 0.24 inches and then ground in a Retsch RM100 motorized mortar and pestle to a fine powder (–325 mesh) that was suitable for XRD analyses. The following processing methods were used by H&M in processing the core samples received from ICP.

11.3 Quantitative XRD

A small amount of each fine powder was placed into a standard sample holder and put into a Panalytical X'pert MPD Pro X-ray diffractometer using copper radiation at 40 kV/40 mega-ampere. Scans were run at angles (theta) of incidence from 10° to 80° with a step size of 0.0156° and a counting time of 200 seconds per step. Once the diffraction patterns had been collected, crystallographic databases from the International Center for Refraction Data (ICDD) and the Inorganic Crystal Structure Database (ICSD) were used to identify the minerals present. Finally, quantitative phase analysis was performed with a Rietveld Refinement analysis, which has a typical accuracy of about 1%. The x-ray diffractometer is calibrated using the National Institute of Standards and Technology (NIST) traceable standard reference material (SRM) 1976. Calibration is performed every quarter or when the instrument requires servicing. Recent certification provided by the lab indicates that instrumental error is almost 10 times better than the allowable error (Okita 2013b, Performance Qualification S/N 201772, March 26, 2013).

11.4 Quantitative and Semi-Quantitative XRF

XRF samples were mixed with 20% Paraffin and pressed in a die at 30 t for 5 minutes to produce a standard 1.57-inch XRF specimen. Each pellet was then tested on a Bruker S4 Wavelength Dispersive X-ray Fluorescence Spectrometer for elements with wavelengths between sodium and uranium. This analysis uses a spectrometer, a sequential instrument to examine one element at a time using varying kilovolt settings, filters, collimators and monochromators optimized for detection of each element. Semi-quantitative analysis was then performed using the Fundamental Parameters method, a standardless technique which takes into account the fluorescence yield, absorption, and matrix effects to estimate the atomic chemical composition. This technique has an accuracy of about 5% for the major elements.

Full quantitative analyses were performed for sodium, chlorine, magnesium, sulfur, potassium, and calcium. The remaining trace elements were analyzed by semi-quantitative analysis. The results are a hybrid of fully quantitative analysis for the major elements with errors of approximately 1%, and semi-quantitative analysis for the trace elements with errors of approximately 10%.

The XRF unit runs calibration standards supplied by Breitlander Calibration Lab. These are used during setup of the instrument, service checks, and for drift correction. Drift correction was performed prior to conducting analysis, and negligible drift was incurred in all cases. The most recent inspection certificate (Okita 2013c, Performance Qualification serial number 202071, March 13, 2013) was provided by H&M for review.

11.5 QA/QC (Quality Assurance and Quality Control) Measures

The sampling program used duplicate, blank, and standard samples inserted into the sample batches for testing alongside the samples from intervals of interest. This allowed for a check and correction of sample test results, as necessary. Duplicate samples were used to provide a measure of the repeatability of test results, including sample homogeneity and testing procedures.

Duplicate samples were assigned a different sample number than their counterpart sample. Blank samples did not contain the material of interest, potassium in this case, and provided a measure for cross-contamination between individual samples as they were prepared and tested. SRMs have a known composition, which allowed for a comparison between the lab test results and the known composition of the standard. The SRM provides a means of comparison to identify instances and degrees of under- or over-reporting of chemical constituents in the sample testing results.

ICP follows a written protocol for the preparation and submission of samples, which includes submitting at least two SRMs and one duplicate sample for every ten samples submitted. Duplicate samples consist of a portion of the cut core which faces the original sample. The core sample is sent “as cut,” and crushing and grinding are completed by the analytical lab. SRMs are submitted as pulp samples, which are already crushed and ground.

An analytical batch consisted of 12 to 20 samples made up of core samples, one or two duplicates, one SRM, and one blank. During Phase 1 exploration, no duplicates were run. The SRM consisted of polyhalite, sylvite, langbeinite, or commercial fertilizer, and the blanks were quartz sand. Upon review of the Phase 1 program, ICP determined that too many standards, and too many standards with poorly established composition, were being used, and that the blank (a silicate) was inappropriate because it was not of similar type to the sample (i.e., sulfate).

During Phases 2 and 2B, the SRM was limited to langbeinite, polyhalite, or arcanite (reagent grade potassium sulfate [K_2SO_4]), and reagent-grade anhydrite was used as the blank. During Phase 3A, SRMs #2a, #2b, and #2c were created from the Langbeinite-M sample. Only SRM #2b was inserted. Langbeinite-M SRM #2 was prepared by RDi Mining Consultants and Laboratory (Denver, Colorado) (RDi) and was exhausted in April 2013. Prior to Phase 3A, ICP required the lab to perform an analytical repeat for one sample in every ten. That action is no longer required as ICP has determined that this method is insensitive to the error that the repeats are designed to detect, and the insertion rate is variable (Okita 2013a, ICP-P3a-GradeThk-IntRelease-Binder1b-20130523.pdf, page 4). AAI recommends that ICP document any modifications to the current QA procedures, including changes to SRMs or rates of insertion.

11.5.1 Standards

ICP in-house standards are used for repeat analysis over time of characterized material. Standards are used to monitor laboratory consistency and to identify sample discrepancies. They are submitted as a pulp and are either an SRM or certified reference material (CRM) or a site-specific standard that may or may not be certified. A CRM has a performance range that is either specified by the certifying entity, or direction is provided on how to determine a performance range. Generally, the performance range is approximately ±2 standard deviations from the mean of the standard, and the standard is expected to perform within this range 95% of the time. AAI reviewed the standards employed by ICP to verify that the assays contained in the database were reliable. Standard samples were submitted in sufficient numbers for preliminary statistical analysis in exploration Phase 3A. These included granular langbeinite (SRM #2 and #2b) and granular polyhalite (SRM #4). Commercial, reagent-grade gypsum (SRM #6), commercial reagent-grade arcanite (SRM #7), commercial potassium chloride fertilizer (sylvite, SRM #8), and crushed, variable-size polyhalite (SRM #10) were also submitted as standard samples, but in such limited numbers that statistical analysis is not warranted at this time. While the overall SRM insertion rate is relatively high, the insertion rate of any single standard is low.

ICP provided vendor certificates for standards (Table 11-1) SRM #6, SRM #7, SRM #8, and SRM #9. Vendor certificates are not available for the prepared bulk standards, so rather than using an expected standard mean, the individual standards were plotted against ±2 standard deviations of the combined average analytical result, as shown in Figures 11-1 through 11-3.

11.5.2 Duplicates

Duplicates are used to monitor sample batches for the precision of the assay and sample homogeneity at each step of preparation. AAI has recommended that ICP insert sample duplicates at every sample split during sample preparation, and that they not be placed in sequential order. When original and duplicate samples are plotted in a scatter plot, perfect analytical precision will plot on 45° x-y slope. Core duplicates are expected to perform within ±15% of the x-y slope.

Table 11-1. ICP Potash Standard Reference Material

Standard Reference Material	Description	Comment
SRM #1 - not used	n/a	Not assigned to any material. May have been quartz
SRM #2 - langbeinite-M	Granular langbeinite	Prepared bullk; Exhausted in Phase 3a
SRM #2b - langbeinite-M	Granular langbeinite	Introduced in Phase 3a, material from bulk supply of SRM #2
SRM #3 - not used	n/a	Not assigned to any material. May have been calcite or dolomite
SRM #4 - polyhalite-H17	Granular polyhalite	Prepared bulk; Powdered grey polyhalite from H17 core
SRM #5 - not used	n/a	Not assigned to any material
SRM #6 - gypsum	Reagent grade	Commercial purchase of reagent grade, CoA from vendor (NoahTech)
SRM #7 - arcanite	Reagent grade	Commercial purchase of reagent grade, CoA from vendor (NoahTech)
SRM #8 - sylvite BCR-113 (file note incorrectly names it BRC-113)	Potassium chloride fertilizer	Commercial purchase of reagent grade, CoA from vendor (IRMM)
SRM #9 - NIST-695	NIST SRM	Trace elements in multi-nutrient fertilizer, powder, CoA from vendor, not currently used
SRM #10 - polyhalite-Hz	Granular polyhalite	Prepared bulk; crushed grey polyhalite from Hazen, not sized, mix of fines to 2 mm
CoA -Certificate of Analysis		

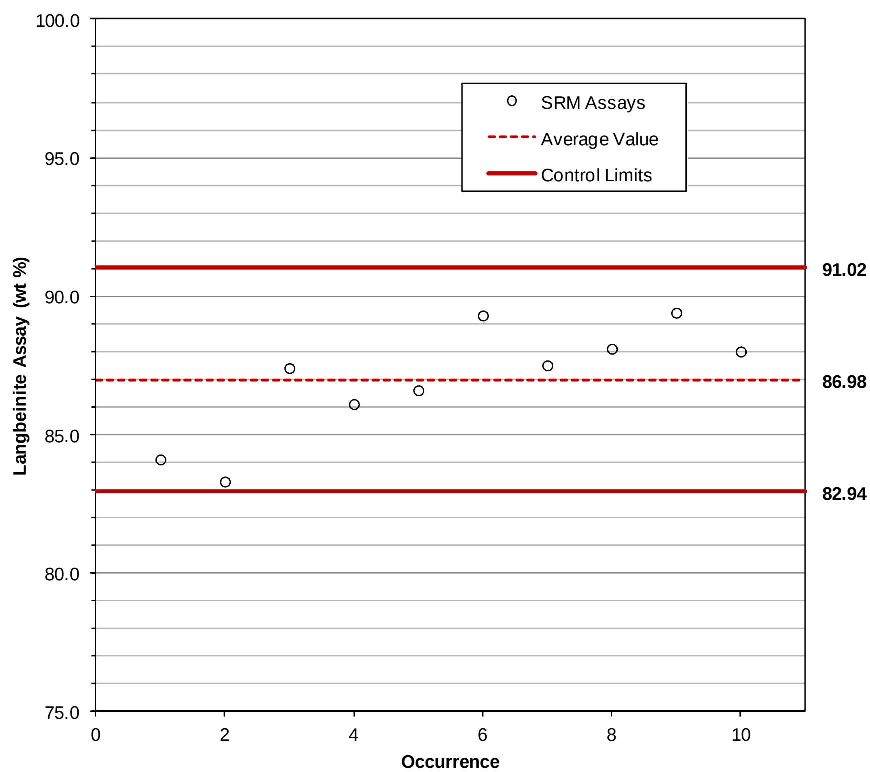


Figure 11-1. Standard Analysis for SRM #2—Langbeinite

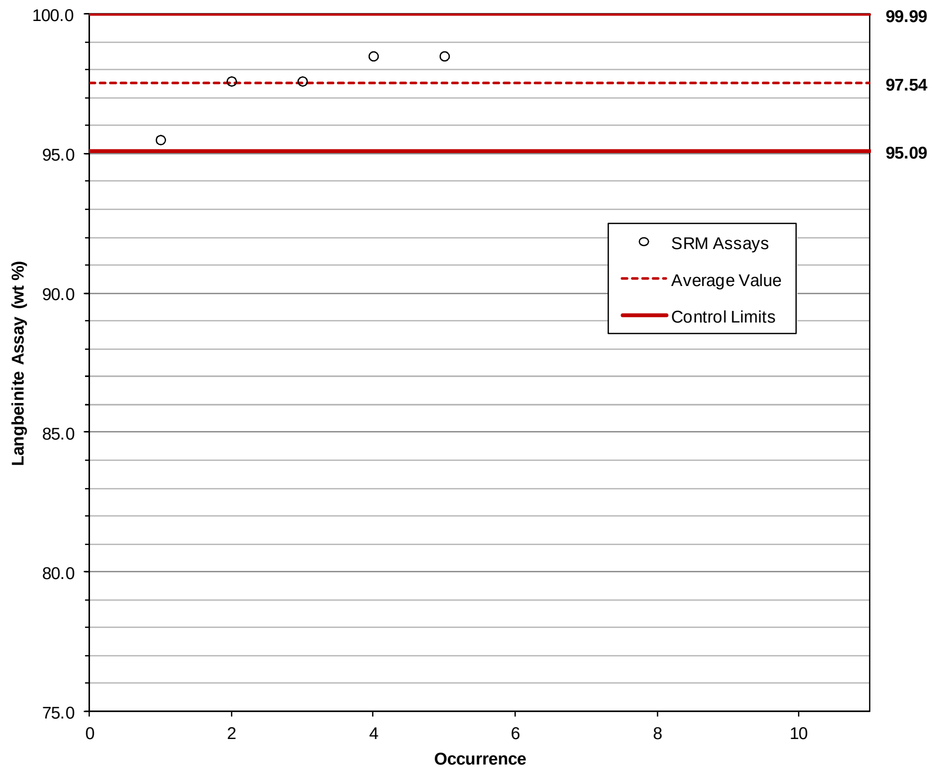


Figure 11-2. Standard Analysis for SRM #2b—Langbeinite

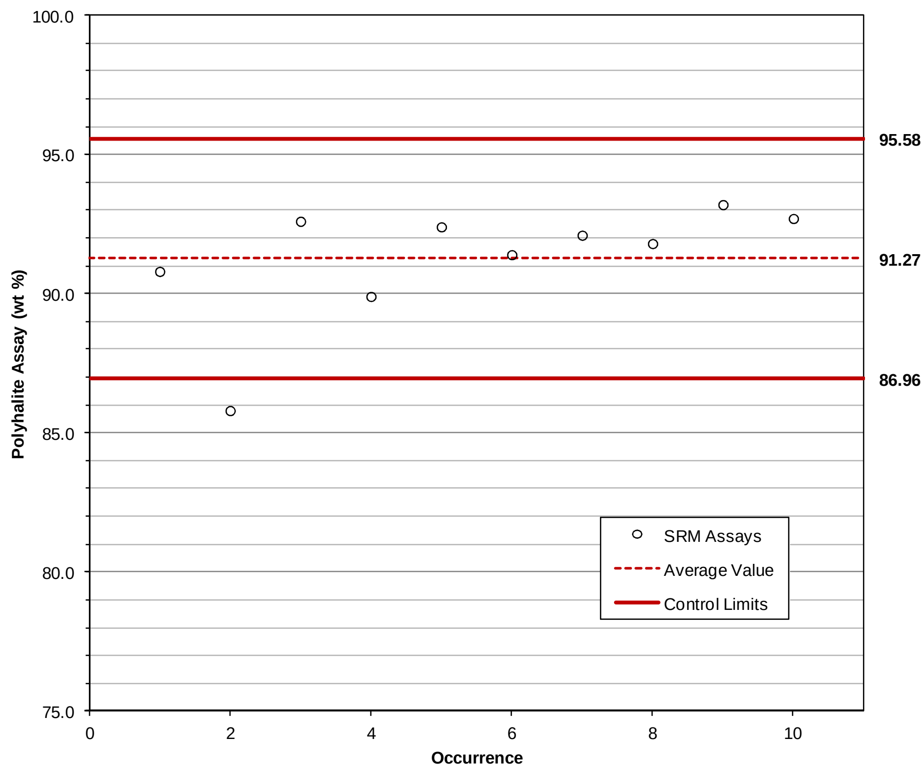


Figure 11-3. Standard Analysis for SRM #4—Polyhalite

During Phase 3A exploration, ICP inserted core duplicates at a rate of one duplicate for every ten samples submitted. Previous exploration phases were subject to varying rates of duplicate submittal. Scatter plots for XRD and XRF duplicate analysis are included as Figures 11-4 and 11-5.

XRD duplicate analysis shows good consistency between duplicate pairs, with a linear trend of y = 1.0016x and R2[2] = 0.9651. Only two of the total 64 duplicate comparisons fall outside of a conservative ±15% envelope about the x-y slope. ICP attributes the outliers specifically to inclined lamination, which results in slight variation in composition between the horizontal sample pairs, and irregular distribution of clotted halite inclusions. Industry standard for the preparation of a duplicate sample stipulates a split of the sample; ICP's procedure uses a separate sample of the quarter core.

XRF duplicate analysis (K_2O) also shows good correlation between duplicate pairs, with a linear trend of y = 0.9848x + 0.2966 and R2 = 0.985. AAI has recommended that ICP define and implement a procedure for re-assaying outliers.

11.5.3 Blanks

The primary purpose of blanks is to trace sources of artificially introduced contamination. In a chemical analysis, the measured value obtained in the absence of a specified component of a sample reflects contamination. The industry standard for a blank is quartz sand (silicate). ICP switched from a quartz blank to reagent-grade anhydrite and arcanite that are referred to as SRM (see Table 10-2). The rate of insertion is low, so is insufficient to make any conclusion as to contamination.

[2] R2 = the radius distance form the source to the sample.

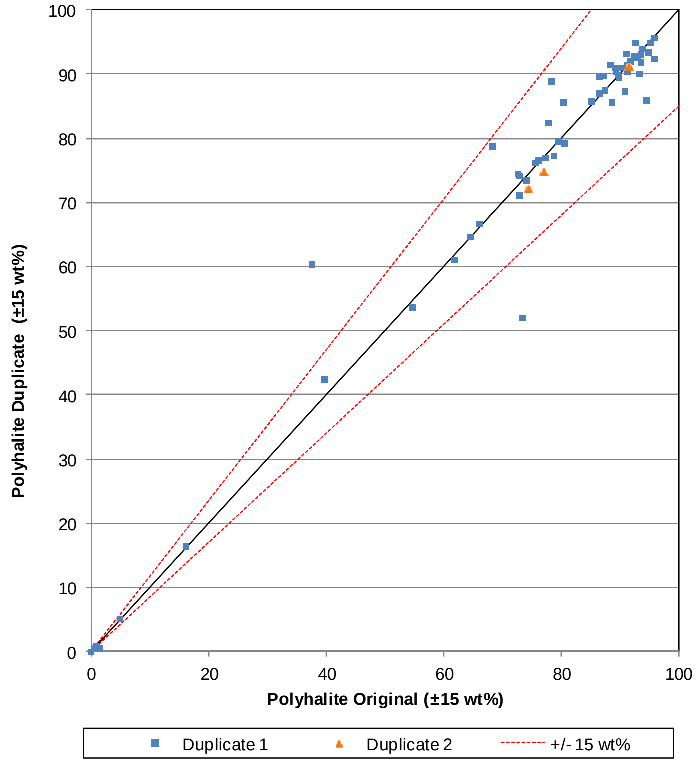


Figure 11-4. XRD Duplicate Analysis

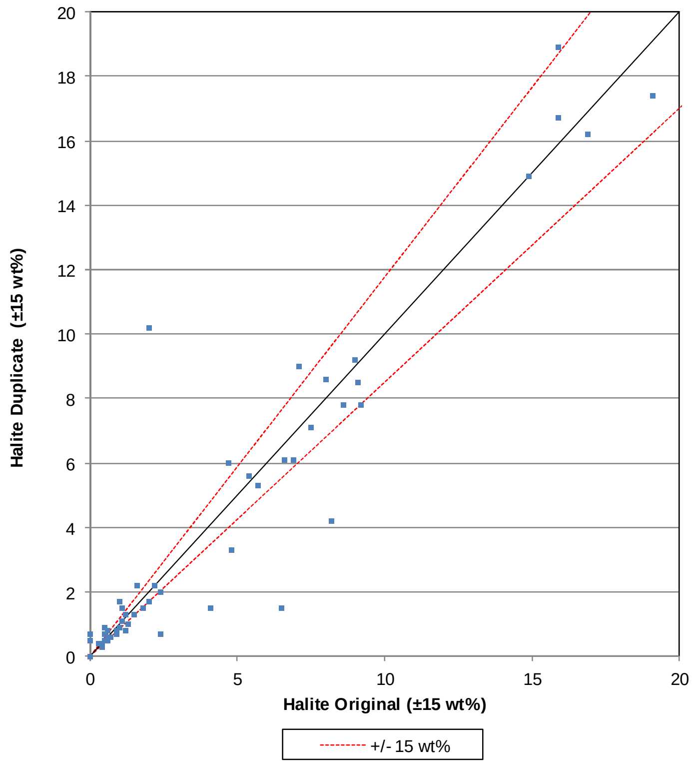


Figure 11-5. XRF Duplicate Analysis

ITEM 12: DATA VERIFICATION

12.1 Site Visits

On September 20–21, 2012, Gary Skaggs, AAI Vice President, Principal and QP, visited the site and inspected freshly recovered core at a drill rig, and also visited ICP's Hobbs office and core preparation and storage facilities. During September 25–28, 2012, Vanessa Santos, AAI Chief Geologist and QP according to NI 43-101, visited the Ochoa Project site and ICP's Hobbs office facilities. Ms. Santos was joined by AAI Vice President of Engineering and Field Services, Leo Gilbride, also a QP, visited the site on September 26–28, 2012 for an on-site inspection of the Project. During the site visit, Ms. Santos and Mr. Gilbride reviewed ICP's drilling procedures, sampling methodology and database management, sample handling and security, and core logging protocols. Ms. Santos and Mr. Gilbride also verified exploration data provided by ICP by checking logs and assay data against core samples, and field checking survey data from Phase 1 and Phase 2 and 2B drilling.

A partial list of the documents supplied by ICP and reviewed are as follows:

- OchoaDatabase5_23_13.xls
- ICP-P3a-Analytical-AsRecd-editedPO-20130519.xls
- ICP-P3a-GradeThk-IntRelease-Binder1b-20130523.pdf
- H-M_binder_2011_P1P2P2b_495pgs.pdf
- ICP_RAT_Protocols_20120706_reviewcopy.pdf
- ICP_Ochoa_CoreSampleLog_20110627.xls
- XRF Calibration Report.pdf
- XRF XRD Analysis and Results.pdf
- Quantitative Chemical Analysis by Wavelength Dispersive X-Ray Fluorescence Spectroscopy.pdf

12.2 Database Audit

AAI received the exploration drill-hole database, updated with Phase 3A drill hole and sample data, from ICP on May 25, 2013 (OchoaDatabase5_23_13.xls). The drill-hole database contained collar coordinates in Excel™ workbook format. All 32 drill holes completed by ICP were included in the database, as were XRD and XRF sample assay intervals, lithology, and QA/QC data for 645 XRF and 646 XRD sample intervals.

Initial review of the database revealed a discrepancy between drill-hole identifiers listed in the database and those included in permit documentation. The discrepancy was found to be the result of an inconsistency in naming convention. Each drill hole was assigned a number (i.e., ICP-021), or drill-hole identifier, during the layout of the drilling exploration. The drill-hole identifier (with three exceptions) did not correspond to the order in which the holes were drilled; rather, the drill hole's number in the drilling sequence was assigned as that particular hole's "alias," which was also listed in the database. There were three notable exceptions: drill holes ICP-001, ICP-002, and ICP-003 were designated in the drill-hole identifier column of the database by their number in the drilling sequence, and their original drill-hole identifier (ICP-021, ICP-022, and ICP-026, respectively) was listed as an alias. This inconsistency in naming convention is a result of the permitting process, whereby these first three holes were identified by their drill sequence number on permit application materials, rather than by their actual drill-hole identifier. For all other drill holes, permit documents reflect the original drill-hole identifier.

AAI completed a manual audit of four of the tables from the database (Header, XRD, XRF, and QA/QC) in an effort to identify errors, overlaps, gaps, total drill-hole length inconsistencies, non-numeric assay values, and/or negative numbers. A variety of minor recordkeeping inconsistencies and/or errors were identified, and all were made known to and have been adequately addressed by ICP, including nine missing standard analytical results, which were added to the database as appropriate. The survey, assay, and geology tables maximum sample depth was compared to the maximum depth reported in the collar table for each drill hole, and no intervals exceeded the reported drill-hole depths.

AAI received original H&M assay certificates in Adobe Portable Document Format (PDF) format for all samples included in the current drill-hole database. A random manual check of greater than 10% of the database against the original certificates, focusing on the polyhalite with occasional spot-checks of secondary constituents, revealed 100% accuracy for those records checked.

AAI has reviewed ICP's internal QA/QC program and believes the program justifies reasonable confidence in the reliability of the data. The pursuit of polyhalite as an economic mineral is recent within the mining industry and, therefore, no industry-recognized standard procedure for analysis exists. H&M's procedures, documentation, and internal check procedures suggest a defensible methodology; however, an independent check of H&M's methods has not yet been completed. AAI has recommended that ICP carry out check samples by a second independent check laboratory. ICP initiated checks with more than one independent commercial laboratory, but was not satisfied that the methodologies and procedure matched the original work by H&M and work was suspended.

AAI has carried out limited comparisons on polyhalite wt-% as determined by XRD plotted against K_2O wt-% as determined by XRF (Figure 12-1). Theoretical polyhalite is 15.62% K_2O. Values greater than 75% are plotted based on 31 drill-holes (354 data points) that have both XRF and XRD values for K_2O. The data shows a high bias to the XRF. Both XRD and XRF data sets were generated by H&M and suggest good precision. Without check samples, accuracy cannot be determined. As the resource was calculated based on polyhalite, the bias would not reflect an over-reporting of grade based on these data.

AAI considers the quality of data collected adequate for use in estimating the mineral resources of the Ochoa Project in support of the FS. Verification efforts confirm that the geologic and geotechnical information, survey data, and assay values included in the Ochoa database accurately represent the associated source documentation.

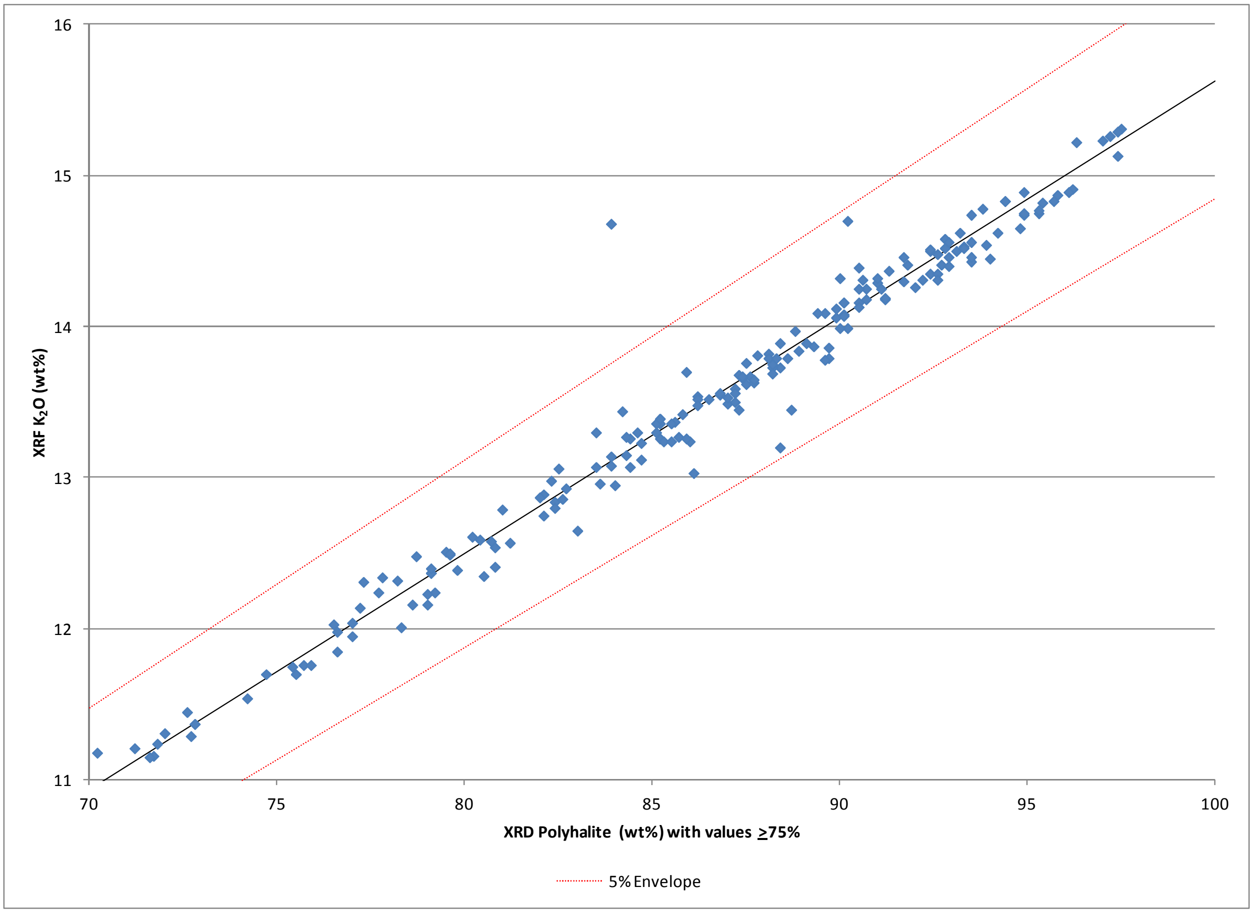


Figure 12-1. XRF and XRD Comparison Chart
(based on data from 31 drill holes with ≥75% polyhalite)

ITEM 13: MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

The process as designed for the Project involves several key unit operations to process conventionally mined ore to produce three grades of SOP: soluble, standard, and granular. Process design development began by ICP confirming that the results from prior research performed by both the USBM and PCA could be duplicated and were thoroughly understood. The knowledge and results obtained from this work by ICP formed the basis for process design and optimization. During the FS, additional test programs and trade-off studies performed by ICP as well as the operation of a pilot-scale plant conducted by ICP and HPD advanced ICP's knowledge of the process and facilitated engineering a commercial design. These programs and trade-off studies include the following:

- Detailed comparison of different crystallization circuit options
- Development and testing of a bench-scale counter-current leach circuit
- Individual parameter-specific testing of each unit operation
- Operating a complete pilot-scale plant from leaching through crystallization with actual production of SOP and leonite crystals from Ochoa ore

The results presented in this Item 13 include test results from the preliminary studies and test programs performed by ICP prior to completion of the final process flow sheets at the end of the FS.

13.2 Process Development Background

In the 1930s and 1940s, the USBM, tasked with performing scientific and engineering research regarding polyhalite processing, conducted extensive work on polyhalite processing to produce SOP. PCA operated a pilot plant in the 1950s for the production of SOP from polyhalite ore. This work formed the basis of the process that has been developed for commercialization of the Ochoa Project.

A summary of the USBM's work, along with the verification test work carried out during the PFS stage, is presented in Sections 13.2.1 and 13.2.2.

13.2.1 USBM Test Work

Extracting potash from polyhalite dates back to the early 1930s (USBM 1944). At that time, the USBM began an extensive research and development program to examine viable processing routes for the production of SOP from the Texas-New Mexican polyhalite mineral deposits (USBM 1944).

The USBM developed several processes, though one process in particular proved the most efficient and economically viable (USBM 1944). It consisted of the following unit operations:

- Conventional drill-and-blast polyhalite mining
- Grinding for size reduction for efficient washing and calcination
- Washing of the crushed ore
- Calcination of the polyhalite ore

- Dissolution (i.e., leaching) where potassium and magnesium in the ore would be dissolved in water, creating a brine
- Evaporative crystallization in outdoor solar ponds where the brine would be left to evaporate, producing SOP crystals
- Dredging SOP crystals out of the pond
- Filtering and granulating into saleable potash products

Multiple-effect forced evaporators were also considered as a method of SOP crystallization, but little research was carried out at that time (USBM 1944).

The USBM refined this process with continual research, increasing its efficiency. The optimal particle grind size for the process was determined and results showed that reducing the mined ore to –10 mesh (i.e., less than 0.0787 inch) would yield the best potassium recovery (USBM 1937).

Washing the ground ore with cold water also proved beneficial to product recovery and grade. The USBM demonstrated that sodium chloride or halite contained in the polyhalite ore could be reduced from 13% to 1% (by weight) by washing the ore with cold water, without any significant loss of potassium and magnesium (USBM 1930a, 1944). Further, the USBM found that halite removal not only increased the grade of SOP, but also eliminated potential corrosion issues with downstream equipment through the removal of harmful chloride ions (USBM 1930a).

Examination of calcined polyhalite chemistry in water at various temperatures showed that SOP could be produced faster if forced to dissolve in boiling water (USBM 1930a and b, 1933, 1944). The production rate and SOP grade were further increased by dissolving the polyhalite in a two-stage, agitated-vessel, counter-current leaching circuit (USBM 1930a, 1944, 1930). Lower grade products like syngenite and polyhalite that formed in the first stage would be dissolved by fresh water in the second stage, increasing recovery of the potassium and magnesium and producing strong solution grade (USBM 1944). The leached solids, depleted of potassium and magnesium, were discarded as waste, mainly in the form of anhydrite and gypsum (USBM 1944).

One of the most significant aspects of the work done by the USBM was determining the relevance of calcining the polyhalite prior to leaching. The calcination step involves heating the ground (–10 mesh) polyhalite to temperatures between 896°F and 968°F, causing the release of water from the mineral crystal lattice. Leaching calcined polyhalite results in a dramatic increase in the rate of potassium and magnesium dissolution, and a corresponding increase in the extraction of potassium and magnesium, when compared to leaching of non-calcined polyhalite. The USBM also found that calcining at temperatures above 968°F is detrimental to recovery and solution grade. A sintered crystal layer on the particle surface formed, which trapped potassium and magnesium inside (USBM 1930c, 1933). Similarly, calcining at temperatures below 896°F resulted in lower recovery (USBM 1930c, 1933). USBM calcining studies were carried out using a rotary kiln.

The studies conducted by the USBM reported extractions of potassium and magnesium into the leach brine upwards of 90% when using the following procedure: grinding to –10 mesh, cold water washing to remove NaCl, calcination of the washed and ground ore at a temperature between 896°F and 968°F, and two-stage counter-current leaching with water containing sodium at 212°F (USBM 1944). The efficiency of crystallization and subsequent filtration was not reported. Nevertheless, the USBM established a very solid process for extracting SOP from polyhalite.

13.2.2 Test Work Performed to Verify USBM Prefeasibility Study Work

In early 2011, Hazen was contracted by ICP to research the processing characteristics of Ochoa polyhalite as part of the PFS. The primary objectives of the research were to validate work and results observed by the USBM as well as to determine processing parameters for future utilization in the PFS.

Hazen tested drill core samples of Ochoa polyhalite. Chemical and X-ray analysis proved the ore was made up of 86% polyhalite for an equivalent of 11% potassium (Hazen 2012). Other minerals contained in the ore were magnesite (4%), anhydrite (3%), halite (2%), and undetermined (5%) (Hazen 2012). The Ochoa polyhalite is slightly higher in polyhalite grade compared to the 75% to 80% Texas-New Mexican polyhalite studied by the USBM (1944). It is also lower in halite content when compared to the 11% to 13% halite observed by the USBM (1944).

The comminution properties of polyhalite were also studied by Hazen (2012). Because of the softness of the mineral, industrial rod-mill grinding of the ore was too aggressive, producing a large amount of fines in the process. As a result, Cage-Paktors were chosen as a less aggressive method of size reduction (Hazen 2012).

Hazen also studied the NaCl removal from polyhalite by cold water washing. Ground polyhalite (–10 mesh) was tumbled in a carboy for 5 minutes in equal parts with a cold solution containing $MgSO_4$, K_2SO_4, and NaCl. The sodium content in the polyhalite was reduced by 98%, with only a 3% loss of potassium and a 5% loss of magnesium. Moreover, the efficiency of washing had no correlation to particle size, nor did washing affect particle size (Hazen 2012). Cold water washing therefore proved to be an effective method for removing halite from polyhalite.

Special attention was given to the polyhalite calcination behavior. Differential thermal analysis (DTA) and thermogravimetric analysis (TGA) of ground polyhalite showed that crystalline water is liberated at the same temperatures indicated in the USBM work. X-ray analysis suggested that at temperatures above 896°F, polyhalite breaks down further to form anhydrite and a solid solution of potassium magnesium calcium sulfate (Hazen 2012). The reaction observed is in agreement with observations by the USBM, and is considered the reaction defining polyhalite calcination.

Polyhalite was calcined by Hazen in a 4-inch-diameter by 14-inch-long kiln in order to determine the optimum calcining temperature. Calcining in the range of 896°F to 968°F was sufficient for reactions to go to completion. Hazen found that the results were in agreement with USBM observations.

Leaching calcined polyhalite was also extensively studied by Hazen (2012).

Potassium sulfate, magnesium sulfate, and calcium sulfate are initially dissolved. Higher liquid-to-solid (L/S) ratios improved dissolution and resulted in better potassium and magnesium extractions at the cost of producing less concentrated brine.

The USBM determined that when calcined polyhalite is dissolved in near atmospheric boiling water, there is the potential to precipitate polyhalite and syngenite from the solution, which can have a negative effect on recovery as both minerals contain potassium. Hazen observed the same effects. The USBM suggested, and Hazen subsequently tested, a two-stage counter-current leaching circuit to re-dissolve syngenite and polyhalite before they leave the syngenite and polyhalite leaching process in the tailings. The results demonstrated that the second stage was effective in reducing the amount of re-formed polyhalite found in the tailings.

13.3 Feasibility Test Work

Test work was conducted by ICP in 2013 during the FS phase of the Project to further define the details of the process design. Because of the limited supply of Ochoa ore, a commercially available polyhalite, referred to in this Item 13 as “gray ore,” was secured by ICP to use as needed in the test program. Detailed chemical analysis and metallurgical test work conducted by ICP determined the gray ore to be essentially identical to Ochoa ore.

The following test work was conducted by ICP during the FS:

- Polyhalite ore crushing tests were performed to confirm the equipment required to produce the particle size distribution (psd) of optimum calcinations.
- Wash tests of polyhalite ore was performed to confirm equipment required for removal of NaCl prior to calcinations.
- Calcination tests we condicuted to confirm optimum temperature range and residence time for conversion of the ore for optimal leach recoveries. The use of a fluid bed calciner was also confirmed.
- Leaching tests were conducted to provide information on retention times and final leach brine strength attainable.
- Crystallization tests were completed to validate the K_2SO_4-$MgSO_4$ phase diagram (D’Ans 1933).

Following the laboratory bench-scale program, ICP asked Hazen to complete a pilot plant demonstration for SOP production from calcined Ochoa polyhalite. Because of the limited supply of Ochoa ore, gray ore was used for grinding and fluid bed calcination studies prior to the demonstration. The leaching and SOP crystallization test circuits were commissioned using calcined gray ore, followed by operations with calcined Ochoa ore.

For the pilot plant demonstration, ICP supplied Hazen with approximately 2 t of gray ore and 100 lbs of Ochoa ore. The ores were ground to –10 mesh. The ground materials were washed, dried, screened, and blended at Hazen prior to calcining. Both ore types were calcined in an indirectly heated 4-inch fluid bed.

The pilot plant leaching circuit consisted of a two-stage counter-current setup with three leach tanks per stage. Leach slurry advanced from tank to tank by gravity overflow in a cascading staircase setup. Because only one centrifuge was available for solid-liquid separation (SLS), the circuit was operated on a semi-continuous basis. The target leaching temperature was approximately atmospheric boiling. A slurry of water and first-stage leached solids was prepared and fed to the second stage. Second-stage leach brine and dry calcined polyhalite were fed to the first stage under a controlled L/S ratio.

Each stage was operated successively using product from the previous stage. The first stage was fed with gray ore for the first five cycles, and then fed with Ochoa ore for four cycles. The first-stage brines derived from the gray and Ochoa ores were stored separately for the leonite dissolver and SOP crystallization operations.

The pilot plant demonstrated the leaching of calcined gray ore and Ochoa polyhalite in a two-stage counter-current leach circuit. All first-stage feed rates were very close to their intended targets.

The brines generated from the leach circuit were treated in a manner consistent with the process flow sheet to produce SOP crystals.

13.4 Full-Scale Pilot Plant

Together with Veolia, Hazen, Gundlach and Metso, ICP conducted a large-scale pilot plant study. ICP supplied Hazen with approximately 22 t of gray ore and 4,000 lbs of Ochoa ore for the large-scale pilot plant activities.

13.4.1 Crushing and Washing

Both gray ore and Ochoa ore (from cores) were crushed to –10 mesh and then washed with water to remove NaCl. Both ores were dried and packaged to be sent for further processing.

13.4.2 Calcination

The gray ore and the Ochoa ore were calcined over several campaigns in a fluid bed rented from Metso.

13.4.3 Leaching

Hazen fabricated the leaching skids for the pilot plant. Two identical skids were manufactured with one representing the first-stage leach and the other representing the second-stage leach.

13.4.4 Pilot Plant Crystallization

The crystallization tests were carried out in three phases. The first phase included the following steps:

- Polyhalite seed preparation
- Ore leaching
- Leonite dissolution (or simulation)
- Evaporative concentration of the brine after leonite dissolution
- Recovery of the polyhalite slurry

The second-phase testing involved SOP crystallization and subsequent centrifugation and SOP drying. The third phase included the leonite crystallization from the SOP mother liquor and subsequent centrifugation and leonite drying.

13.5 SOP Granulation

Granulation experiments were conducted by FEECO International to obtain data on the granulation parameters associated with SOP. This included the type and morphology of granules generated, as well as potential binders. Four series of tests were conducted using different sized raw material. The material used for the test work was purchased raw soluble grade SOP.

ICP will conduct future granulation test work, using different quantities and types of binders for SOP granulation to determine parameters for selecting the type of binder for plant operation and the optimal amount necessary to obtain market-grade granulated SOP.

13.6 Water Quality

ICP commissioned HWCC to perform bench-scale testing on the Capitan Reef water studies to demonstrate the treatment of a water sample received from the Capitan.

HWCC performed High Recovery Membrane (HRM) and Interstage Precipitation Reactor (IPR) bench testing on a sample of Capitan Reef well water. Two stages of HRMs were utilized with IPR treatment performed between the HRM stages.

13.7 Summary of Mineral Processing Test Work

It is the author's opinion that, after reviewing the detailed information provided to the author by ICP and ICP's various consultants, appropriate mineral processing test work has been conducted by ICP and its consultants in order to define the equipment required for the unit operations of:

- Polyhalite ore crushing
- Removal of NaCl from polyhalite ore
- Calcination of polyhalite ore
- Leaching of calcined polyhalite ore for recovery of K_2SO_4 into a leach brine
- Crystallization of SOP from a leach brine

The author, in preparing this Item 13, has reviewed the detailed information with respect to mineral processing and metallurgical test work, provided by ICP and its various consultants, and is satisfied that the test work conducted and the interpretation of the results obtained, all as describe in this Item 13, have been carried out and recorded in accordance with best practices and meet generally accepted professional standards.

ITEM 14: MINERAL RESOURCE ESTIMATES

14.1 General

The mineral resource for the Property comprises polyhalite mineralization within the Ochoa polyhalite bed, which is contained in the Tamarisk Member of the Rustler Formation. The Ochoa polyhalite bed occurs over most of the Property, with the exception of various detached leases to the east. The mineralization occurs as a generally undisturbed, flat-lying bed ranging between 4 and 6 ft thick inside the margins of the depositional basin. The bed dips gently to the southeast within the boundaries of the Property, flattening from a dip of up to 2° in the north to less than 0.5° in the south. Local steepening can occur at the basin margins.

The Ochoa polyhalite bed is the subject of this TR. This section identifies that portion of the Ochoa bed which qualifies as an NI 43-101 Mineral Resource and Mineral Reserve. The Mineral Reserve represents that portion of the Mineral Resource projected to be recoverable by the room-and-pillar mine plan developed in this study.

The Ochoa bed is estimated to contain a 1,017.8-Mt Measured plus Indicated Resource at an average grade of 83.9 wt-% polyhalite ($K_2Ca_2Mg(SO_4)_4 \cdot 2H_2O$),[3] based on core drilling and core chemical analyses from 32 exploration holes drilled by ICP from December 2009 through April 2013. Another 855 petroleum wells in the area provided supplemental definition of bed thickness and continuity from wireline geophysical logs.

No other Exploration Targets, Mineral Resources, or Mineral Reserves are known to exist on the Property at this time.

14.2 Polyhalite Resources

14.2.1 Methodology

The location of the 32 polyhalite exploration holes and 855 petroleum wells[4] used in the resource estimate are shown in Figure 14-1. The figure also shows the structural elevation contours of the Ochoa bed, as defined by the combined potash boreholes and petroleum wells. To the southwest and northeast, the Mineral Resource is limited by the margin of the depositional basin, which generally coincides with the Property boundaries. The Ochoa bed persists beyond the Property to the northwest and southeast along the axis of the basin. The Mineral Resource is limited by the Property boundaries in those directions.

Sample assays for the 32 exploration holes were compiled by ICP in a computer-based Microsoft Excel™ spreadsheet and provided to AAI for resource modeling.[5] Core recovery was sufficiently high in all holes to support accurate assay compositing. Values within the assay database were spot-checked against Certificates of Analysis and found to be of sufficient accuracy for resource modeling. Drill-hole collar coordinates were surveyed by a Licensed Professional Surveyor (LPS) and provided in State Plane North American Datum of 1983 (NAD83) coordinates.

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[3] Pure polyhalite equivalent to 28.89% potassium sulfate (K_2SO_4).
[4] Petroleum wells only used to estimate bed thickness and elevation.
[5] Exploration hole database updated by ICP through 25 May 2013, including Phase 3A analytical data, and provided to AAI on 25 May 2013.

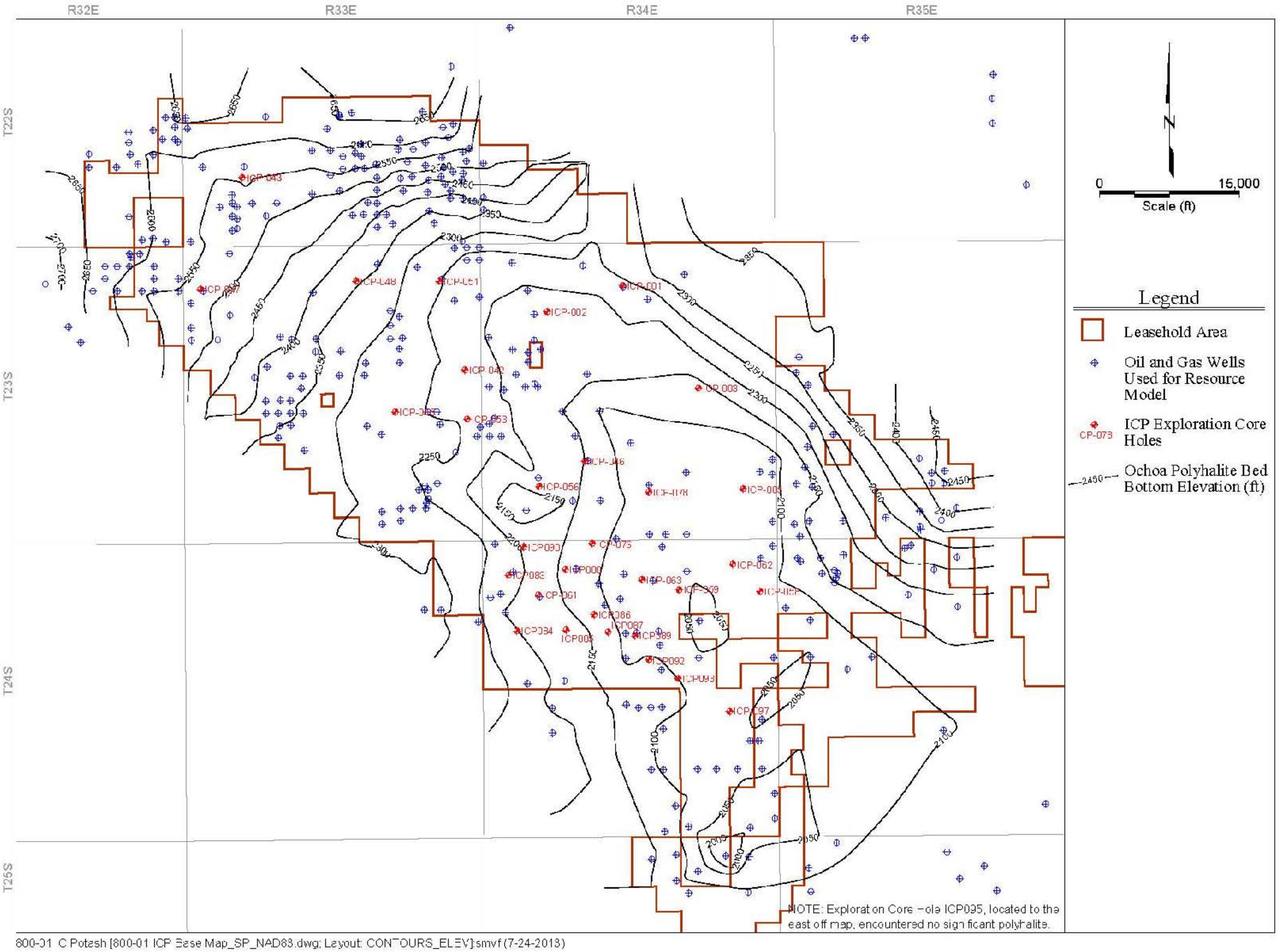


Figure 14-1. Plan Map of Ochoa Polyhalite Bed Structure and Exploration Holes

Downhole directional surveys of the drill holes were conducted. The borehole deviations were insignificant; therefore, all holes were treated as true vertical holes. The cored thickness was treated as the true bed thickness in all holes. Any difference between cored and true thickness is estimated to be negligible considering the near-flat dip of the bed and vertical-drilling character of the overburden.

Seam correlations were made using Carlson Mining 2013 Software™ (Carlson 2013), an industry-recognized commercial-grade geologic and mine modeling software system that runs within AutoDesk Inc.'s AutoCAD 2013©. Strong continuity of the Ochoa bed was evident in all reporting holes across the Property. Ochoa bed tops and bottoms were picked from the sample assays and, in some holes, corroborated against natural gamma logs.

The core was ordinarily sampled and assayed on 0.30-ft to 0.60-ft lengths. ICP geologists attempted to split samples at lithologic contacts at the top and bottom of the polyhalite bed to increase assay resolution wherever possible.

Quality parameters were composited as length-weighted averages of the individual sample assays over the polyhalite thickness. The polyhalite bed top and bottom contacts were generally sharp and readily distinguishable by an abrupt drop in polyhalite grade. Where gradational, contacts were defined by a 50.0% polyhalite cutoff applied to the individual samples. In rare instances, a sub-50.0% polyhalite interval was included in the bed composite where the bed was split by the lower-grade interval and the presence of the split did not significantly diminish the composite grade of the bed. Composite values for the drill holes used in the resource estimate are summarized in Table 14-1.

The drill-hole composites were applied to a gridded-seam model using Carlson Mining's Geology Module 2013 for calculating the resource tonnage and grade parameters. The bed was gridded into a single layer of 500-ft-square blocks of variable vertical thickness representing the local thickness of the zone. Block thickness and grade values were estimated from neighboring drill holes (point data) using ordinary kriging models. Kriging was selected in all cases because it provided the most reliable, statistically unbiased estimator where sufficient spatial data were available.

Semivariograms of zone thickness and quality parameters, including polyhalite grade, were generated from the drill-hole composite data. An anisotropic semivariogram model was developed for bed thickness because directionality was evident in the data. Spatial continuity is dominant along the NW-SE strike of the basin. The directional semivariograms used in the resource estimate are illustrated in Figure 14-2. Comparison of the thickness semivariograms reveals a minor-axis:major-axis "range" anisotropy ratio of approximately 0.46 between the NW-SE and NE-SW directions.

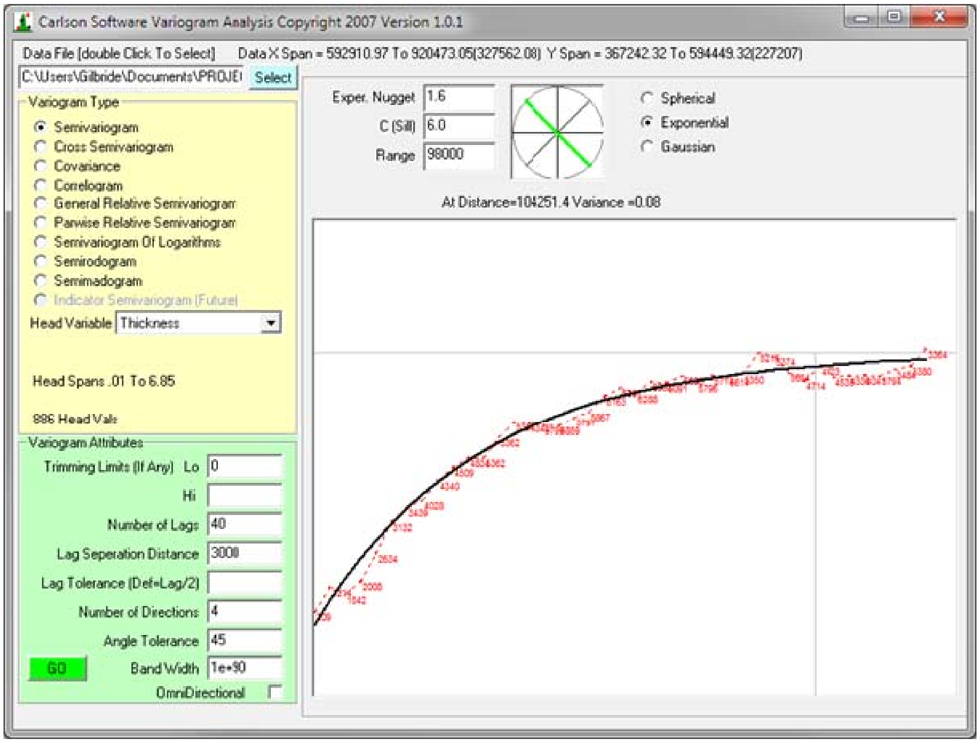
Omni-directional semivariogram models were developed for the principal quality parameters—polyhalite, anhydrite ($CaSO_4$), halite (NaCl), and magnesite ($MgCO_3$)—based on up to 31 ICP boreholes reporting assays. Gypsum ($CaSO_4 \cdot 2H_2O$) content was negligible in all reporting holes and was not modeled. The quality data lacked sufficient density to discern any directional trends. The omni-directional spherical model semivariogram for composite polyhalite grade is shown in Figure 14-3. All semivariogram parameters used for resource modeling are summarized in Table 14-2. Table 14-2 is based on an exploration data cutoff of May 25, 2013. The maximum number of data points used for estimation was limited to the closest 10 points within a radius of influence (ROI) of 10.0 miles.

Table 14-1. Drill Hole Assay Composites Used for Mineral Resource Estimation—Ochoa Polyhalite Bed†

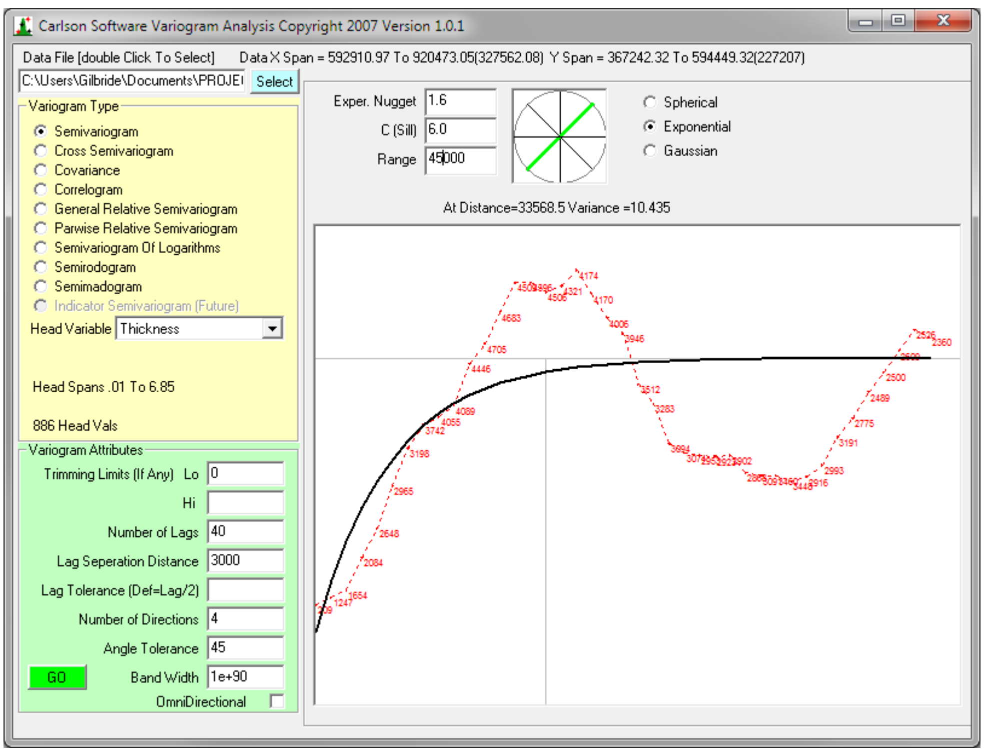
Drill Hole ID	Easting State Plane NAD83	Northing State Plane NAD83	Collar Surface Elevation (ft)	Bed Top Depth (ft)	Bed Thickness (ft)	Polyhalite (wt %)	Equivalent K_2SO_4 (wt %)	Anhydrite (wt %)	Gypsum (wt %)	Halite (wt %)	Magnesite (wt %)
ICP-005(016/027)	789,877.1	462,426.8	3,627.1	1,546.2	5.45	91.7	26.5	2.23	0.01	1.74	4.33
ICP-021(001)	776,952.0	484,139.9	3,633.0	1,394.7	5.40	84.0	24.3	3.63	0.01	3.15	8.01
ICP-022(002)	768,852.7	481,330.5	3,700.7	1,524.4	4.28	83.7	24.2	3.60	0.01	1.98	9.38
ICP-026(003)	785,071.0	473,170.6	3,690.1	1,554.7	4.05	80.0	23.1	4.72	0.01	3.93	11.31
ICP-042(007)	760,044.1	475,098.6	3,726.8	1,488.0	5.26	85.8	24.8	5.57	0.01	1.67	5.69
ICP-043(005)	736,170.9	495,629.4	3,561.3	992.1	6.28	81.7	23.6	3.63	0.01	5.81	8.25
ICP-045(008)	752,561.3	470,629.7	3,692.4	1,420.9	5.65	79.4	22.9	3.97	0.01	6.40	8.97
ICP-046(014)	772,896.7	465,395.3	3,673.5	1,519.3	5.50	86.0	24.9	3.57	0.01	2.91	7.49
ICP-047(004)	731,687.8	483,716.6	3,519.4	975.0	4.26	80.7	23.3	7.12	0.01	2.32	8.02
ICP-048(009)	748,357.4	484,609.0	3,677.5	1,318.7	4.88	77.8	22.5	5.37	0.01	3.44	12.41
ICP-051(006)	757,275.1	484,597.8	3,747.0	1,483.5	5.28	79.8	23.1	2.72	0.01	5.02	11.58
ICP-053(015)	760,294.6	469,862.4	3,693.5	1,440.7	5.15	88.3	25.5	2.10	0.01	1.69	7.96
ICP-056(013)	768,016.4	462,690.2	3,666.9	1,495.0	5.95	89.5	25.9	2.99	0.01	2.77	4.75
ICP-058(019)	791,673.5	451,511.8	3,624.1	1,534.5	4.50	80.9	23.4	2.79		4.91	12.04
ICP-059(020)	783,012.4	451,667.6	3,607.8	1,538.6	4.90	84.7	24.5	2.05	0.01	3.58	9.71
ICP-061(012)	767,904.1	451,133.2	3,627.1	1,451.8	6.25	92.4	26.7	1.97	0.01	1.76	3.88
ICP-062(010)	788,750.9	454,398.9	3,631.9	1,562.6	5.73	81.6	23.6	8.24	0.01	1.49	7.06
ICP-063(011)	779,035.3	452,735.3	3,587.3	1,508.3	4.23	80.0	23.1	5.40	0.02	5.60	9.01
ICP-076(018)	773,682.7	456,577.6	3,657.2	1,535.7	4.55	83.0	24.0	7.63	0.01	0.72	8.69
ICP-078(017)	779,760.6	462,099.9	3,664.5	1,572.9	5.15	80.7	23.3	1.74	0.01	6.75	10.79
ICP-083(028)	764,713.8	453,206.8	3,634.9	1,407.4	5.15	89.0	25.7				5.67
ICP-084(027)	765,606.3	447,289.7	3,576.8	1,370.7	5.35	89.3	25.8				5.44
ICP-085(024)	770,882.9	447,366.8	3,595.6	1,410.3	5.52	85.7	24.8				8.15
ICP-086(023)	773,873.8	449,021.3	3,609.4	1,481.5	5.57	90.1	26.0				4.45
ICP-087(022)	775,388.3	447,169.9	3,610.3	1,477.4	6.10	90.3	26.1				4.77
ICP-088(030)	770,818.7	453,836.7	3,651.7	1,506.9	5.05	88.5	25.6	4.32			
ICP-089(021)	778,291.1	446,749.5	3,614.6	1,515.2	5.60	85.2	24.6				6.28
ICP-090(029)	766,339.1	456,255.9	3,648.0	1,447.6	5.75	91.2	26.4				4.53
ICP-092(032)	779,789.6	444,238.6	3,624.1	1,516.6	5.03	91.1	26.3				5.10
ICP-093(031)	782,886.6	442,231.0	3,599.9	1,492.9	4.87	90.0	26.0				5.56
ICP-095(025)	851,048.1	438,539.3	3,364.0	1,520.2	Negligible polyhalite bed present						
ICP-097(026)	788,417.8	438,679.0	3,581.9	1,507.2	4.60	79.0	22.8	4.67			9.25

† Mineral Resource also includes bed thickness defined by elogs from 855 petroleum wells.

Table is based on geologic and assay data received by AAI by May 24, 2013. Data developed after that date was not used in determining Resources or Reserves and will be in included in subsequent reports.



a) Bed Thickness (feet)—NW-SE Direction



b) Bed Thickness (feet)—NE-SW Direction

Figure 14-2. Directional Exponential Semivariograms—Ochoa Polyhalite Bed Thickness

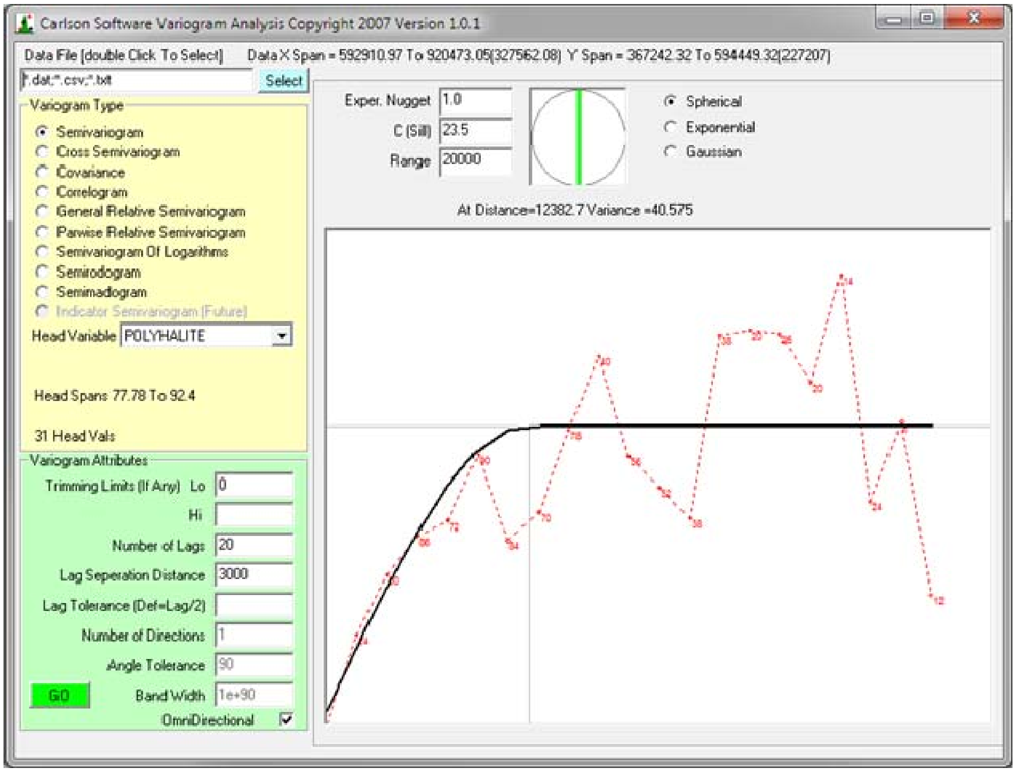


Figure 14-3. Omni-Directional Spherical Semivariogram—Ochoa Bed Polyhalite Grade

Table 14-2. Resource Model Kriging Parameters—Ochoa Polyhalite Bed

Variable	Exploration Holes Used	Semivariogram Model Type	Nugget	Sill	Range (ft)	Orientation	Major Axis Anisotropy Azimuth	Range Anisotropy Ratio	Search Radius (miles)	Point Data Limit
Bed thickness (ft)	886	Exponential	1.60	6.00	98,000	Directional	N45°W	0.46	10.0	10
Polyhalite (wt %)	31	Spherical	1.00	23.50	20,000	Omnidirectional			10.0	10
Anhydrite (wt %)	22	Spherical	1.00	3.00	4,000	Omnidirectional			10.0	10
Halite (wt %)	20	Spherical	1.00	3.00	6,000	Omnidirectional			10.0	10
Magnesite (wt %)	30	Spherical	1.00	6.30	20,000	Omnidirectional			10.0	10

Figures 14-4 and 14-5 are contour maps of the modeled thickness and composite polyhalite grade contours for the Ochoa bed.

Grids were created for top and bottom elevations of the polyhalite bed based on drill-hole intercept elevations, including all potash boreholes and petroleum wells. Standard triangulation was used for grid estimation. A bed overburden (depth) grid was created by subtracting the respective ground surface and top-of-bed elevation grids. The surface elevation grid was generated from a commercially available United States Geological Survey (USGS) 7.5-minute digital elevation model. The relatively flat structure and surface topography is illustrated in the 3D resource model in Figure 14-6.

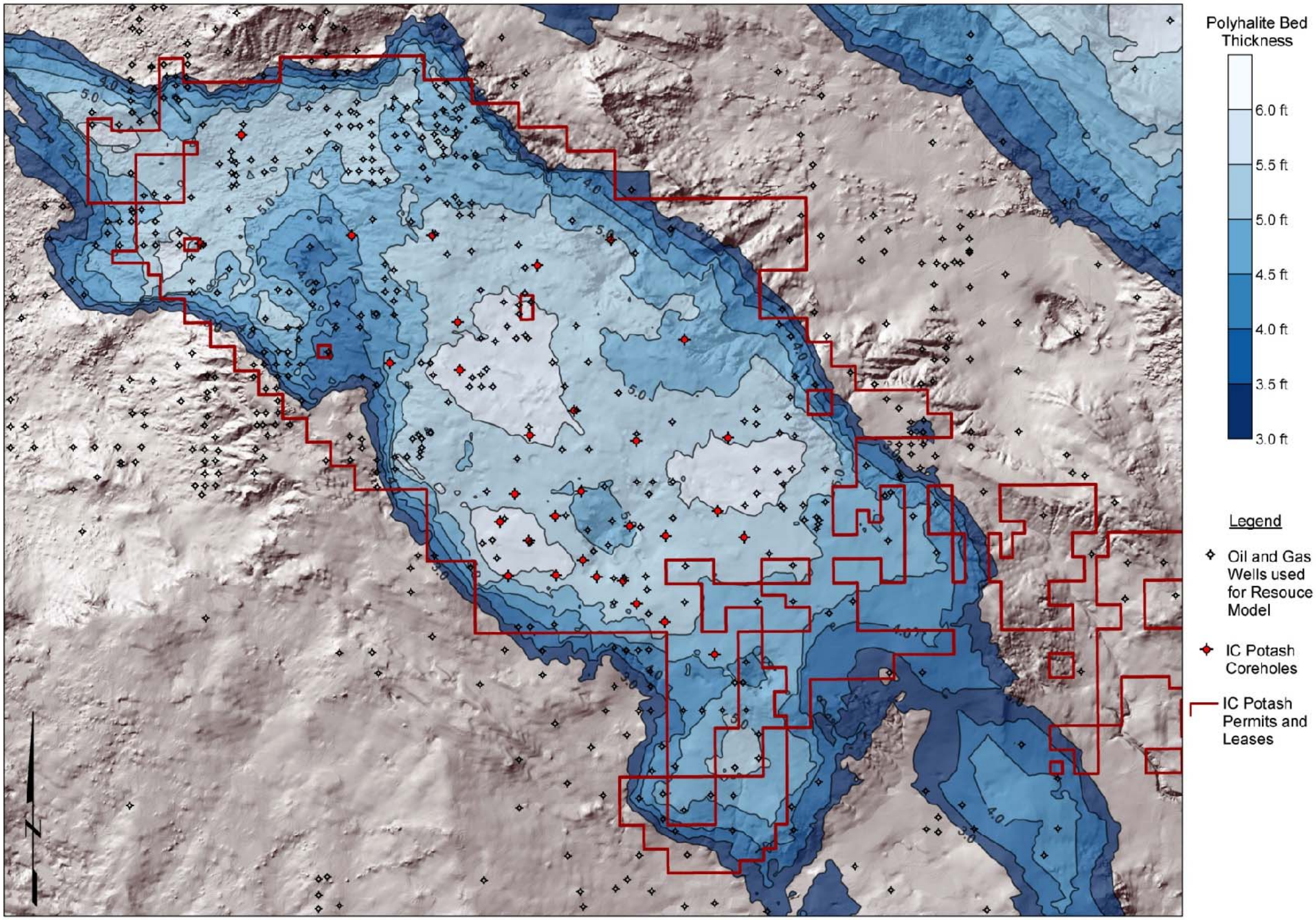


Figure 14-4. Thickness Contours—Ochoa Polyhalite Bed

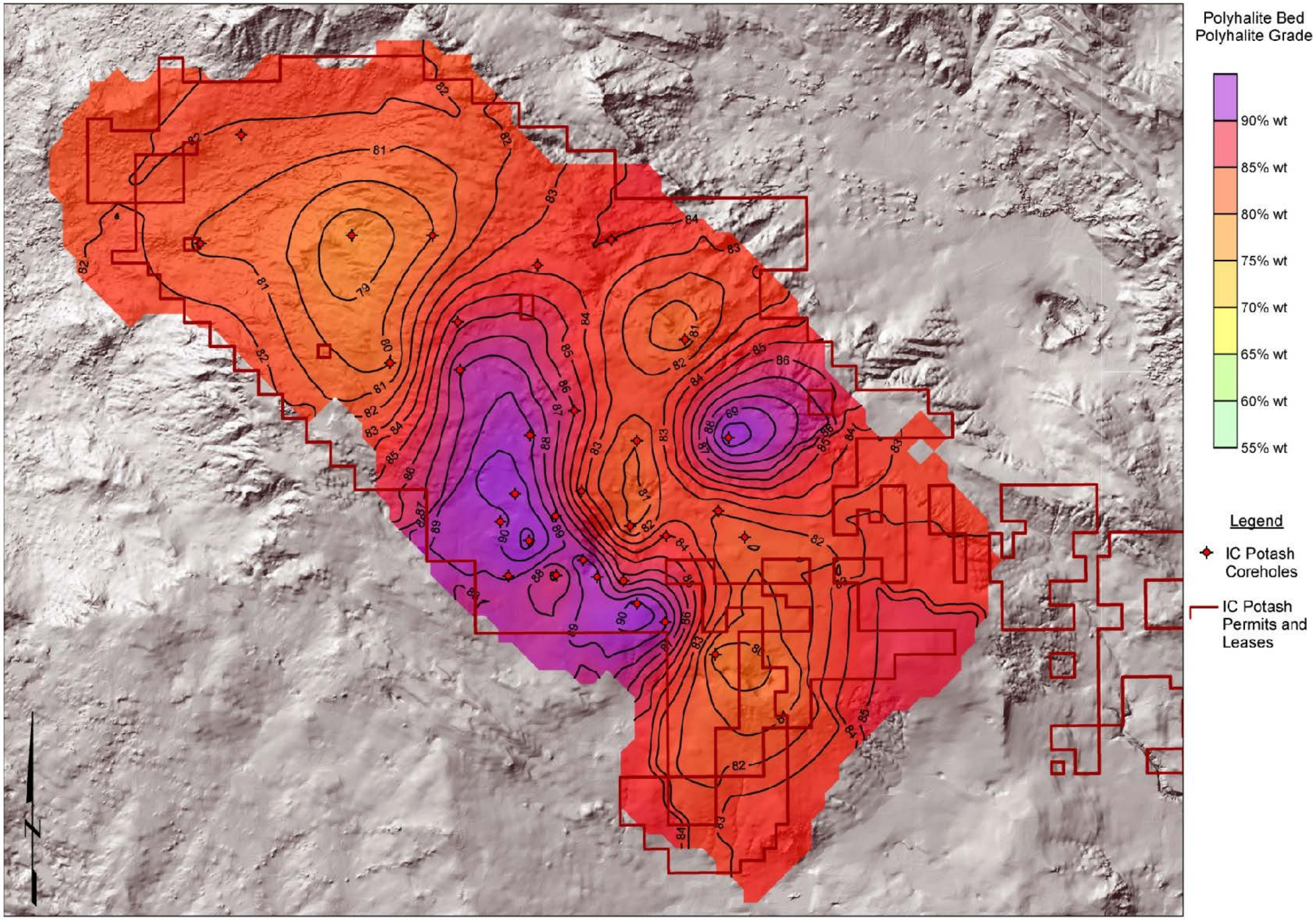


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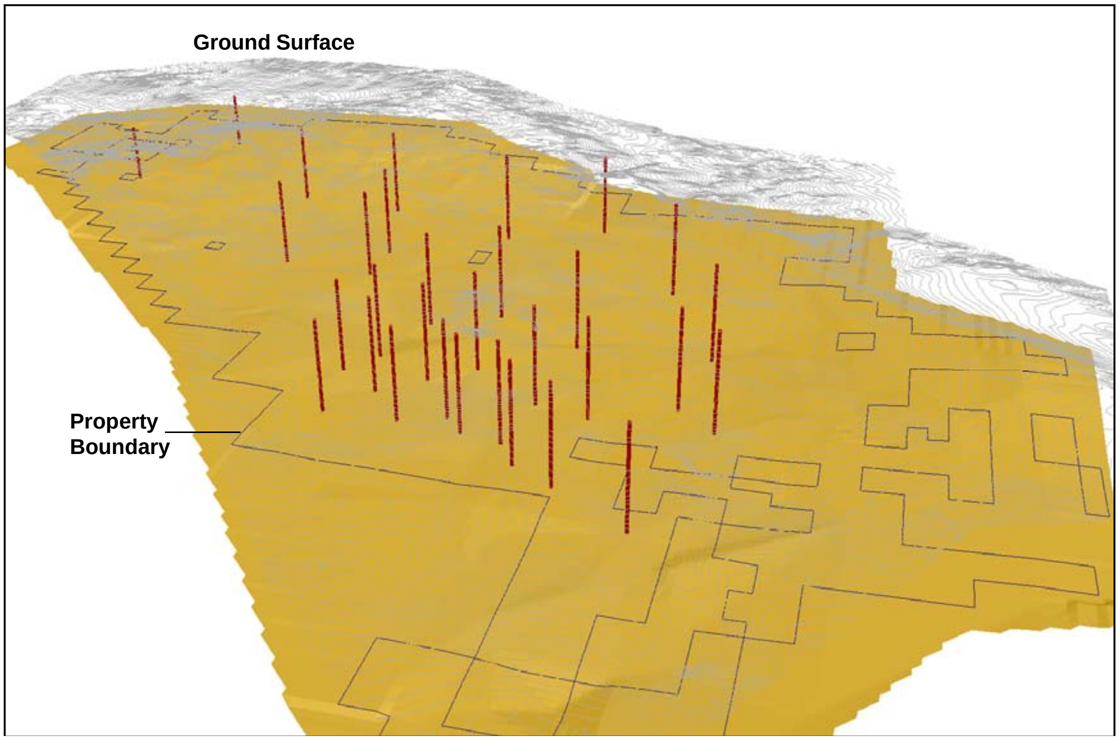
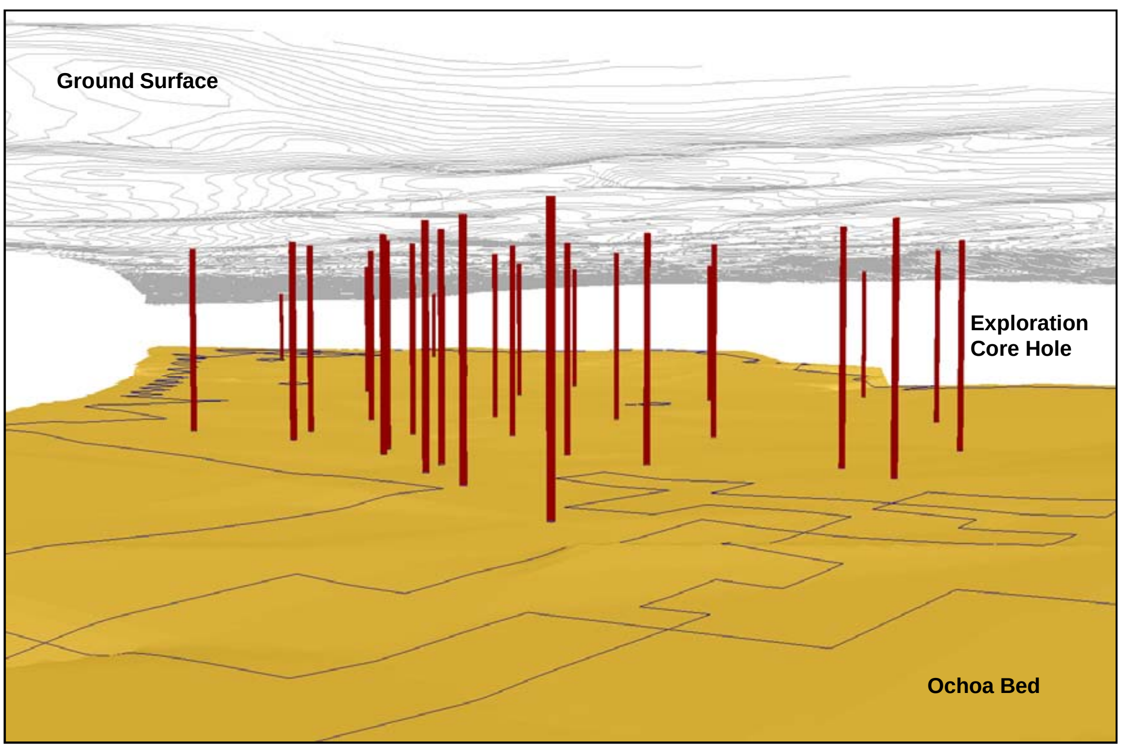
Figure 14-5. Polyhalite Grade Contours—Ochoa Polyhalite Bed



a) Isometric Subsurface View to Northwest



b) Isometric Subsurface View Updip to Southeast

Figure 14-6. Three-Dimensional Model of the Ochoa Polyhalite Bed

Polyhalite tonnages are based on an average bulk density of 173.5 tons per cubic foot (t/ft^3) derived by Chemrox and Gustavson (2009) from core hole density tests and process and rock mechanics tests conducted in 2009 and additional tests conducted in 2011 (Gustavson 2011a). AAI rock mechanics testing completed in 2012 and 2013 confirms a similar range of polyhalite bulk densities which supports the Gustavson average bulk density. Rock mechanics testing indicates that the natural moisture content of the polyhalite is negligible (i.e., typically <1%).

In addition to the main polyhalite bed, quality grids (polyhalite, anhydrite, halite, and magnesite) were calculated using an inverse distance squared (ID^2) algorithm for 0.5-ft layers extending to a depth of 2.0 ft beyond the polyhalite bed into the roof and floor. The grids were based on composited assays for the 0.5-ft layers. The roof and floor grids were applied to estimating out-of-seam dilution (OSD) as part of the Mineral Reserves analysis.

14.2.2 Definitions and Applicable Standards

For this report, AAI, in accordance with NI 43-101 requirements, has used the definitions of “resource” and “reserve” as published in the CIMDS that were adopted November 27, 2010 (CIM 2010). In this standard, a **Mineral Resource** is defined as

> *…a concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized organic material including base or precious metals, coal, and industrial minerals in or on the Earth’s crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.*

A Mineral Resource is not an inventory of all mineralization drilled or sampled, regardless of cutoff grade, likely mining dimensions, location, or continuity. A Mineral Resource is a realistic inventory of mineralization which, under assumed and justifiable technical, economic, and development conditions, might, in whole or in part, become economically extractable.

A **Mineral Reserve** is defined as

> *…the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This study must include adequate information on mining, processing, metallurgical, economic and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified. A Mineral Reserve includes diluting materials and allowances for losses that may occur when the material is mined. A Mineral Reserve is subdivided into two classes, “proven” and “probable,” with the level of confidence reducing with each class respectively.*

The CIMDS provides for a direct relationship between Indicated Mineral Resources and Probable Mineral Reserves, and between Measured Mineral Resources and Proven Mineral Reserves. Inferred Mineral Resources cannot be combined or reported with other categories.

Mineral Resources are subdivided into classes of Measured, Indicated, and Inferred, with the level of confidence reducing with each class, respectively. Polyhalite resources are reported as in-situ tonnage and are not adjusted for mining losses or mining recovery. Property-specific criteria applied to the Mineral Resource classifications by the authors of this report are listed under the CIMDS definitions (CIMDS definitions in italics), as follows:

> **Inferred Mineral Resource**—*that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes.*
>
> <u>Property-specific criteria for classification</u>: Polyhalite mineralization located between 1.5 miles and 3.0 miles of an exploration core hole reporting assays and with a bed thickness equal to or greater than 4.0 ft and a composite grade equal to or greater than 65.0% polyhalite by weight.
>
> **Indicated Mineral Resource**—*that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics, can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed.*
>
> <u>Property-specific criteria for classification</u>: Polyhalite mineralization located between 0.75 miles and 1.5 miles of an exploration core hole reporting assays and with a bed thickness equal to or greater than 4.0 ft and a composite grade equal to or greater than 65.0% polyhalite by weight.
>
> **Measured Mineral Resource**—*that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough to confirm both geological and grade continuity.*
>
> <u>Property-specific criteria for classification</u>: Polyhalite mineralization located within 0.75 miles of an exploration core hole reporting assays and with a bed thickness equal to or greater than 4.0 ft and a composite grade equal to or greater than 65.0% polyhalite by weight.

CIMDS states that for the reporting of industrial mineral resources and reserves, issuers are to use the above definitions. CIM provides further guidance on reporting practice under Best Practice Guidelines for Industrial Minerals adopted by CIM Council on 23 November 2003 (CIM 2003).

Table 14-3 summarizes the resource classification criteria applied to the Mineral Resource defined in terms of equivalent radial distance (or ROI) around a drill hole.

Table 14-3. Resource Classification Criteria Applied to the Ochoa Polyhalite Bed

Resource Classification	Composite Grade Cutoff	Bed Thickness Cutoff	Distance from Drill Hole
Measured	65.0% Polyhalite	4.0 ft	Located within 0.75-mile radius from an exploration hole
Indicated	65.0% Polyhalite	4.0 ft	Located between 0.75-mile and 1.5-mile radius from an exploration hole
Inferred	65.0% Polyhalite	4.0 ft	Located between 1.5-mile and 3.0-mile radius from an exploration hole

Resource cutoffs of a 4.0-ft bed thickness and 65.0% polyhalite grade are considered reasonably conservative lower limits for potentially economic conventional underground mining in the Ochoa bed. A 65.0% polyhalite cutoff is equivalent to 10.0% K_2O, which is an economically reasonable cutoff commonly applied to potassium projects. These resource cutoffs do not preclude the possibility that thinner and/or lower grade polyhalite could be mined locally and remain economic as part of a larger mining operation.

Table 14-4 summarizes the Ochoa bed polyhalite Mineral Resource for the Property. The resource is reported on a dry tonnage basis. Resource classification areas are shown in plan view on the map in Figure 14-7.

Mineral Resources are inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves have not demonstrated economic viability.

Table 14-4. Ochoa Project Mineral Resource (effective date 31 May 2013)

	Average Thickness (ft)	Resource Area (acres)	In-Place Tons[1,2,3] (millions)	Polyhalite (wt %)	Equivalent K_2SO_4 (wt %)	Anhydrite (wt %)	Halite (wt %)	Magnesite (wt %)
MEASURED[4]	5.2	26,166	511.7	84.5	24.4	4.02	3.27	7.94
INDICATED[5]	5.0	26,698	506.0	83.3	24.1	4.00	3.30	8.61
TOTAL M&I	5.1	52,865	1,017.8	83.9	24.2	4.01	3.28	8.27
INFERRED[6]	4.8	15,634	284.0	82.6	23.9	4.11	3.37	8.82

[1] Average in-situ bulk density of 173.5 pounds per cubic foot (pcf).

[2] Bed thickness cutoff 4.0 ft, composite grade cutoff 65.0% polyhalite, excludes out-of-seam dilution.

[3] Mineral Resource includes Mineral Reserves.

[4] Measured Resource located within 0.75-mile radius from an exploration core hole.

[5] Indicated Resource located between 0.75-mile and 1.5-mile radius from an exploration core hole.

[6] Inferred Resource located between 1.5-mile and 3.0-mile radius from an exploration core hole.

Note: Gypsum weight percent negligible for all resource classifications.

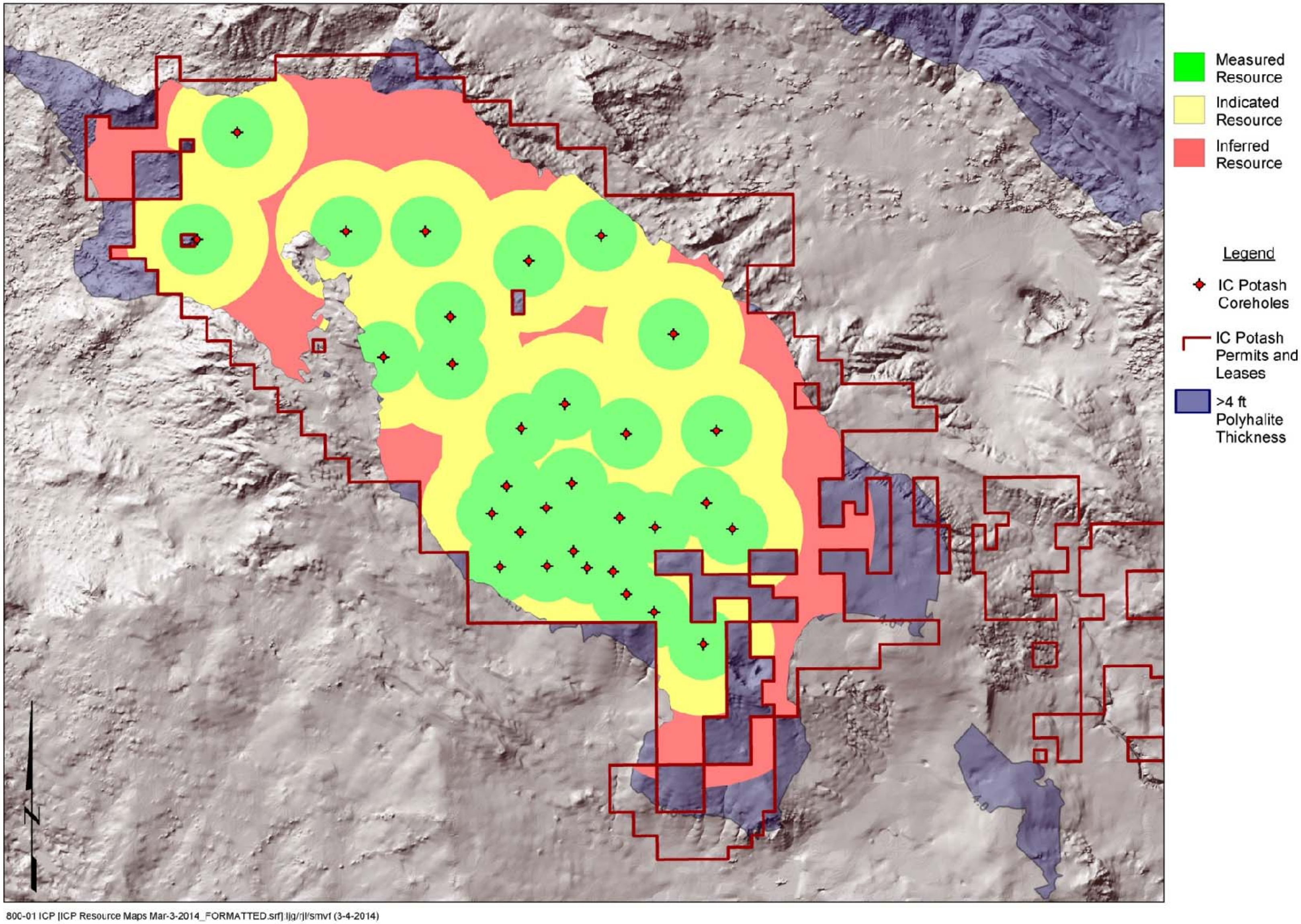


Figure 14-7. Mineral Resource Classification Areas—Ochoa Polyhalite Bed

No reduction has been applied to the resource for possible undiscovered localized geological features, including faults, scours, channels, and other structural disturbances which may or may not affect economic mining. The presence of such structures at the prospective mining horizon and the extent to which these features could impact mining are risk factors. The relatively flat structure indicated by the high density of petroleum wells across the Property suggests that such risk is generally low.

ITEM 15: MINERAL RESERVE ESTIMATES

15.1 Mine Plan

The mining method selected for the FS is the underground room-and-pillar method, similar to the methods commonly used in potash, coal, and trona underground mines. No pillar extraction or retreat mining is proposed. All mining is in the polyhalite ore bed. This mining method is well proven and fits the following parameters:

- Geologic type Low
- Deposit type Underground mining
- Seam dip <0.5 to 2° (flat)
- Minimum mining height 62 inches (equipment constrained)
- Product (plant feed) ROM polyhalite ore (–8 inches)

Mine projections were developed based on deposit parameters, geotechnical analysis, and equipment constraints.

Mine production and ore grade determinations were modeled using Carlson Mining 2013 Software© (2013) (Carlson) Underground Mining Module, historically referred to as SurvCADD™. This is the predominant mine planning software used by US underground mine operators for near flat-lying, bedded seam deposits, including coal, potash, and trona. It is well suited for the Ochoa polyhalite deposit.

Mine projections were developed in AutoCAD 2013™ (Autodesk, Inc. 2013) based on the Ochoa resource grids discussed in Section 14.2.1. The resource grids describe true bed thickness, elevation, depth of cover, dip, and the following quality parameters: polyhalite, anhydrite, halite, magnesite, and K_2SO_4 (equivalent). Projections also accounted for oil, gas, and disposal wells located within the mine boundary. For the FS, mining recovery was limited to 60% extraction ratio in the panels; however, geotechnical modeling suggests higher extraction ratios are possible.

The mine layout was developed for the majority of the Property; however, the mine projections for the Reserve determination were limited to the 50-year FS timeline and to M&I Resources in accordance with the definition of Mineral Reserves per CIMDS. Mining was constrained by property boundaries, ore bed thickness, polyhalite ore grade contour of 66%,[6] and 200-ft-radius barrier pillars around drilled gas, oil, and disposal wells. Permitted but undrilled well sites were ignored, as the timing and duration of any such well development is undeterminable and will be handled on a case-by-case basis during ongoing operations, similar to US coal mine practice.

Significant M&I Resources with reasonable expectations of economic extraction exist on the Property beyond the 50-Year Mine Plan boundary. This statement is based on the following:

- The geologic properties of the Property east, north, and west of the 50-Year Mine Plan boundary are similar to those in the 50-Year Mine Plan.

[6] The 66% ore grade contour was used as a target for drawing mine projections to ensure maintaining cutoff grade.

- Sufficient M&I Resources are available within the Property and outside the 50-Year Mine Plan boundary that would permit the extension of the 50-Year Mine Plan by an estimated 25 plus years. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
- Mining methods could be the same as the 50-Year Mine Plan, with expected similar economic results.

Figure 15-1 illustrates the mine projections could be applied to the majority of the Property in support of the above statement.

Mine timing (unit scheduling) was developed using Carlson based on bed volumetrics, production rates for each mining section (unit), location (mains, panel development, panel room retreat), and work schedules. Each continuous miner section was scheduled by year for 50 years of the FS mine plan. Results were summarized monthly and quarterly for the first few analysis years of mining and annually for the remainder of the 50-Year Mine Plan for the economic analysis. Figure 15-2 shows 50-Year Mine Plan boundary overlain on the M&I and Inferred Resource areas.

OSD was calculated in the geologic model based on ore bed thickness using a minimum mining height of 62 inches (equipment constraint). Dilution was calculated on a grid cell by grid cell basis, and the characteristics of the mine roof and floor were taken into consideration. Roof dilution was taken at 2 inches, and floor dilution was taken at 4 inches, for a total OSD of 6 inches, or 9.7% of the minimum mining height. The 6 inches of OSD is an addition to the minimum equipment mining height, or polyhalite thickness when it is greater than the minimum mining height. Because the floor is weaker than the roof, and on average also has higher polyhalite grade, when additional height must be cut to accommodate the mining equipment, it is cut from the floor.

Figure 15-3 illustrates the amount of OSD when the polyhalite ore bed is greater than the 62-inch minimum mining height. Figure 15-4 shows the amount of OSD when the ore bed thickness is less than the 62-inch minimum mining height. Figure 15-5 shows the amount of floor that needs to be cut to maintain the minimum mining height. The final head grade and mined tons were calculated as a mixture of ore and rock cut out-of-seam.

Mine tonnage, timing, and ore grade were determined, a CAPEX and OPEX budget was prepared, and a pre-tax and after-tax cash flow analysis was conducted to determine economics. A marketing study was performed by CRU for the PFS (Gustavson 2012) and revised by ICP for the FS, and environmental and permitting requirements were identified and studies have been completed or are underway. A Draft EIS has been published by the BLM and the public comment period has been closed. A Record of Decision is expected by spring 2014.

15.2 Polyhalite Reserves

Table 15-1 states the M&I tonnage converted to Proven and Probable Reserve tonnage based on the 50-Year Mine Plan. No Inferred tons were included in the Reserve estimate. **Resource totals stated in Table 14-4 are *inclusive* of the Reserves stated in Table 15-1.**

Various risks are associated with mining the Reserves which are independent of geologic confidence. Mineral Reserves could be adversely affected by mining conditions. Ore grade could be adversely affected by mining conditions and continuous miner operator differentiation of polyhalite vertical extents. Permitting delays would adversely impact the Project implementation schedule but should not impact Mineral Reserves. Legal challenges could reduce available Mineral Reserves. Reduced productivity would increase operating and capital costs adversely, affecting Project economics. Unfavorable court decisions on permit challenges could result in not receiving necessary permits.

The QP has reviewed the FS and is satisfied that the CIMDS modifying factors have been adequately addressed; therefore, all Measured tons within the FS' 50-Year Mine Plan are presently classified as Proven tons.

Table 15-1. Ochoa Project Mineral Reserves (effective date January 9, 2014)

	Average Mined Thickness[1] (ft)	50-Year Mine Plan Mined Area (million ft^2)	ROM Mined Tons[2,3] (millions)	Mining Recovery[4] (%)	Polyhalite (wt %)	Equivalent K_2SO_4 (wt %)	Anhydrite (wt %)	Halite (wt %)	Magnesite (wt %)
PROVEN	5.9	246	125.0	47.1%	78.42	22.66	11.29	3.66	7.79
PROBABLE	5.9	113	57.4	64.8%	77.20	22.31	11.60	3.65	8.30
TOTAL P&P	5.9	359	182.4	51.5%	78.05	22.55	11.39	3.66	8.08

[1] Bed thickness cutoff 4.0 ft, composite grade cutoff 66.0% polyhalite, includes out-of-seam dilution.

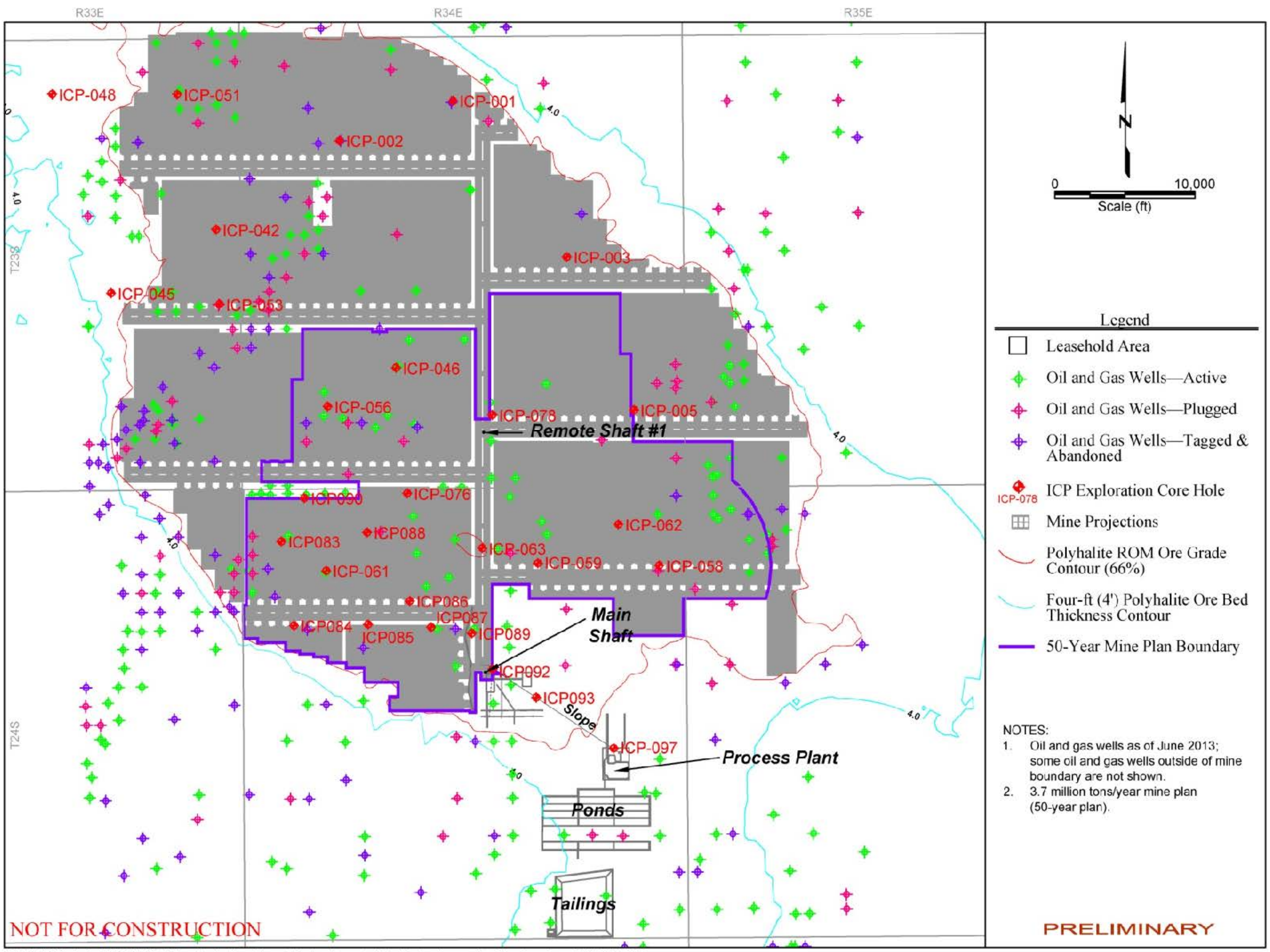
[2] Average in-situ bulk density of 173.5 pcf.

[3] No inferred tons mined

[4] Areal recovery (mined area) inside 50 Year Mine Plan boundary

Note: Gypsum weight percent negligible for all resource classifications.

Mineral Reserves are included in Mineral Resources



800-01 IC Potash [800-01 ICP_Mine Projection Maps_TR.dwg; Layout: Mine Projections w Surf Features]:smvf (12-20-2013)

Figure 15-1. Life-of-Mine Projections, Property Boundaries, and Surface Facilities

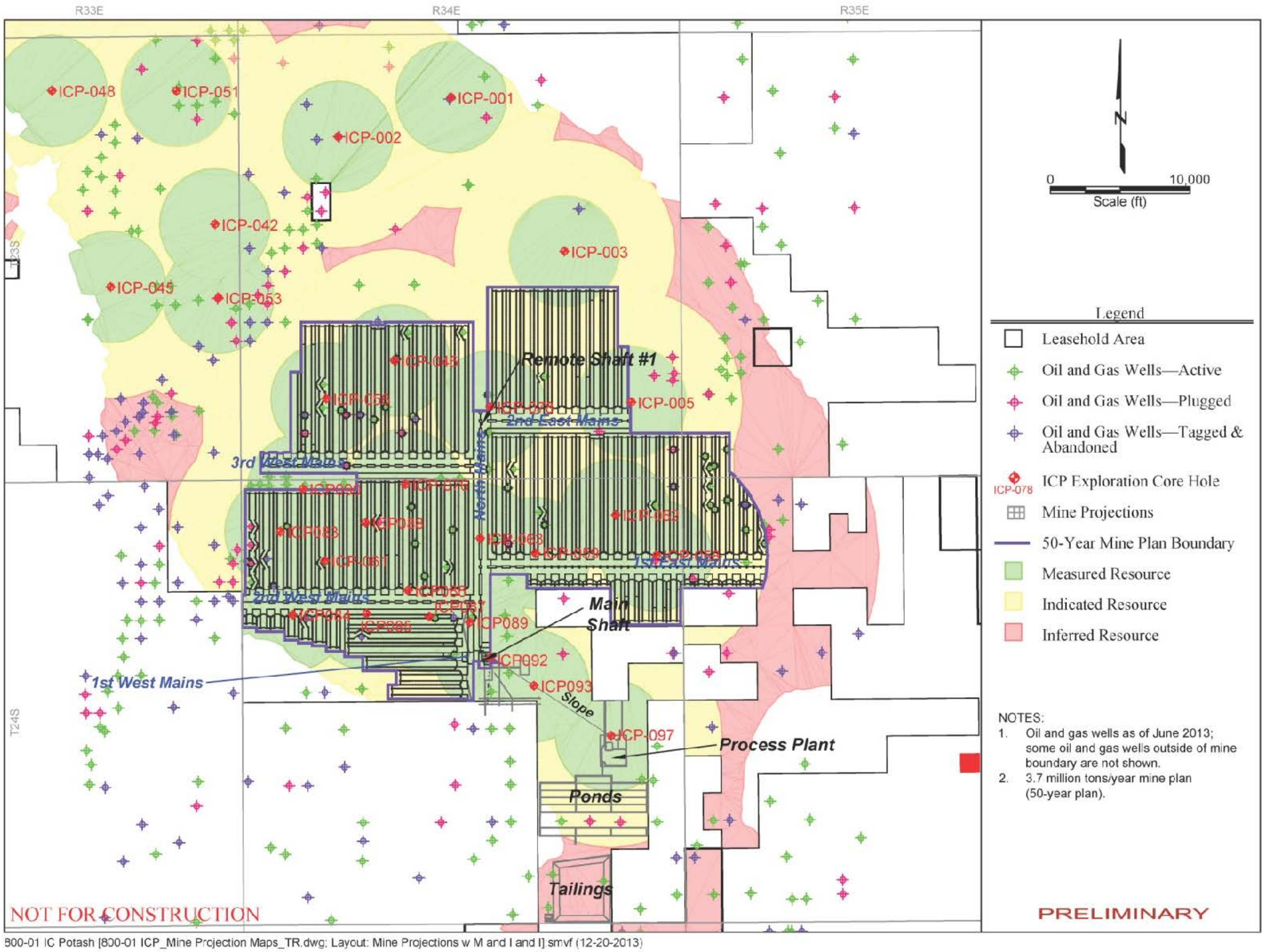


Figure 15-2. 50-Year Mine Projections with Measured, Indicated, and Inferred Resource Areas

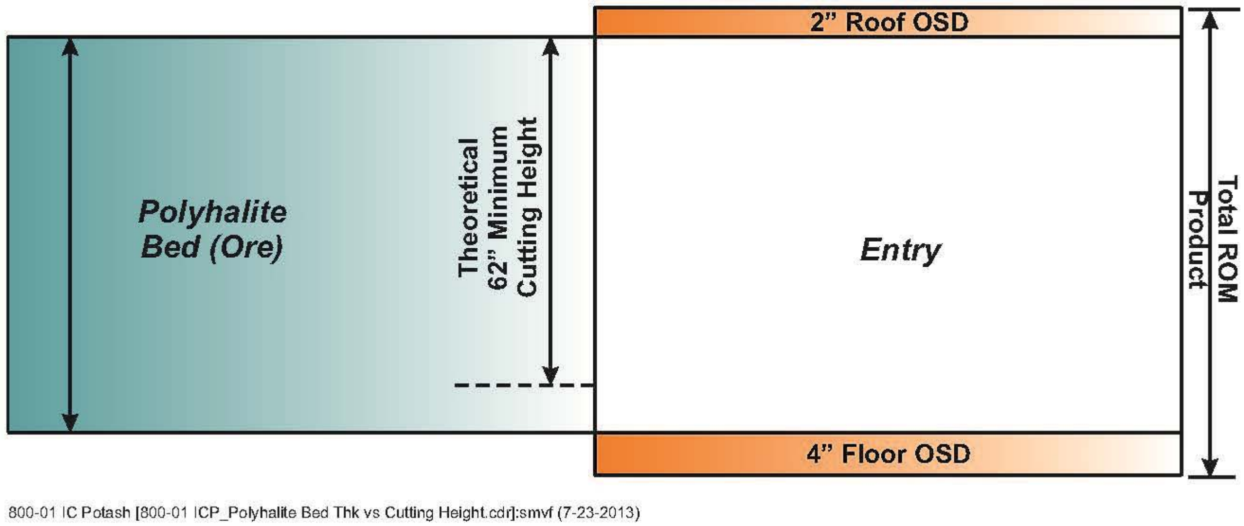


Figure 15-3. OSD When Ore Bed Height is Greater Than the 62-inch Minimum Mining Height

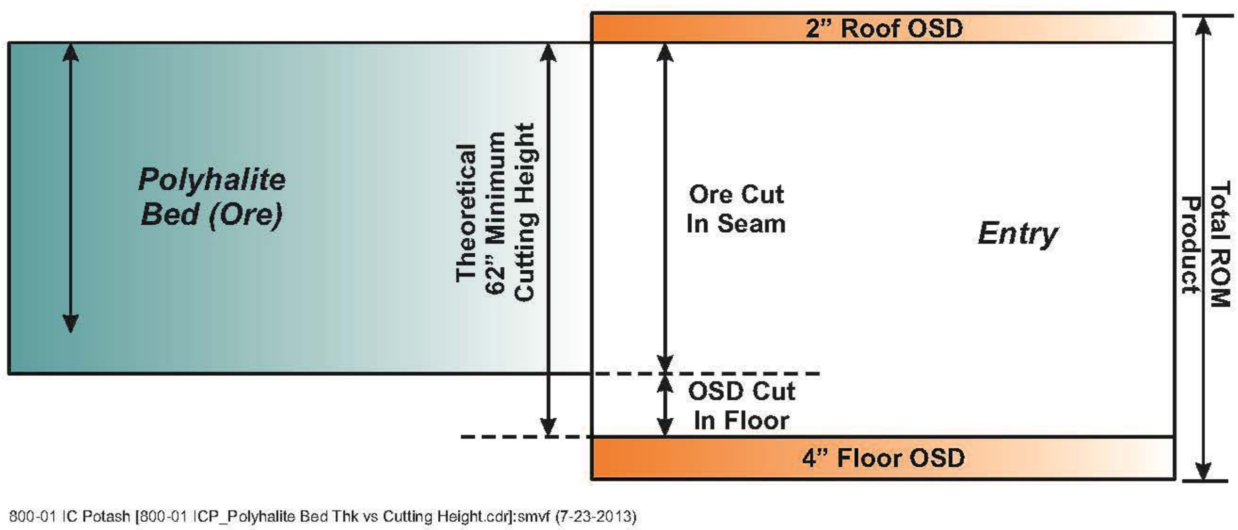
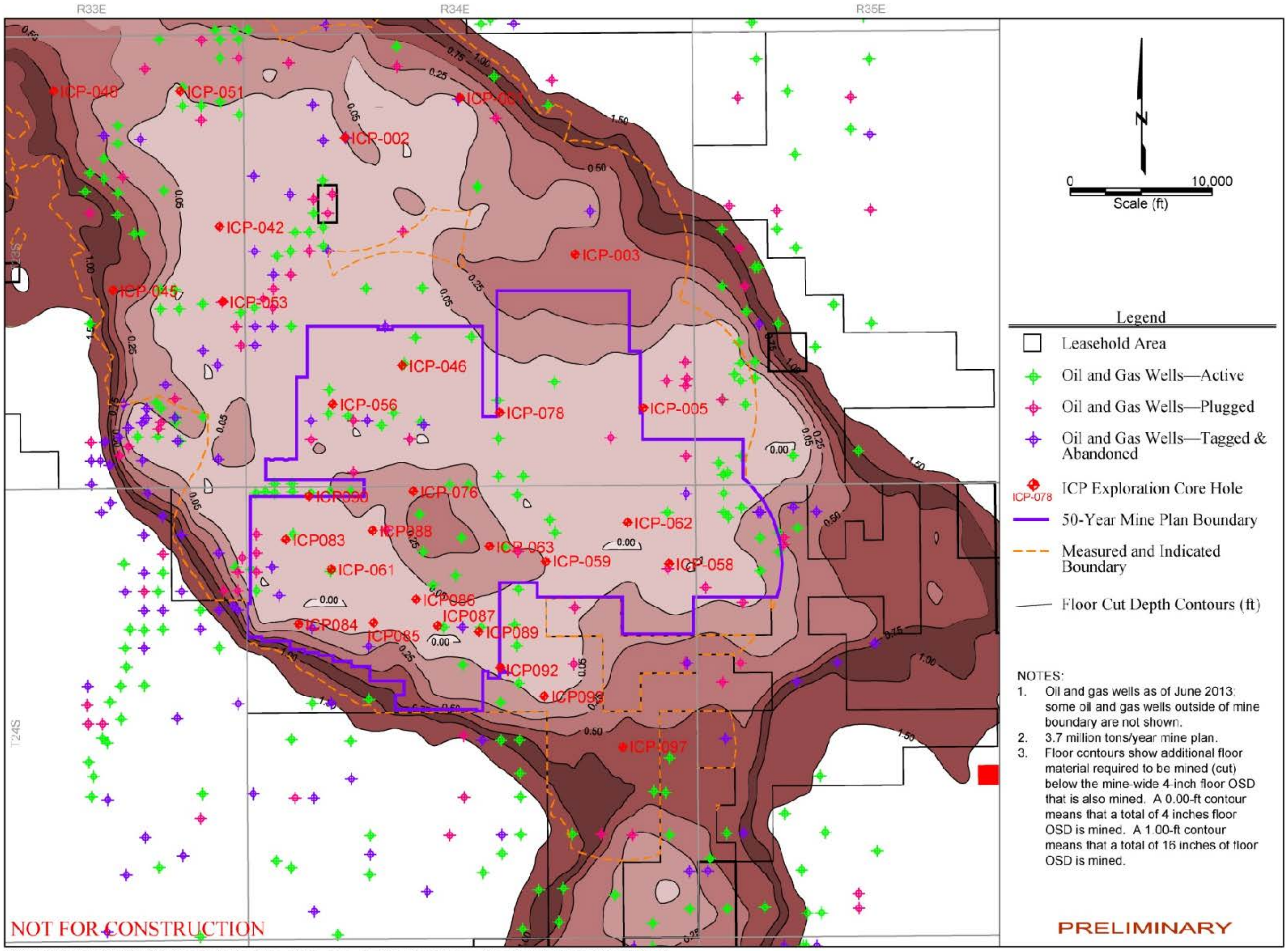


Figure 15-4. OSD When Ore Bed Height is Less Than the 62-inch Minimum Mining Height



800-01 IC Potash [800-01 ICP_Mine Projection Maps_TR.dwg; Layout: Mine Projections w Floor Cut Depth]:smvf (12-20-2013)

Figure 15-5. Floor Cut Depth for a 62-inch Minimum Mining Height

ITEM 16: MINING METHODS

16.1 General

The FS (SNCL 2014) is based on the underground room-and-pillar mining method based on an average of 60% extraction in the production panels. No secondary (pillaring) recovery is planned at this time. As discussed in Item 15, this mining method was selected because the polyhalite ore bed is a tabular, strata-bound deposit suitable for mining by heavy-duty type coal and potash mining equipment used for medium bed thicknesses.

ICP plans to construct and operate, for 50 years, an underground polyhalite mine designed to provide a nominal 3.7 Mtpy of ROM polyhalite ore to a processing plant located nearby on the surface. The ROM ore grade will average 78.05% polyhalite over the 50-Year Mine Plan area. The deposit ranges from 1,300 ft to 1,635 ft below the surface in the 50-Year Mine Plan area.

All mining takes place in the ore bed, with additional height taken in main and submain entries (mains/submains) for ventilation and long-term convergence allowances. Long life mains and submains are protected with barrier pillars, as are abandoned and existing gas, oil, and disposal wells. Higher extraction ratios may be feasible contingent on results from additional geotechnical modeling.

The polyhalite ore bed will be accessed via a 25-ft-diameter, two-compartment mine ventilation and service shaft, and an 8.5° slope approximately 2 miles long. The 1,525-ft-deep shaft will have an intake air compartment equipped with an escape hoist system and electrical and communication cables. The second compartment will be used for return air and two 11-ft-diameter exhausting mine fans located on the surface will be connected to the shaft. The return air compartment will also house mine freshwater and mine drainage water pipelines. The slope will contain a 60-inch-wide belt conveyor for ore and waste (gob) haulage, a 12-ft-wide vehicle roadway for mining equipment, personnel travel, and supply transportation.

The shaft and slope bottom area will contain a mine equipment repair shop, warehouse, shift foremen, shop and warehouse supervisors' offices, parking areas for crew and supervisor vehicles, diesel fueling station, and an electrical switchgear room for the mine's high-voltage electrical distribution system's circuit breakers and disconnect switches.

The characteristics of the ore body and its location in an active gas and oil producing region create a number of mine design and operational concerns that may limit productivity and impose other constraints. Various mitigation measures may, to some degree, offset the concerns listed below. These measures are incorporated into the FS mine plan and economics to the extent the data allow.

The primary concerns addressed in the FS are:

- Polyhalite ore is strong, brittle (microcrystalline structure), and non-viscoelastic (no creep).
- The strong microcrystalline structure and high unconfined compressive strengths increase cuttability difficulty, requiring high horsepower, heavy-duty continuous miners.

- The immediate anhydrite roof features a thin mudstone parting that exhibits little to no adhesion to the roof above the parting horizon. The parting ranges from about 3 inches to around 18 inches above the top of the polyhalite ore bed.
- The mine floor is weaker than the polyhalite, which may result in additional OSD with the heavy continuous miners.
- There are over 750 gas and oil wells penetrating the ore bed on the Property or near it in the region.
- The mine slope must penetrate, at a shallow angle, through approximately 1,300 ft of weak, fractured overburden.
- Ore grade control may be difficult at times as bed extents are hard to determine visually.
- Mains and submains and respective barrier pillars must be designed to minimize convergence over an extended time.
- Groundwater inflows during shaft and slope construction must be controlled.
- Methane has not been detected in or near the ore zone, but the mine must be designed as a gassy mine due to the presence of the gas and oil wells.
- Adequate ventilation is required for potential methane migration from corrupt well bores, diesel particulate matter (DPM), and respirable dust or K40 radon daughters, if present.

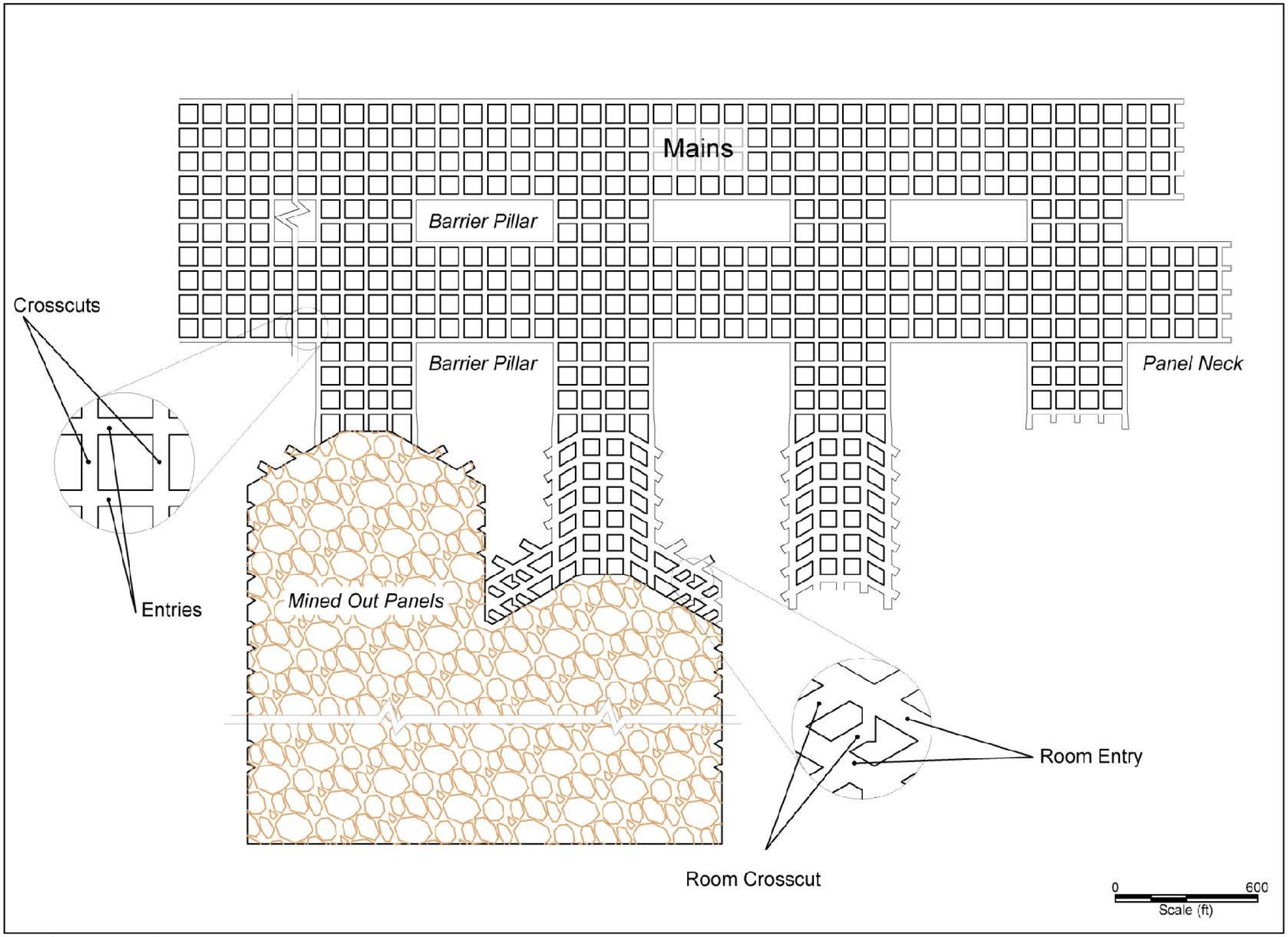
Polyhalite is incombustible and polyhalite dust is considered non-explosive.

16.2 Mine Design and Development

Construction of the mine slope is on the mine development critical path. Shaft and slope construction can begin nearly simultaneously as soon as the necessary permits are received and the shaft and slope site preparation are completed. Ore bed development will commence from the slope bottom after the permanent slope belt conveyor is installed. Mining will proceed to the shaft to establish permanent ventilation and provide for a secondary means of escape. Once permanent ventilation and the secondary excapeway is established, additional continuous miner units can be commissioned as space becomes available.

The mine is designed as a room-and-pillar mine, with all extraction done on a first mining basis. For the FS base case, no pillar retreat (secondary mining, pillar extraction, depillaring) is anticipated. Mine projections are divided into mains, submains, and production panels (Figure 16-1). Production panels are further subdivided into advance entries and crosscuts, and retreat rooms and room crosscuts. Mains (main entries) are developed five to seven entries wide, and production panels are developed five entries wide to accommodate the dual-split super section (DSSS) arrangement of mining equipment and double-split ventilation concept of simultaneously operating two continuous mining sections side-by-side within a set of entries, using one belt conveyor.

The mine will feature heavy-duty drum-type continuous miners, shuttle cars and articulated haulers, feeder-breakers, and dual-boom roof bolters for the production equipment. At full production of 3.7-Mtpy ROM ore, seven continuous miners deployed in three DSSS and one single section will be required. Underground ore haulage will be with 42-inch belt conveyors in the production panels, and 60-inch-wide belt conveyors in the mains and slope. Underground power will be provided by a 12,470-volt (V) distribution system, with appropriate



800-01 IC Potash [800-1-T1-SB05 Typical Layout_TR.dwg].smvf (1-16-2014)

Figure 16-1. Typical Mains and Panel Layout with Nomenclature

step-down transformers located throughout the mine for mobile and stationary electrically powered equipment. Transportation of personnel and supplies will be by self-propelled diesel or battery powered equipment.

The shaft and slope portal locations were set during the preliminary project planning stage (in the PFS stage) and were based primarily on land tenure and regulatory agency considerations and constraints. The shaft and slope bottom area is located on the south side of the main ore body, approximately halfway across the east/west dimension of the mineable ore boundaries. Primary development into the ore body proper is via a double set of north entries (North Mains), with double sets of East and West Mains mined perpendicular to the North Mains at periodic intervals, to form discrete mining districts.

The shaft and slope bottom area is designed to facilitate the rapid addition of continuous mining sections, making for a quick ramp-up schedule for ore production without compromising the long-term functionality of the shaft and slope bottom area infrastructure.

Figures 16-2 through 16-6 show the polyhalite ore bed thickness, the ROM ore grade contours, the base of ore bed structure (elevations), and the mining thickness contours, respectively.

16.2.1 Geotechnical Mine Design

Unlike most other potash deposits, the Ochoa polyhalite bed consists of a strong, hard, brittle, microcrystalline material that does not exhibit viscoelastic properties. The hard, brittle bed is sandwiched between layers of plastic/elastic salts that exhibit typical salt creep rates. Halite and anhydrite salts extend from tens to a couple of hundred feet above the polyhalite ore bed, and tens of feet below the polyhalite bed. These salts are at times interbedded with other thin strata (e.g., dolomite, mudstones). As a result of the different physical characteristics of polyhalite versus other salts, standard Carlsbad potash practices regarding polyhalite pillar behavior and mining recoveries cannot be assumed to hold true for the Ochoa Mine design.

Therefore, a comprehensive geotechnical mine design evaluation was conducted for estimating pillar sizes, entry widths, ground support practices, and anhydrite roof standup time. In addition, shaft lining design and surface subsidence were also evaluated. Previous Phase 1, 2, and 2B years' core drilling and laboratory testing; 2012 and 2013 Phase 3A core drilling; core photographs; geologist's logs; and physical properties testing of selected Phase 3A drilling program cores were the basis for the geotechnical mine design. Selected core samples from the Phase 3A drilling program were tested by AAI at its Grand Junction, Colorado, rock mechanics laboratory.

As part of Phase 3A, ICP drilled three core holes for the purposes of geotechnical testing near the shaft location and along the slope orientation, for mining and for shaft and slope design. All three drill holes were continuously cored, with a 3-inch core diameter, from surface through the points of interest, to collect samples for geotechnical testing. Core logging conducted by AAI entailed describing and classifying the unconsolidated (soil) and consolidated (rock) materials retrieved from the drill hole as coring advanced.

Lithologic logging provided rock descriptions through classification of type, color, texture, hardness, and accessory minerals. The materials were classified using visual, manual, and field analytical methods of examination and were recorded manually on graphic log forms. Geotechnical data were logged on the same form, characterizing all structural features through

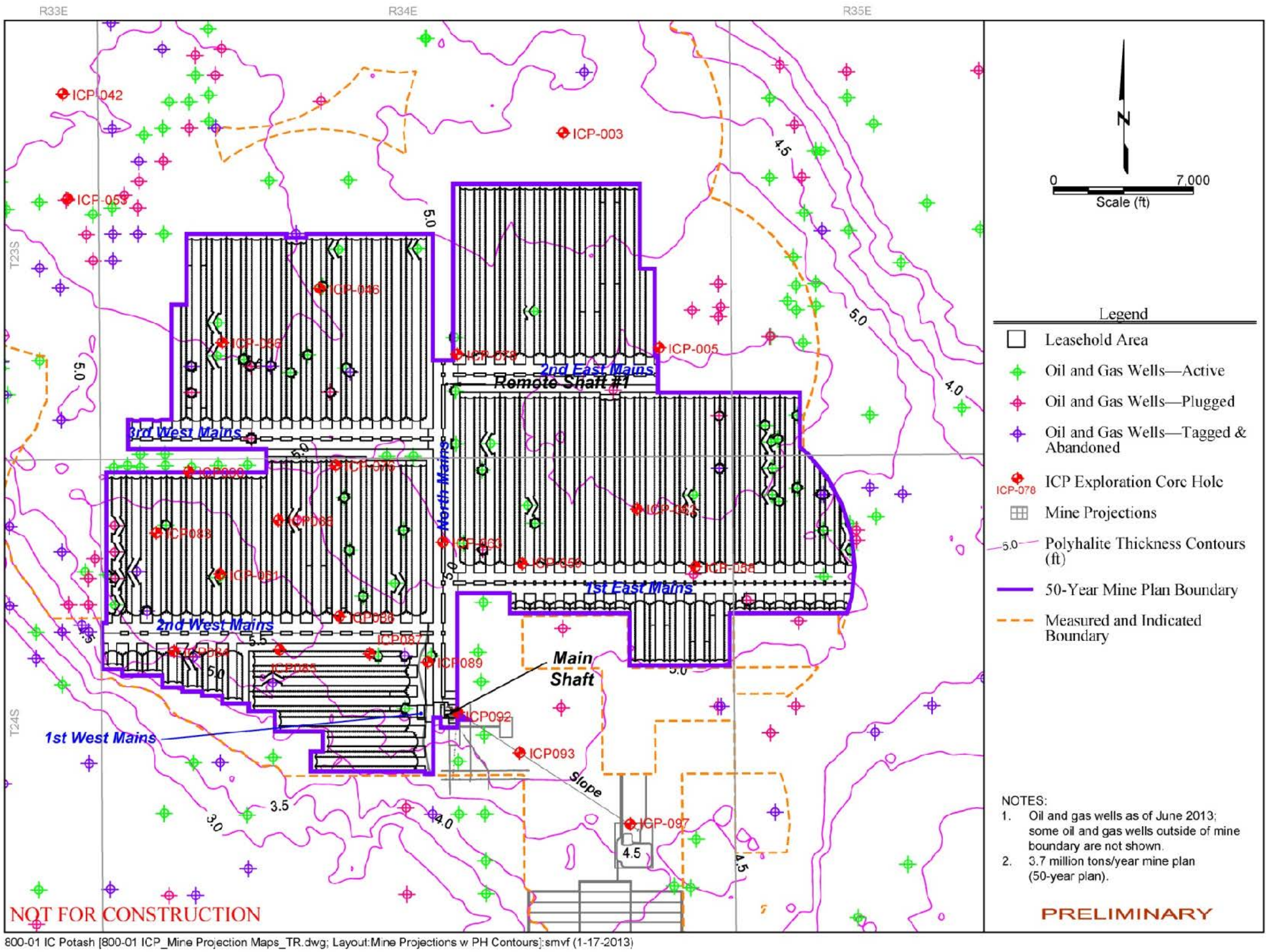


Figure 16-2. 50-Year Mine Projections, Polyhalite Thickness Contours

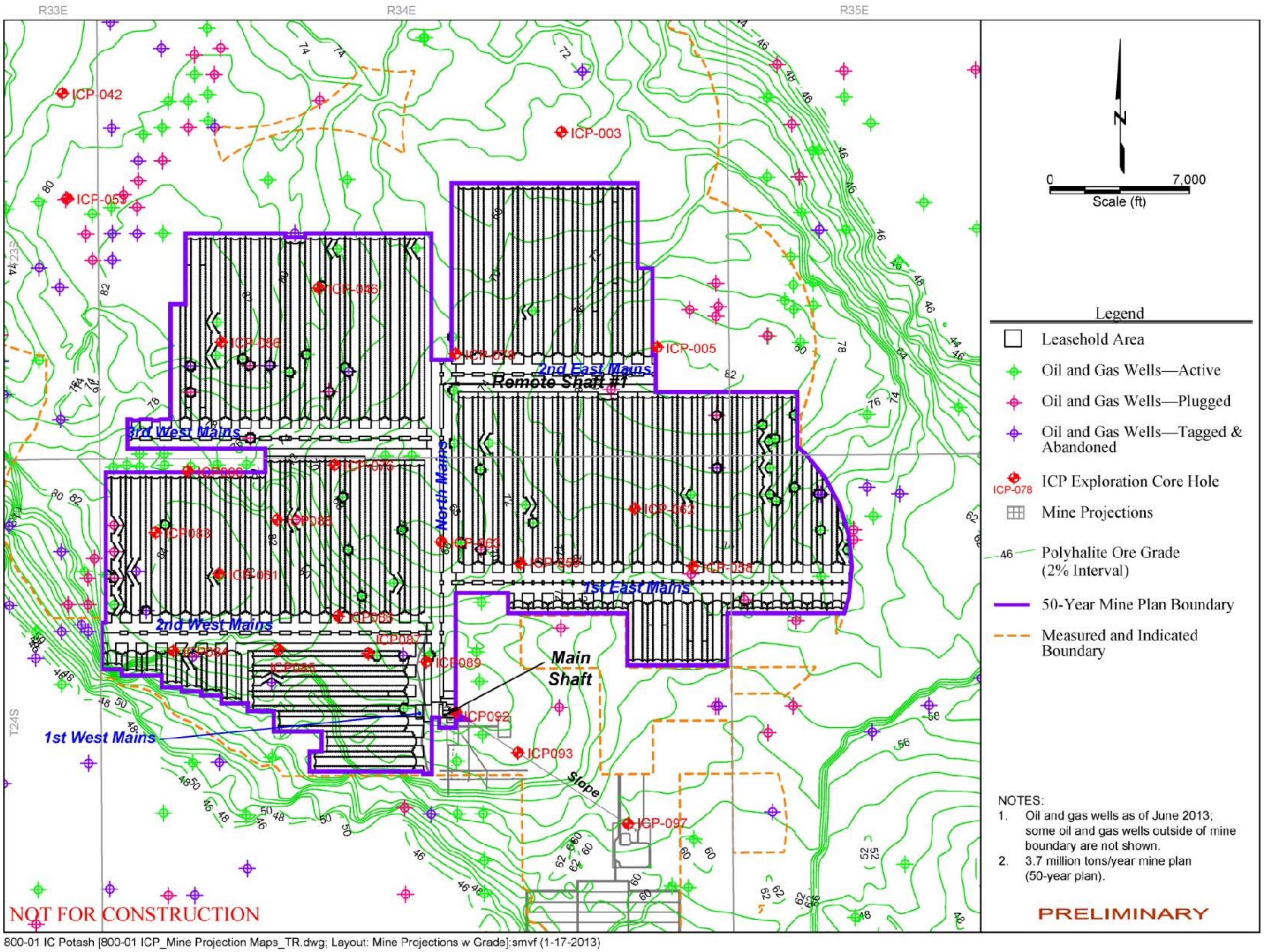


Figure 16-3. 50-Year Mine Projections, Polyhalite Ore Grade

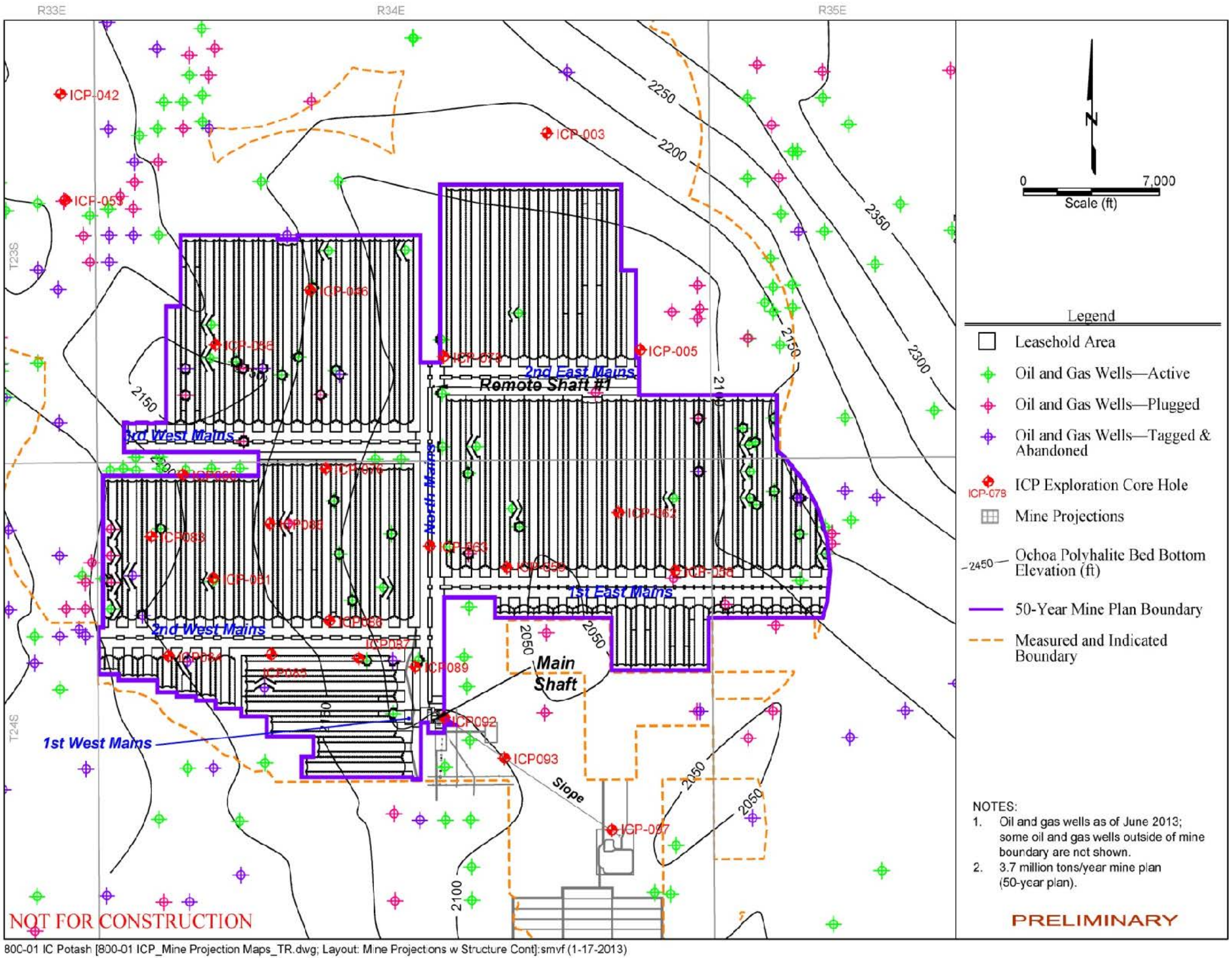


Figure 16-4. 50-Year Mine Projections, Base of Ore Structure Contours

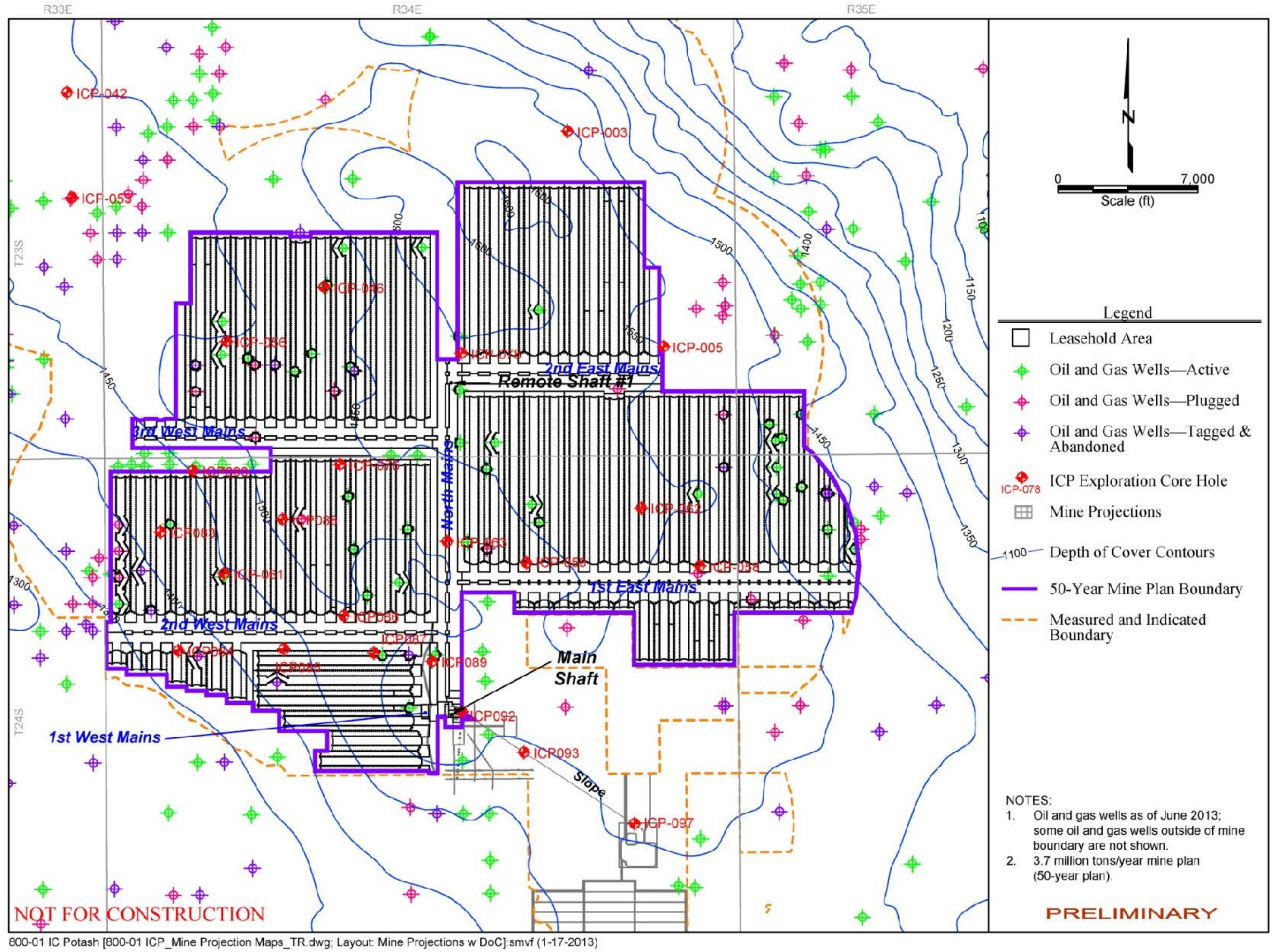


Figure 16-5. 50-Year Mine Projections, Depth of Cover

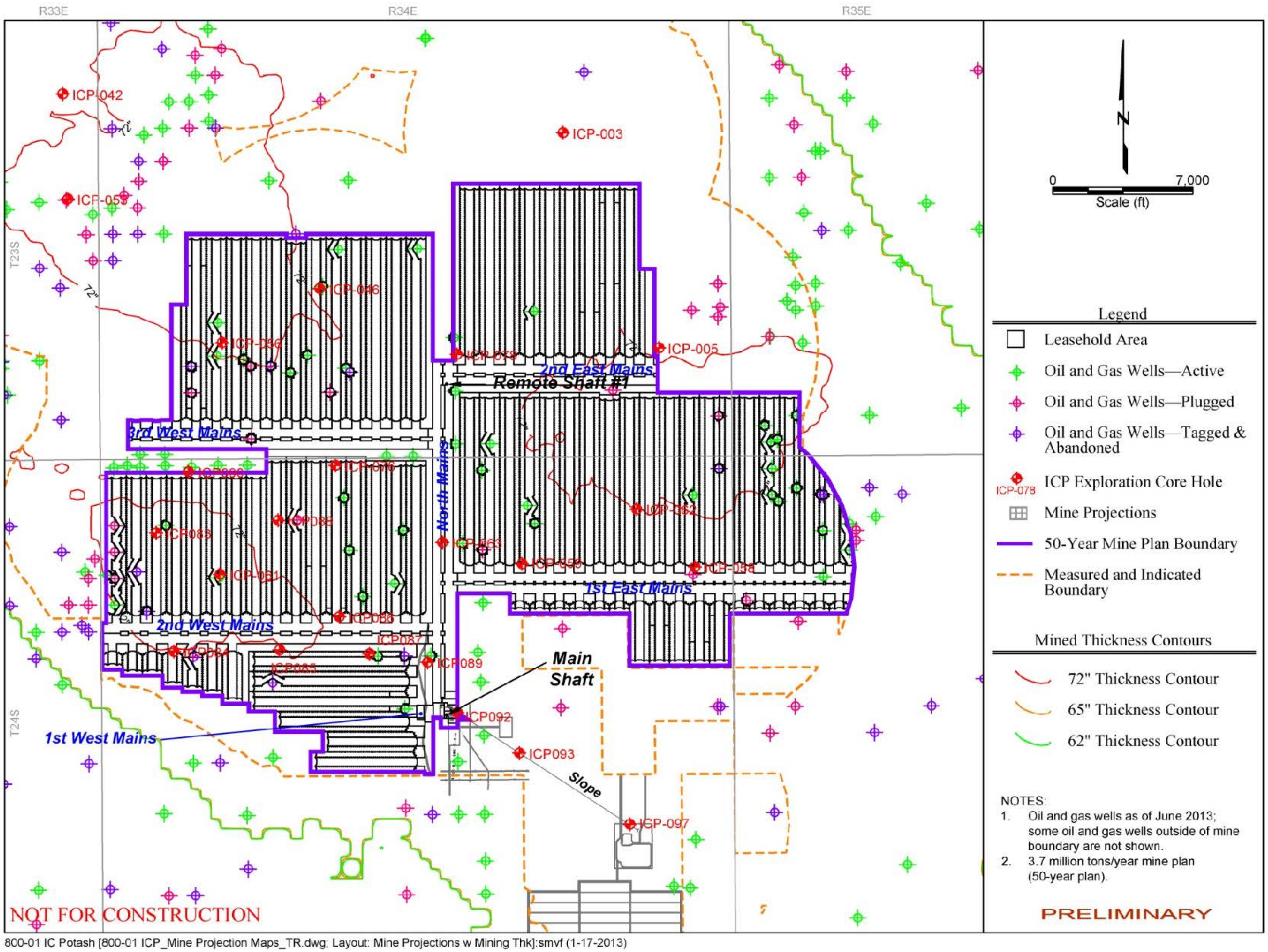


Figure 16-6. 50-Year Mine Projections, Mined Thickness

type and class, infill compositions, measured angles, planarity, roughness, and depth. Rock quality designation (RQD) was also calculated for each core run along with core recovery percentages.

Core Testing—A four-part mine geotechnical testing program was developed for the Project. Each part was assigned a priority number. Priorities 1 and 2 were the basis for the geotechnical testing for the FS. Priorities 3 and 4 can be undertaken during detailed mine design and initial mining operations. Table 16-1 outlines the four mine geotechnical priorities for the Project.

Samples for geotechnical testing were selected based on a predetermined sample selection plan which targeted various lithologic intervals in support of mine feasibility design and shaft and decline slope design efforts. Additionally, samples were selected within the ore zone to test spatial and vertical variations in ore strength throughout the Property to assist with the quantification of the ores' mineability characteristics. Samples for direct shear testing were selected by AAI and shipped to Advanced Terra Testing, Inc., in Lakewood, Colorado, where the tests were performed.

Table 16-2 summarizes all geotechnical laboratory tests performed. Priority 1 testing focused on information relevant to mine feasibility, and Priority 2 targeted data relevant to shaft and slope design. All testing was performed between February and April 2013.

Priority 1 results for uniaxial compressive strength (UCS) tests with elastic properties (UCS-E) are presented in Tables 16-3 to 16-6. Polyhalite UCS ranged from a low of 6,208 psi (interbedded polyhalite and floor material) to 22,838 psi. While the latter was located outside of the mine plan boundary, it is nonetheless indicative of the range of strengths. The lowest UCS test result in the defined ore bed zone is 8,525 psi, with a majority of the values being in the 9,000 to 12,000 psi range. The highest UCS value tested that was within the 50-Year Mine Plan area was from drill hole ICP-083. This result was from a core sample taken from the linear cutting test core and is not listed in Tables 16-3 to 16-6.

Tables 16-7 to 16-9 show the triaxial compressive strength (TCS) results for the roof, ore zone, and floor, respectively.

Splitting tensile strength tests (Brazilian) were carried out throughout the mining horizon for the Priority 1 testing program where the structural integrity of the rock core did not allow sufficient sample length to perform a UCS test. The Brazilian test results are presented in Table 16-10.

Slake durability testing was performed for the Priority 1 and 2 testing, both over the mining horizon and in varying lithologies at the proposed shaft location. The results are presented in Table 16-11.

The Priority 2 testing plan focused on UCS-E testing to support shaft and decline design. Table 16-12 summarizes the testing results.

Geotechnical Modeling—For mine geotechnical modeling, the polyhalite ore zone is taken as approximately 1,500 ft below surface and averages 5 ft thick. Typically, the ore zone is immediately underlain by a few feet of brecciated mudstone, with halite and anhydrite layers deeper into the main floor. The halite and anhydrite layers are on the order of tens of feet thick. The polyhalite is directly overlain by a thin anhydrite layer, ranging in thickness from 0 to 2.5 ft,

Table 16-1. Ochoa Project Recommended Geotechnical Properties Testing

	Main Roof (>10 ft)	Immediate Roof (0-10 ft)	Mining Horizon	Immediate Floor (0-10 ft)	Main Floor (>10 ft)	Total Tests per Core Hole	No. of Core Holes	Total No. of Tests
	Halite	Halite/ Anhydrite	Polyhalite	Halite/ Anhydrite	Halite			
Priority 1 Testing—Mine Feasibility Design								
Conduct testing to statistically characterize mining horizon and immediate host strata. Test in 3 holes spatially representative of property.								
UCS w/ Elastic Properties	-	3	4	3	-	10	3	30
Triaxial w/ Elastic Properties	-	4	4	4	-	12	3	36
Brazilian Tensile	-	4	2	4	-	10	3	30
Slake Durability†	-	1	1	1	-	3	3	9
Specific Gravity	-	7	8	7	-	22	3	66
Direct Shear‡	-	1	-	1	-	2	3	6
Total						59		177
† Slake durability tests of anhydrite and polyhalite.								
‡ Direct shear tests of lithologic contacts: halite-anhydrite, anhydrite-polyhalite.								
Priority 2 Testing—Shaft and Decline Design								
Conduct testing to characterize overburden for shaft and decline design. Test in (a) shaft pilot hole and (b) representative delince hole(s). Testing also supports analysis of subsidence and petroleum well impacts.								
UCS w/ Elastic Properties	30	-	-	-	-	30	2	60
Specific Gravity	30	-	-	-	-	30	2	60
Direct Shear†	5	-	-	-	-	5	2	10
Total						65		130
† Direct shear tests of representative lithologic contacts.								
Priority 3 Testing—Creep Testing for Final Mine Design								
Conduct testing to characterize time-dependent deformation at mining horizon and in immediate host strata. Test in 2 holes spatially representative of property.								
Creep†	-	2	2	2	-	6	2	12
Total						6		12
† Minimum recommended creep test duration 3 months; 6-month test preferred.								
Priority 4 Testing—Mine Final Design								
Expand geotechnical database in advance of final design. Test in 3 infill holes to increase data density over property.								
UCS w/ Elastic Properties	-	3	4	3	-	10	3	30
Triaxial w/ Elastic Properties	-	4	4	4	-	12	3	36
Brazilian Tensile	-	4	2	4	-	10	3	30
Slake Durability	-	1	1	1	-	3	3	9
Specific Gravity	-	7	8	7	-	22	3	66
Direct Shear	-	1	-	1	-	2	3	6
Total						59		177
Total for Priority 1–4 Testing	65	42	40	42	-	189		496

Table 16-2. Geotechnical Test Matrix

Test Type	Priority 1	Priority 2
Uniaxial Compressive Strength with Elastic Properties	54	51
Triaxial Compressive Strength with Elastic Properties	30	
Indirect Tensile Strength (Brazillian) tests	34	
Slake Durability	9	9
Specific Gravity	83	51
Direct Shear*	3	6

*Direct shear tests performed by Advanced Terra Testing.

Table 16-3. ICP-093 Mining Horizon UCS-E Results

Depth (ft)		Lithology	UCS	Young's Modulus	Density
To	From		(psi)	(Mpsi)	(pcf)
1,489.40	1,490.00	Roof—Halite/anhydrite	4,372	0.67	137.0
1,491.00	1,491.52	Roof—Halite/anhydrite	4,663	2.08	147.1
1,491.65	1,492.05	Roof—Halite/anhydrite	2,958	1.52	171.4
1,493.02	1,493.60	Ore—Polyhalite	8,685	3.63	170.6
1,493.64	1,494.04	Ore—Polyhalite	8,525	3.74	171.8
1,494.10	1,494.64	Ore—Polyhalite	10,760	4.73	171.0
1,494.83	1,495.18	Ore—Polyhalite	9,217	3.63	169.2
1,495.20	1,495.74	Ore—Polyhalite	11,182	3.55	169.6
1,496.04	1,496.40	Ore—Polyhalite	8,708	3.75	170.6
1,496.49	1,497.00	Ore—Polyhalite	9,591	3.68	168.0
1,497.00	1,497.40	Ore—Polyhalite	9,139	4.20	170.7
1,497.47	1,497.83	Ore—Polyhalite	6,208	3.95	168.9
1,497.98	1,498.51	Floor—Halite/anhydrite	5,827	2.70	174.6
1,499.20	1,499.76	Floor—Halite/anhydrite	987	0.12	139.3

Table 16-4. ICP-083 Mining Horizon UCS-E Results

Depth (ft)		Lithology	UCS	Young's Modulus	Density
To	From		(psi)	(Mpsi)	(pcf)
1,404.95	1,405.45	Roof—Halite/anhydrite	4,552	0.62	139.9
1,406.35	1,406.70	Roof—Halite/anhydrite	3,012	1.13	165.9
1,411.15	1,411.60	Ore—Polyhalite	9,642	4.33	169.5
1,411.60	1,412.05	Ore—Polyhalite	10,741	4.60	171.3
1,412.05	1,412.50	Ore—Polyhalite	9,382	3.54	166.7
1,412.05	1,412.50	Ore—Polyhalite	8,785	3.97	165.6
1,412.80	1,413.30	Floor—Halite/anhydrite	5,035	2.35	173.5
1,413.30	1,413.80	Floor—Halite/anhydrite	6,115	2.94	176.0
1,413.80	1,414.30	Floor—Halite/anhydrite	4,002	1.80	165.3
1,420.65	1,421.20	Floor—Halite/anhydrite	1,472	0.74	140.2
1,421.75	1,422.30	Floor—Halite/anhydrite	2,100	0.87	139.4
1,421.20	1,421.75	Floor—Halite/anhydrite	1,942	0.45	137.7

Table 16-5. ICP-089 Mining Horizon UCS-E Results

Depth (ft)		Lithology	UCS	Young's Modulus	Density
To	From		(psi)	(Mpsi)	(pcf)
1,507.10	1,507.65	Roof—Halite/anhydrite	2,548	0.92	134.80
1,516.30	1,516.75	Ore—Polyhalite	12,774	4.63	173.60
1,516.75	1,517.20	Ore—Polyhalite	11,302	4.93	173.40
1,516.75	1,517.20	Ore—Polyhalite	12,614	5.16	173.80
1,529.80	1,530.40	Floor—Halite/anhydrite	1,910	0.36	135.30
1,530.40	1,531.00	Floor—Halite/anhydrite	2,041	0.41	136.10
1,531.00	1,531.60	Floor—Halite/anhydrite	2,593	0.65	134.50

Table 16-6. ICP-097 Mining Horizon UCS-E Results

Depth (ft)		Lithology	UCS	Young's Modulus	Density
To	From		(psi)	(Mpsi)	(pcf)
1,496.15	1,496.75	Roof—Halite/anhydrite	3,966	0.39	135.1
1,497.35	1,497.80	Roof—Halite/anhydrite	4,376	0.76	137.1
1,507.65	1,508.20	Ore—Polyhalite	20,154	6.44	174.4
1,507.65	1,508.20	Ore—Polyhalite	21,858	6.70	174.3
1,508.20	1,508.75	Ore—Polyhalite	18,166	7.70	173.7

Table 16-7. ICP Triaxial Roof Zone Results

Depth (ft)		Exploration Hole No.	Lithology	Confining Pressure	Triaxial Strength	Young's Modulus	Density
To	From			(psi)	(psi)	(Mpsi)	(pcf)
1,406.35	1,406.70	ICP-083	Roof—Anhydrite	1,700	5,326	0.72	165.6
1,405.45	1,405.95	ICP-083	Roof—Halite/anhydrite	200	7,673	0.45	138.9
1,405.45	1,405.95	ICP-083	Roof—Halite/anhydrite	400	8,067	0.24	138.6
1,405.45	1,405.95	ICP-083	Roof—Halite/anhydrite	800	5,213	0.76	138.8
1,507.10	1,507.65	ICP-089	Roof—Halite/anhydrite	1,200	6,045	0.31	134.5
1,496.15	1,496.75	ICP-097	Roof—Halite/anhydrite	1,700	5,185	0.33	134.6
1,497.35	1,497.80	ICP-097	Roof—Halite/anhydrite	1,200	5,287	0.42	135.6
1,497.35	1,497.80	ICP-097	Roof—Halite/anhydrite	1,700	5,693	0.77	136.2

Table 16-8. ICP Triaxial Floor Zone Results

Depth (ft)		Exploration Hole No.	Lithology	Confining Pressure (psi)	Triaxial Strength (psi)	Young's Modulus (Mpsi)	Density (pcf)
To	From						
1,412.80	1,413.30	ICP-083	Floor—Halite/anhydrite	200	6,615	2.19	174.5
1,412.80	1,413.30	ICP-083	Floor—Halite/anhydrite	400	7,523	2.28	173.8
1,413.30	1,413.80	ICP-083	Floor—Halite/anhydrite	800	4,479	0.38	173.3
1,420.65	1,421.20	ICP-083	Floor—Halite/anhydrite	1,200	4,162	0.39	141.6
1,420.65	1,421.20	ICP-083	Floor—Halite/anhydrite	1,700	4,543	0.57	141.3
1,421.20	1,421.75	ICP-083	Floor—Halite/anhydrite	200	1,980	0.29	137.6
1,421.20	1,421.75	ICP-083	Floor—Halite/anhydrite	400	1,665	0.27	135.7
1,421.75	1,422.30	ICP-083	Floor—Halite/anhydrite	800	3,043	0.40	137.2
1,529.80	1,530.40	ICP-089	Floor—Halite/anhydrite	1,200	3,744	0.33	133.8
1,529.80	1,530.40	ICP-089	Floor—Halite/anhydrite	1,700	4,349	0.27	135.6
1,531.00	1,531.60	ICP-089	Floor—Halite/anhydrite	1,200	5,248	0.51	134.1
1,531.00	1,531.60	ICP-089	Floor—Halite/anhydrite	1,700	6,431	0.49	134.6

Table 16-9. ICP Triaxial Ore Zone Results

Depth (ft)		Exploration Hole No.	Lithology	Confining Pressure (psi)	Triaxial Strength (psi)	Young's Modulus (Mpsi)	Density (pcf)
To	From						
1,411.15	1,411.60	ICP-083	Ore—Polyhalite	200	10,678	3.56	172.2
1,411.15	1,411.60	ICP-083	Ore—Polyhalite	400	14,459	3.45	172.6
1,411.60	1,412.05	ICP-083	Ore—Polyhalite	1,200	16,499	3.46	171.4
1,412.05	1,412.50	ICP-083	Ore—Polyhalite	800	12,057	2.84	167.5
1,516.30	1,516.75	ICP-089	Ore—Polyhalite	1,700	22,838	3.25	169.7
1,516.30	1,516.75	ICP-089	Ore—Polyhalite	200	14,087	3.39	173.3
1,516.75	1,517.20	ICP-089	Ore—Polyhalite	400	17,238	2.86	169.2
1,507.65	1,508.20	ICP-097	Ore—Polyhalite	800	26,825	4.58	174.5
1,508.20	1,508.75	ICP-097	Ore—Polyhalite	1,200	20,553	3.75	173.3
1,508.20	1,508.75	ICP-097	Ore—Polyhalite	1,700	21,801	4.27	172.9

Table 16-10. ICP Brazilian Test Results

Depth (ft)		Exploration Hole No.	Lithology	Splitting Tensile Strength (psi)	Density (pcf)
To	From				
1,420.65	1,421.20	ICP-083	Floor—Halite/anhydrite	231	130.5
1,420.65	1,421.20	ICP-083	Floor—Halite/anhydrite	186	134.4
1,420.65	1,421.20	ICP-083	Floor—Halite/anhydrite	182	135.9
1,420.65	1,421.20	ICP-083	Floor—Halite/anhydrite	226	134.0
1,413.80	1,414.30	ICP-083	Floor—Halite/anhydrite	567	158.1
1,413.80	1,414.30	ICP-083	Floor—Halite/anhydrite	362	148.4
1,413.80	1,414.30	ICP-083	Floor—Halite/anhydrite	515	162.0
1,413.80	1,414.30	ICP-083	Floor—Halite/anhydrite	926	169.0
1,529.80	1,531.60	ICP-089	Floor—Halite/anhydrite	196	128.1
1,529.80	1,531.60	ICP-089	Floor—Halite/anhydrite	475	134.0
1,529.80	1,531.60	ICP-089	Floor—Halite/anhydrite	333	130.1
1,529.80	1,531.60	ICP-089	Floor—Halite/anhydrite	363	126.9
1,411.60	1,412.05	ICP-083	Ore—Polyhalite	599	167.8
1,411.60	1,412.05	ICP-083	Ore—Polyhalite	1,049	167.5
1,516.30	1,517.20	ICP-089	Ore—Polyhalite	886	168.8
1,516.30	1,517.20	ICP-089	Ore—Polyhalite	1,002	162.4
1,520.17	1,520.30	ICP-092	Ore—Polyhalite	901	167.7
1,507.65	1,508.75	ICP-097	Ore—Polyhalite	972	170.1
1,507.65	1,508.75	ICP-097	Ore—Polyhalite	715	165.1
1,405.45	1,406.35	ICP-083	Roof—Halite/anhydrite	310	125.5
1,405.45	1,406.35	ICP-083	Roof—Halite/anhydrite	514	128.1
1,405.45	1,406.35	ICP-083	Roof—Halite/anhydrite	407	122.2
1,405.45	1,406.35	ICP-083	Roof—Halite/anhydrite	468	125.6
1,507.10	1,507.65	ICP-089	Roof—Halite/anhydrite	105	111.7
1,507.10	1,507.65	ICP-089	Roof—Halite/anhydrite	316	128.2
1,507.10	1,507.65	ICP-089	Roof—Halite/anhydrite	218	125.2
1,507.10	1,507.65	ICP-089	Roof—Halite/anhydrite	139	117.7
1,515.40	1,515.55	ICP-092	Roof—Halite/anhydrite	407	138.4
1,490.85	1,491.00	ICP-093	Roof—Halite/anhydrite	346	134.9
1,492.50	1,492.60	ICP-093	Roof—Halite/anhydrite	1,018	176.6
1,496.15	1,497.80	ICP-097	Roof—Halite/anhydrite	259	127.0
1,496.15	1,497.80	ICP-097	Roof—Halite/anhydrite	277	128.3
1,496.15	1,497.80	ICP-097	Roof—Halite/anhydrite	370	131.1
1,496.15	1,497.80	ICP-097	Roof—Halite/anhydrite	167	101.6

Table 16-11. ICP Slake Durability Results

Depth (ft) To	Depth (ft) From	Exploration Hole No.	Lithology	Slake Durability Type	Slake Durability Index
1,404.75	1,405.45	ICP-083	Roof—Halite/anhydrite	II	11.9
1,507.10	1,507.65	ICP-089	Roof—Halite/anhydrite	III	54.8
1,497.35	1,497.80	ICP-097	Roof—Halite/anhydrite	II	5.2
1,516.30	1,516.35	ICP-089	Ore—Polyhalite	I	96.4
1,411.15	1,411.60	ICP-083	Ore—Polyhalite	I	96.1
1,507.65	1,508.20	ICP-097	Ore—Polyhalite	I	97.1
1,413.30	1,413.80	ICP-083	Floor—Halite/anhydrite	I	97.2
1,421.75	1,422.30	ICP-083	Floor—Halite/anhydrite	II	11.3
1,529.80	1,530.40	ICP-089	Floor—Halite/anhydrite	II	19.0
67.00	67.80	ICP-092	Mudstone	III	6.1
189.00	189.40	ICP-092	Sandstone	I	95.0
283.70	284.10	ICP-092	Sandstone	II	91.4
371.30	371.80	ICP-092	Mudstone siltstone	I	95.6
472.70	473.20	ICP-092	Sandstone	I	96.7
696.50	697.00	ICP-092	Sandstone	I	98.5
1,067.70	1,068.10	ICP-092	Siltstone	II	91.7
1,321.50	1,322.40	ICP-092	Halite	III	11.9
1,359.20	1,359.60	ICP-092	Dolomite	I	92.9

Table 16-12. ICP Priority 2 UCS-E Results

Depth (ft) To	Depth (ft) From	Exploration Hole No.	Lithology	UCS (psi)	Young's Modulus (Mpsi)	Density (pcf)
54.20	54.80	ICP-097	Sandstone mudstone	646	0.07	147.5
57.40	58.00	ICP-097	Sandstone mudstone	395	0.03	145.9
67.00	67.50	ICP-092	Mudstone	62	0.02	124.0
78.60	79.20	ICP-097	Mudstone siltstone	989	0.06	149.3
81.40	82.00	ICP-097	Siltstone mudstone	1,001	0.05	149.7
94.70	95.20	ICP-092	Mudstone sandstone	155	0.04	149.3
107.60	108.20	ICP-097	Mudstone siltstone	1,767	0.15	149.3
109.60	110.20	ICP-097	Mudstone siltstone	1,815	0.23	149.0
125.20	125.70	ICP-092	Sandstone	2,022	0.24	147.1
126.80	127.40	ICP-097	Sandstone mudstone	2,414	0.52	150.0
128.60	129.20	ICP-097	Sandstone mudstone	366	0.03	152.8

Table 16-12. ICP Priority 2 UCS-E Results *(concluded)*

Depth (ft) To	Depth (ft) From	Exploration Hole No.	Lithology	UCS (psi)	Young's Modulus (Mpsi)	Density (pcf)
154.80	155.30	ICP-092	Siltstone mudstone	1,849	0.21	157.0
187.60	188.10	ICP-092	Sandstone	4,812	0.83	175.6
227.00	227.50	ICP-092	Sandstone siltstone	3,329	0.46	98.4
260.60	261.10	ICP-092	Mudstone siltstone	2,763	0.42	157.6
283.10	283.60	ICP-092	Sandstone	1,572	0.27	155.7
327.30	327.80	ICP-092	Mudstone siltstone	4,792	0.64	161.5
440.00	440.50	ICP-092	Mudstone siltstone	3,927	0.64	161.8
472.20	472.70	ICP-092	Sandstone	690	0.12	146.1
557.30	557.80	ICP-092	Conglomerate sandstone	4,188	0.89	152.3
619.60	620.10	ICP-092	Sandstone	3,056	0.65	143.0
694.10	694.60	ICP-092	Mudstone siltstone	4,385	0.70	156.8
723.00	723.60	ICP-093	Sandstone siltstone	4,982	1.30	129.6
739.20	739.75	ICP-093	Sandstone siltstone	3,036	0.62	151.3
817.50	818.10	ICP-093	Siltstone	8,495	1.80	154.0
842.20	842.70	ICP-092	Siltstone mudstone	3,745	1.03	155.3
825.40	826.00	ICP-093	Siltstone mudstone	8,989	1.87	153.8
826.00	826.60	ICP-093	Siltstone mudstone	3,649	0.64	155.6
867.20	867.70	ICP-092	Siltstone mudstone	4,776	1.04	152.9
848.40	849.00	ICP-093	Siltstone mudstone	4,839	1.70	155.0
852.60	853.20	ICP-093	Siltstone sandstone	5,422	3.76	158.4
863.70	864.30	ICP-093	Sandstone	10,689	2.75	155.7
883.70	884.30	ICP-093	Siltstone	6,011	3.86	154.9
945.90	946.40	ICP-092	Siltstone sandstone	13,281	3.24	161.2
1,027.75	1,028.30	ICP-093	Mudstone	8,378	1.79	162.5
1,068.25	1,068.75	ICP-092	Siltstone	6,391	1.98	160.8
1,094.10	1,094.70	ICP-093	Sandstone siltstone	10,275	3.85	162.8
1,103.60	1,104.20	ICP-093	Siltstone mudstone	10,192	2.61	160.4
1,121.30	1,121.90	ICP-093	Sandstone siltstone	9,389	2.42	159.7
1,141.30	1,141.90	ICP-093	Siltstone	9,528	2.53	159.6
1,277.56	1,278.10	ICP-092	Anhydrite	16,805	10.10	184.7
1,313.03	1,313.56	ICP-092	Argillaceous halite	3,408	1.70	134.9
1,346.95	1,347.95	ICP-092	Anhydrite	20,349	7.32	183.3
1,360.30	1,360.82	ICP-092	Dolomite	3,814	8.62	155.9
1,433.00	1,433.53	ICP-092	Halite	2,869	0.71	135.2
1,593.00	1,593.55	ICP-092	Dolomite	7,407	2.24	160.4
1,576.00	1,576.55	ICP-092	Anhydrite	14,645	7.17	183.3
1,561.00	1,561.51	ICP-092	Halite	3,019	2.04	135.5
1,485.20	1,485.72	ICP-092	Halite	3,096	1.06	135.2
1,546.00	1,546.55	ICP-092	Halite	2,580	0.93	135.5

with an average of approximately 1.5 ft. In mining the polyhalite, it would be advantageous to keep this anhydrite layer in the roof, either permanently in the immediate roof, or at least temporarily as the ore is mined, so that it can be segregated from the ore without being transported to the surface. However, this may prove difficult as a clay-filled joint usually is present near the top of the anhydrite layer, just below the transition to the overlying halite, and the anhydrite layer may not be self-supporting in the roof as the polyhalite is mined below it. In recovered core, this joint is usually open, because the clay material is typically washed out by the drilling fluids.

The halite of the immediate and main roof is between 40 and 140 ft thick. Above this halite layer, thick sequences of anhydrite, halite and, occasionally, dolomite extend for about an additional 150 ft. The ore zone and surrounding strata are competent and intact, with RQD values consistently near 100%. Above the anhydrite/halite sequences, from approximately 1,300 ft of depth to the surface, the strata primarily consist of various layers of sandstone, mudstone, and siltstone.

A summary of average values from the testing for the near-seam rock types is given in Table 16-13.

Table 16-13. ICP Mechanical Properties—Summary of Laboratory Test Results

Strata	UCS (psi)	Tensile Strength (psi)	Specific Weight (pcf)	Young's Modulus (Mpsi)	Poisson's Ratio	Cohesion (psi)	Angle of Friction (°)
Halite (roof)	3,970	290	133	0.89	0.35	1,510	22.8
Anhydrite (roof)	8,830	290	172	1.29	0.44	1,510	22.8
Polyhalite (ore)	9,630	870	168	1.73	0.37	4,650	31.8
Halite/anhydrite (floor)	3,030	380	145	2.16	0.37	1,260	20.1

Pillar design was conducted using 3D numerical (Itasca FLAC3D, Version 5, 2014) and empirical modeling techniques. 3D modeling was used to evaluate the 90% and 60% extraction ratios proposed in the PFS (Gustavson 2012), and the potential standup time for the thin layer of roof anhydrite. Modeling results indicated that the 90% extraction ratio may be problematic, and a 60% extraction ratio was adopted for the FS. Modeling results indicate that higher extraction ratios may be possible. Figures 16-7 and 16-8 show the pillar dimensions for the 90% and 60% production panel extraction ratios.

In addition to FLAC3D numerical modeling techniques, for each of the pillar geometries, stability factors (SFs) were calculated using various empirical design methods applicable for both hard rock and soft rock environments. The six methods used were:

- United States Bureau of Mines (USBM, Obert and Duvall 1967)
- Canada Center for Mineral and Energy Technology (CANMET, Hedley and Grant 1972)
- South African Council for Scientific and Industrial Research (CSIR, Bieniawski 1984)
- Stacey-Page Pillar Strength Formula (Stacey and Page 1986)
- Hardy-Agapito Method (Hardy and Agapito 1975)
- Abel-Wilson Method (Abel 1988; Wilson and Ashwin 1972)

Table 16-14 summarizes the pillar strengths and resulting SFs for the various pillars (90% extraction panel entry pillars on development, 90% extraction panel entry pillars on retreat, and 90% extraction room pillars; mains pillars, 60% extraction panel entry pillars, 60% extraction room pillars). The pillar heights correspond to the minimum height in the mains (6.5 ft), the expected mining heights around the shaft and slope bottom (8 ft and 12 ft, respectively), and the minimum mining height in the panels (5.2 ft). Two cases of mining height in the panels are represented, and assume the anhydrite roof is taken in addition to a 5-ft ore height: 5.5-ft mining height represents 0.5 ft of anhydrite and 6.5 ft represent 1.5 ft of anhydrite.

As shown in Table 16-14, mains and 60% extraction pillar SFs are predicted to be adequate regardless of the method used to estimate pillar strength. The 90% extraction pillar SFs indicate that pillar yielding (failure) will likely occur, as is appropriate for a yield pillar design. Successful yield pillar designs require that yielding occur in a controlled and time-dependent manner, allowing for safe conditions at the pillar line where men and equipment are working. However, if the yielding is excessive or uncontrolled at the pillar line, or if the failure occurs suddenly or violently, unacceptable conditions such as instantaneous or cascading pillar failure (CPF), roof falls, floor heave, or rock bursts may result. The empirical design approaches give little insight into the mechanism of pillar yielding; therefore, pillar stability and the mechanisms of potential failure were examined using the 3D numerical modeling program FLAC3D.

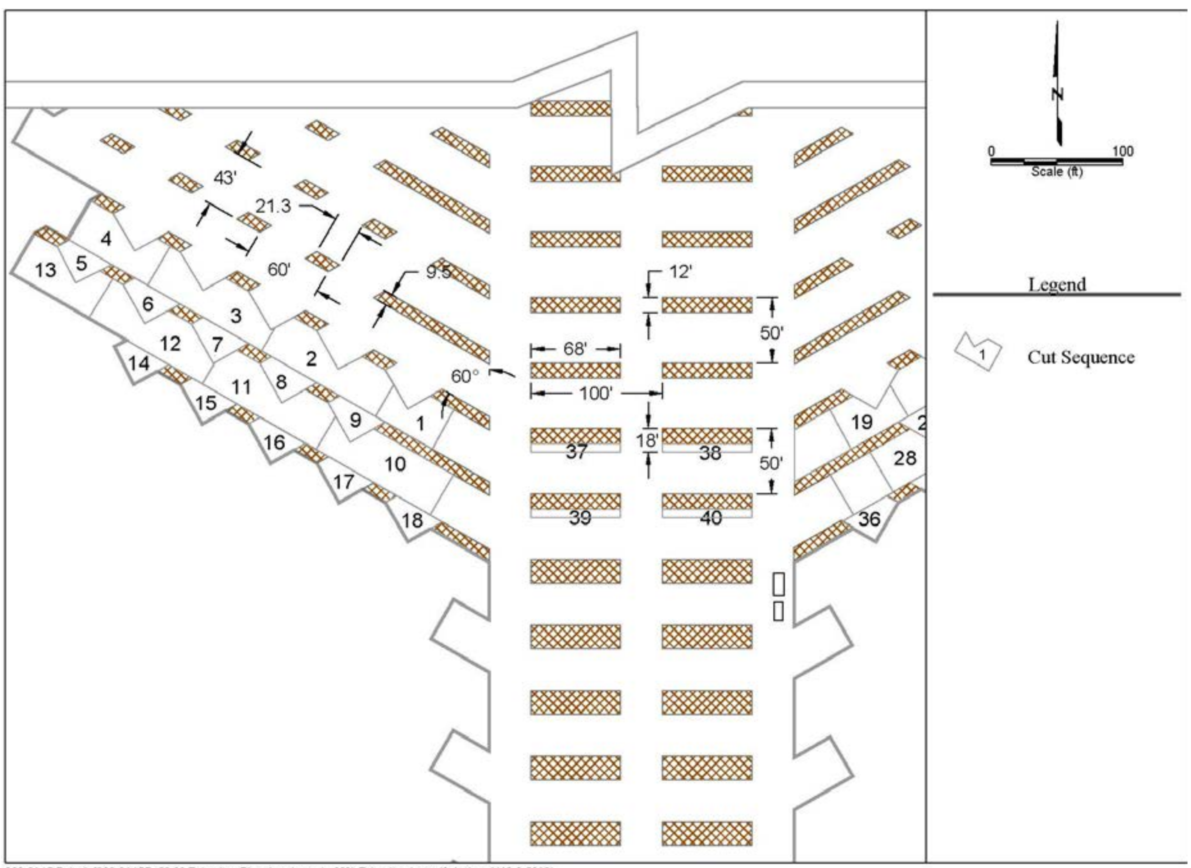


Figure 16-7. Pillar Dimensions for 90% Extraction Plan

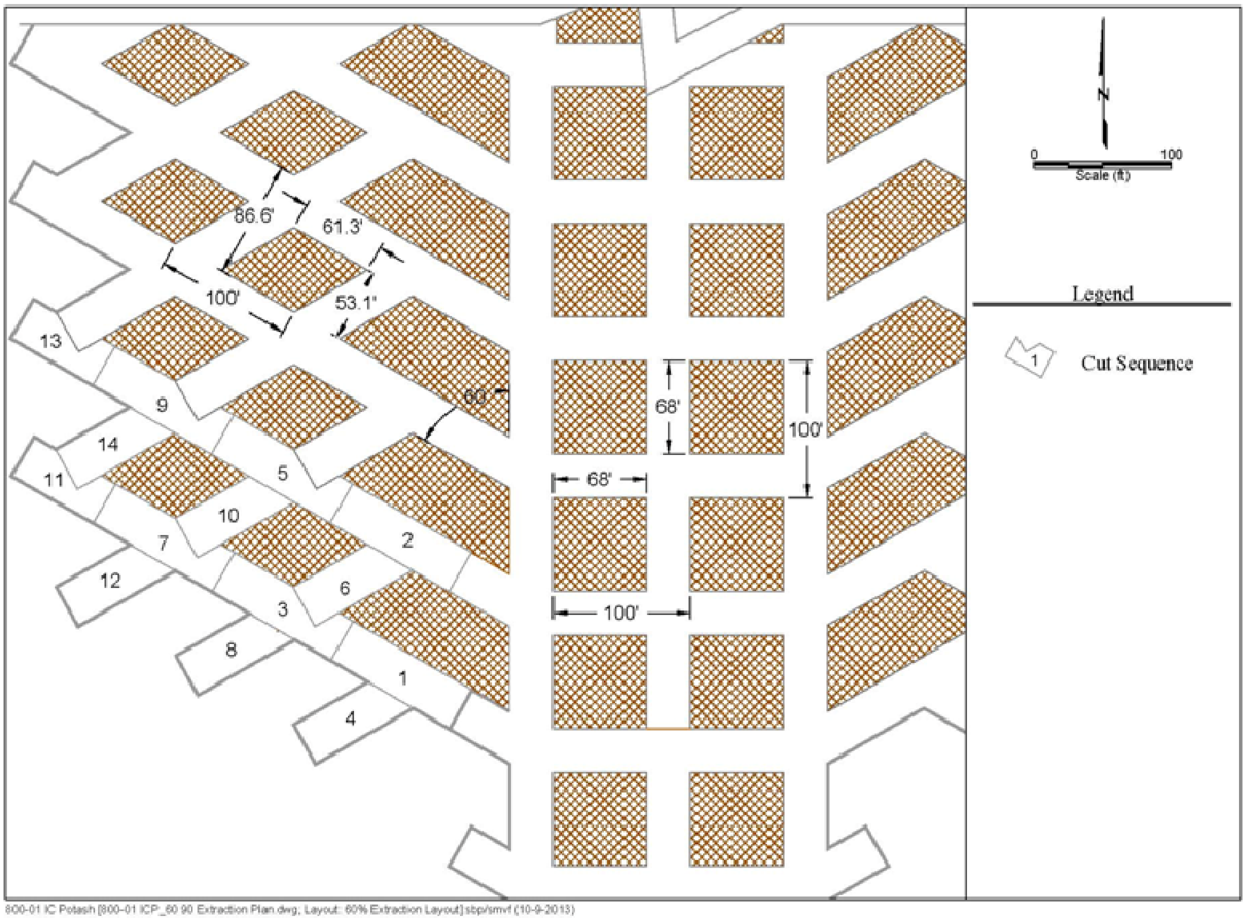


Figure 16-8. Pillar Dimensions for 60% Extraction Plan

Table 16-14. Summary of Emprirical Pillar Strengths and Resulting Stability Factors

Case	Design Dimensions										
	Pillar Length (ft)	Pillar Width (ft)	Crosscut Center (ft)	Entry Center (ft)	Least Pillar Dimension (ft)	Room Width (ft)	CrosscutWidth (ft)	Pillar Height (ft)	Effective Width (ft)	Extraction Ratio	Pillar Stress (psi)
Mains, 6.5 ft high	77	77	100	100	77	23	23	6.5	77.0	40.7%	2,530
Mains, 8 ft high	77	77	100	100	77	23	23	8	77.0	40.7%	2,530
Mains, 12 ft high	77	77	100	100	77	23	23	12	77.0	40.7%	2,530
60% Extraction, Panel Entries, 5.2 ft high	68	68	100	100	68	32	32	5.2	68.0	53.8%	3,244
60% Extraction, Panel Entries, 5.5 ft high	68	68	100	100	68	32	32	5.5	68.0	53.8%	3,244
60% Extraction, Panel Entries, 6.5 ft high	68	68	100	100	68	32	32	6.5	68.0	53.8%	3,244
60% Extraction, Rooms, 5.2 ft high	61.3	53.1	100	86.6	53.1	33.5	33.5	5.2	56.9	62.4%	3,991
60% Extraction, Rooms, 5.5 ft high	61.3	53.1	100	86.6	53.1	33.5	33.5	5.5	56.9	62.4%	3,991
60% Extraction, Rooms, 6.5 ft high	61.3	53.1	100	86.6	53.1	33.5	33.5	6.5	56.9	62.4%	3,991
90% Extraction, Panel Entries, Development, 5.2 ft high	68	18	50	100	18	32	32	5.2	28.5	75.5%	6,127
90% Extraction, Panel Entries, Development, 5.5 ft high	68	18	50	100	18	32	32	5.5	28.5	75.5%	6,127
90% Extraction, Panel Entries, Development, 6.5 ft high	68	18	50	100	18	32	32	6.5	28.5	75.5%	6,127
90% Extraction, Panel Entries, Retreat, 5.2 ft high	68	11.6	50	100	11.6	32	38.4	5.2	19.8	84.2%	9,508
90% Extraction, Panel Entries, Retreat, 5.5 ft high	68	11.6	50	100	11.6	32	38.4	5.5	19.8	84.2%	9,508
90% Extraction, Panel Entries, Retreat, 6.5 ft high	68	11.6	50	100	11.6	32	38.4	6.5	19.8	84.2%	9,508
90% Extraction, Rooms, 5.2 ft high	21.3	9.5	60	43	9.5	33.5	33.5	5.2	13.1	92.2%	19,125
90% Extraction, Rooms, 5.5 ft high	21.3	9.5	60	43	9.5	33.5	33.5	5.5	13.1	92.2%	19,125
90% Extraction, Rooms, 6.5 ft high	21.3	9.5	60	43	9.5	33.5	33.5	6.5	13.1	92.2%	19,125

Table 16-14. Summary of Emprirical Pillar Strengths and Resulting Stability Factors *(concluded)*

Case	Pillar Strength (psi)						Resulting Stability Factors					
	USBM (Obert-Duvall)	CANMET (Hedley-Grant)	CSIR (Bieniawski)	Stacy-Page	Hardy-Agapito	Abel-Wilson	USBM (Obert-Duvall)	CANMET (Hedley-Grant)	CSIR (Bieniawski)	Stacy-Page	Hardy-Agapito	Abel-Wilson
Mains, 6.5 ft high	9,472	14,528	13,632	14,971	20,213	8,222	3.74	5.74	5.39	5.92	7.99	3.25
Mains, 8 ft high	8,101	12,433	11,409	12,946	16,591	8,194	3.20	4.91	4.51	5.12	6.56	3.24
Mains, 12 ft high	6,122	9,173	8,199	9,747	11,283	8,117	2.42	3.63	3.24	3.85	4.46	3.21
60% Extraction, Panel Entries, 5.2 ft high	10,231	16,139	14,863	16,447	23,205	8,233	3.15	4.98	4.58	5.07	7.15	2.54
60% Extraction, Panel Entries, 5.5 ft high	9,791	15,474	14,149	15,814	21,999	8,227	3.02	4.77	4.36	4.87	6.78	2.54
60% Extraction, Panel Entries, 6.5 ft high	8,617	13,652	12,246	14,069	18,768	8,206	2.66	4.21	3.78	4.34	5.79	2.53
60% Extraction, Rooms, 5.2 ft high	8,463	14,262	11,996	15,046	19,683	8,211	2.12	3.57	3.01	3.77	4.93	2.06
60% Extraction, Rooms, 5.5 ft high	8,119	13,674	11,439	14,467	18,660	8,203	2.03	3.43	2.87	3.63	4.68	2.06
60% Extraction, Rooms, 6.5 ft high	7,203	12,064	9,953	12,870	15,919	8,177	1.80	3.02	2.49	3.22	3.99	2.05
90% Extraction, Panel Entries, Development, 5.2 ft high	4,298	8,304	5,242	10,641	8,971	8,079	0.70	1.36	0.86	1.74	1.46	1.32
90% Extraction, Panel Entries, Development, 5.5 ft high	4,182	7,962	5,053	10,232	8,505	8,056	0.68	1.30	0.82	1.67	1.39	1.31
90% Extraction, Panel Entries, Development, 6.5 ft high	3,871	7,024	4,550	9,102	7,256	8,011	0.63	1.15	0.74	1.49	1.18	1.31
90% Extraction, Panel Entries, Retreat, 5.2 ft high	3,539	6,666	4,011	8,879	6,553	7,959	0.37	0.70	0.42	0.93	0.69	0.84
90% Extraction, Panel Entries, Retreat, 5.5 ft high	3,464	6,391	3,889	8,537	6,212	7,941	0.36	0.67	0.41	0.90	0.65	0.84
90% Extraction, Panel Entries, Retreat, 6.5 ft high	3,263	5,639	3,564	7,595	5,300	7,871	0.34	0.59	0.37	0.80	0.56	0.83
90% Extraction, Rooms, 5.2 ft high	3,290	6,032	3,607	7,230	6,515	7,750	0.17	0.32	0.19	0.38	0.34	0.41
90% Extraction, Rooms, 5.5 ft high	3,228	5,784	3,507	6,951	6,176	7,750	0.17	0.30	0.18	0.36	0.32	0.41
90% Extraction, Rooms, 6.5 ft high	3,064	5,103	3,241	6,184	5,269	7,613	0.16	0.27	0.17	0.32	0.28	0.40

The production panel plans were evaluated using the FLAC3D program, a 3D finite-difference numerical modeling code. This approach allows judgments to be made regarding pillar, roof, and floor stability for the pillar and entry dimensions of the 90% extraction and 60% extraction plans. In addition, larger models were created to study the adequacy of the barriers between the production panels and the mains.

A simulated model lithology is shown in Figure 16-9. Polyhalite thickness was set at 5 ft in all models. Immediate roof anhydrite thickness was set at 0.5 ft for the pillar analysis models (roof stability analysis models, discussed later, also incorporated 1.5 ft of anhydrite). The mining height included both the polyhalite and immediate roof anhydrite layers. Main roof halite and floor halite/anhydrite were assumed to be of infinite thickness, for modeling convenience. Creep of the main roof halite is anticipated with time, and this creep behavior was simulated in the model. All strata were assumed to be flat-lying and of constant thickness. The only discontinuity modeled was the polyhalite/anhydrite contact in the immediate roof.

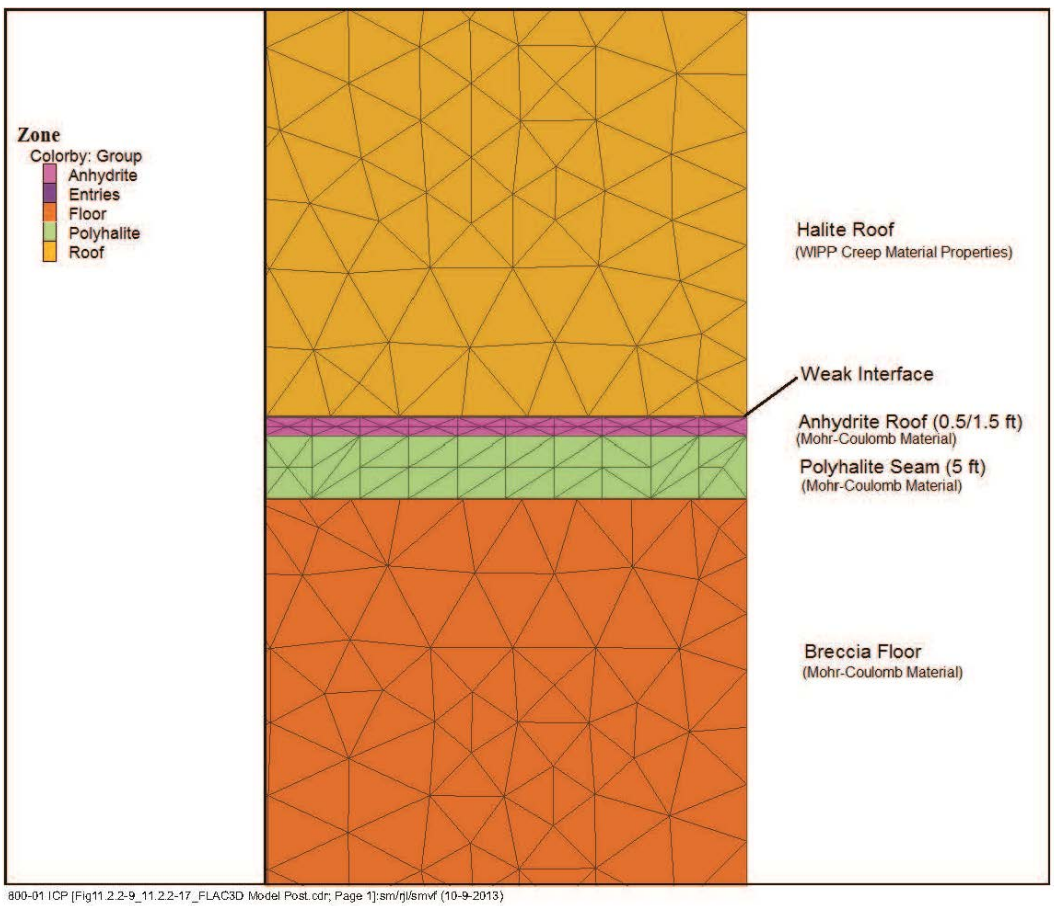


Figure 16-9. Representative Model Section

Laboratory test results from Table 16-13 were used as the basis for model input parameters. Because laboratory values tend to overestimate rock mass values, the laboratory values were scaled by applying the Hoek-Brown criterion (Hoek, Carranza-Torres, and Corkum 2002). Hoek-Brown parameters used to convert laboratory parameters to rock mass parameters were selected based on previous experience with similar rock types and engineering judgment, and are presented Table 16-15. Resulting rock mass properties used in the models are presented in Table 16-16.

Table 16-15. Hoek-Brown Parameters

Strata	Geological Strength Index	mi*
Halite (roof)	95	10
Anhydrite (roof)	85	12
Polyhalite (ore)	85	12
Halite/anhydrite (floor)	55	18
"mi" is a material constant and a measure of texture (consequently, internal friction angle) of the intact rock grains.		

Table 16-16. Summary of Model Input Parameters

Strata	Cohesion (psi)	Angle of Friction (°)	Tensile Strength (psi)	UCS (psi)	Bulk Modulus (Mpsi)	Shear Modulus (Mpsi)
Halite (roof)	625	42	260	3,000	0.9	0.3
Anhydrite (roof)	945	41	220	3,825	3.1	0.4
Polyhalite (ore)	970	41	220	3,950	2.0	0.6
Halite/anhydrite (floor)	195	37	5	250	1.2	0.3
Halite/anhydrite roof interface	0	45.2	0	n/a	n/a	n/a
n/a = Not available						

Creep parameters for the main roof halite were assigned based on the WIPP model within FLAC3D (Table 16-17), with the exception of the bulk and shear moduli, which were derived from the available laboratory data. A review of historical creep test data suggest that the polyhalite does not creep; therefore, all materials other than the roof halite were modeled using the assumption of linear elasticity up to a failure limit defined by Mohr-Coulomb failure criteria. Post-failure behavior of the polyhalite was simulated in two ways: (1) perfectly plastic and (2) strain softening. These two post-failure models were included to study the range of likely conditions that might be associated with a yield pillar design (90% extraction). In both approaches, the floor and immediate roof anhydrite were perfectly plastic.

Table 16-17. Halite Roof Creep Parameters

Parameter	Value
Activation energy (cal/mol)	12,000
A-constant	4.56
B-constant	127
D-constant (Pa-4.9/sec)	5.79^{-36}
N-constant	4.9
Gas constant (cal/mol/Kelvin)	1.987
Critical strain rate	5.39^{-8}

Since creep behavior is sensitive to temperature, the developed models were assigned a constant temperature of 76°F (298 Kelvin). Groundwater was not simulated in the models. Vertical gravitational loads were applied to the models; horizontal stresses were not explicitly applied but were generated in the model based on the Poisson's effect.

Because creep of the main roof halite was included in the modeling, time affects the results. The time in the models represents production panel development, followed by 3 months of retreat mining. Even though the panel development and retreat will involve multiple unit operations in reality, the cumulative excavation stages were simulated in a single step followed by an appropriate period of creep, in order to optimize the computational effort.

The analysis for the 90% and 60% extraction ratio production panel plans were run using perfectly plastic post-failure behavior and also with strain-softening behavior. The latter was developed by setting the post-failure strength of the polyhalite at 30% of its peak strength.

Results for the 90% extraction ratio production panels modeled with plastic post-failure behavior show that significantly high vertical stresses close to 13,000 psi are indicated in the panel pillars and in the abutment zones around the panel. Relatively high stress concentrations are also indicated in the ribs of the room with active retreat operations, with vertical stress values close to 7,250 psi. Results for the 90% extraction ratio production panels modeled with strain-softening behavior show that the production pillars have yielded and largely transferred the overburden loads to the abutments around the panel. This leads to stress concentrations in excess of 13,000 psi in the yet-to-be-mined polyhalite at the panel periphery. This degree of stress concentration at and near the active face may result in untenable ground conditions. Vertical displacement shows pillar deformation of approximately 4 ft in the pillar immediately outby the face, and complete closure toward the center of the panel.

Models of the 60% extraction plan were also analyzed using perfectly plastic and strain softening post-failure behavior of the polyhalite. In contrast to the 90% extraction plan, model results show production pillars to be stable, with yielding only occurring a small distance into pillar ribs. Neither floor nor roof failure is indicated. Displacement results indicate a minimal level of pillar deformation (less than 0.4 inches). The small magnitudes of floor heave indicated (0.6 inches), arising from elastic deformation, are unlikely to translate to conditions that would have an adverse impact on operations. Similar results are shown for the 60% extraction plan with strain-softening polyhalite properties. As expected, because yielded pillar ribs are allowed to carry a lower load than in the perfectly plastic case, some additional rib yielding is indicated as well as marginally greater closures. Figure 16-10 illustrates a sample FLAC3D output showing vertical stress in the 90% extraction panel, and Figure 16-11 shows the same output for the 60% extraction panel.

Overall, the 90% extraction models indicate pillar failure not only inby active mining, but near and outby the working face. The perfectly plastic and strain-softening models illustrate two post-failure behaviors, neither of which is likely to result in acceptable mining conditions. The strain-softening models indicate complete closure at the center of the panel and high levels of closure at and near the face. The perfectly plastic models likewise show excessive yielding near the face. Additionally, the models suggest another hazard that could occur should the post-failure behavior of the polyhalite fall somewhere between the two assumptions. If overstressed pillars have a more brittle post-failure behavior (sudden failure and release of energy), rockbursts and CPFs are possible. CPF can occur when similar-sized pillars with low SFs are used over large areas. Failure in one pillar results in stress transfer to adjacent pillars, which, in turn, fail. In their mildest form (slow pillar squeezes), this failure may take weeks to progress, and such behavior is desirable in yield pillar design. In their most severe form, failures can occur almost instantaneously, resulting in severe air blasts, damage to equipment, and loss of life.

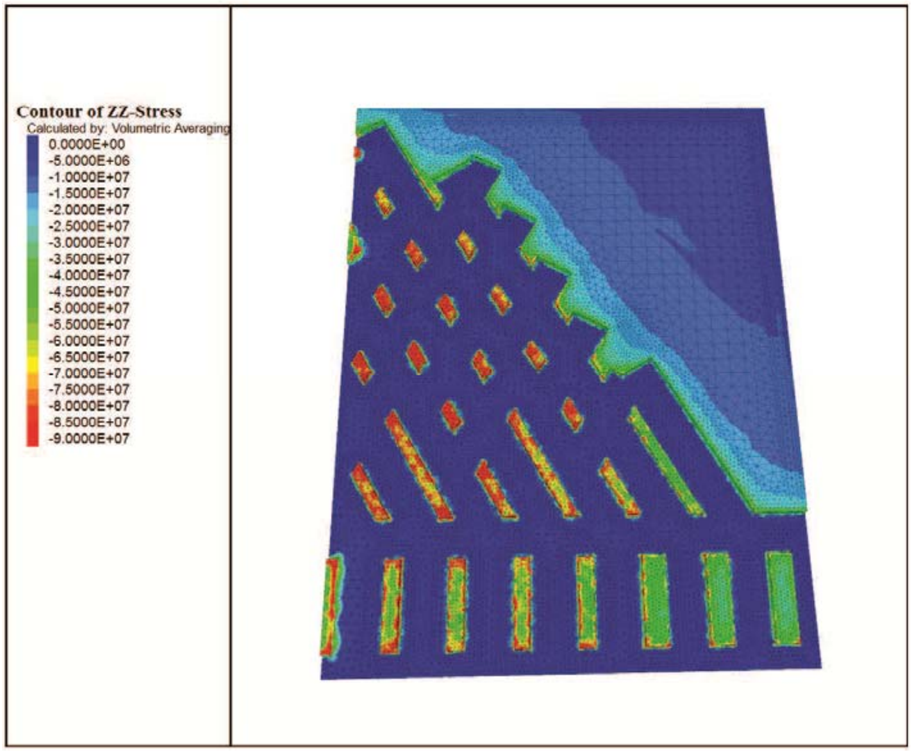


Figure 16-10. Vertical Stress (MPa) **in the High-Extraction** (90%) **Panel**

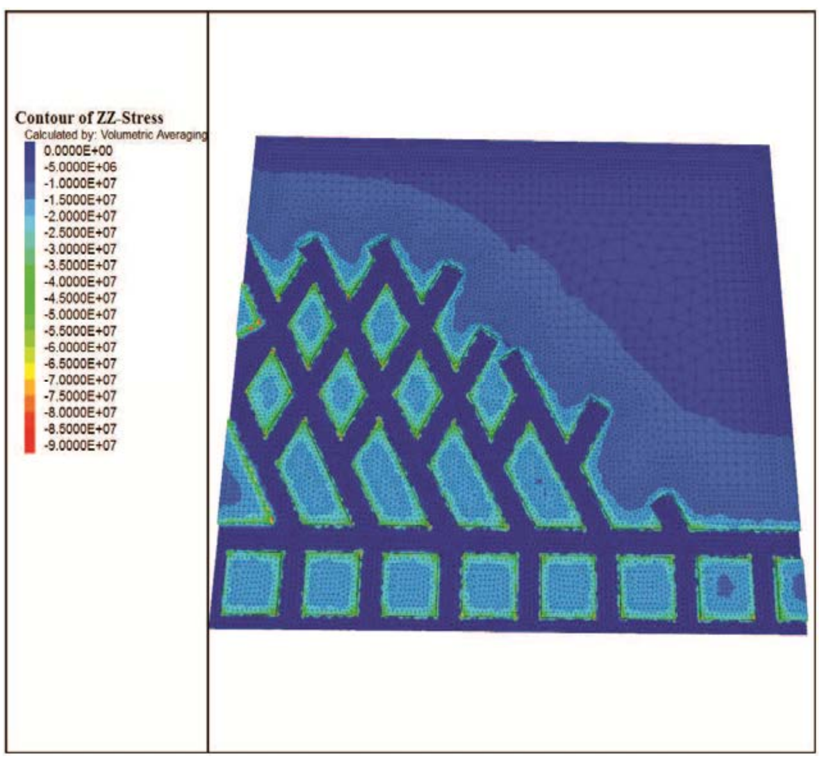


Figure 16-11. Vertical Stress (MPa) **in the Low-Extraction** (60%) **Panel**

It is difficult at this point of design to reliably characterize the post-failure behavior of polyhalite yield pillars, and indeed, it may not be possible to completely characterize that behavior prior to mining. While experimentation with a yield pillar design may prove successful, this analysis indicates that such a design would not be successful. For this reason, AAI recommended that the 60% extraction plan be used as the basis for the FS. The 60% extraction ratio is conservative and the results from that modeling indicate that higher extraction ratios are likely feasible.

The modeling results indicate that the entry widths associated with the mains/submains (23 ft) and the 60% extraction panels (up to 33.5 ft) appear feasible.

The adequacy of the mains/submains pillars and the width of the barrier pillars were evaluated for both the 90% and 60% extraction ratio cases. The sequential mining of two adjacent panels was simulated with each panel being allowed to creep for 6 months after completion of secondary mining. The vertical stress contours in and around the first 60% extraction panel are presented in Figure 16-12. The vertical stress distribution 6 months after retreat of the first panel does not indicate an unusually significant stress transfer to the adjacent unmined panel to the right, as the peak stress concentration in the unmined panel is about 1.5 times the virgin stress.

Figure 16-13 shows the vertical stress distribution 6 months after the second panel has been mined, and indicates essentially no stress transfer across the 400-ft end-panel barrier separating the production panel from the mains/submains. A similar analysis was performed on the 90% extraction plan, although results are limited to first panel mining (a model with second panel mining was terminated due to time requirements). The vertical stress distribution following retreat mining in the first panel of the 90% extraction plan is shown in Figure 16-14. As expected, the abutment zone extends further from the panel into the barrier; however, the locations of the mains/ submains still see very little of the abutment load.

Based on these results, it appears that (1) a 400-ft end-panel barrier pillar width is adequate to protect the mains/submains, (2) the mains/submains pillars (100-ft by 100-ft centers, 77 ft by 77 ft rib-to-rib) are adequately sized, and (3) stresses transferred from one 60% extraction panel are unlikely to significantly impact the adjacent panel.

Roof Support Design—Roof stability and roof support design was addressed using empirical and analytical roof support estimation techniques. These techniques included National Institute of Occupational Safety and Health (NIOSH) methods such as the Analysis of Roof Bolting Systems (ARBS) program (NIOSH 2003). In this method, roof lithology, strength, entry span, and depth of cover loading conditions are calculated and compared to support capacities from NIOSH's empirical database to determine acceptable bolting systems.

Given the thin anhydrite layer underlying the massive halite roof, with the intervening clay-filled joint, it is likely that roof bolts would primarily act to suspend the anhydrite or to ensure that a beam of adequate thickness is developed in the halite. The analysis described below addresses this assertion. Table 16-18 illustrates the results of the ARBS analysis.

From a dead-weight suspension standpoint, the same system could be used to support the anhydrite layer in the production panels. With the same support density as discussed above (with six bolts per row across the 33.5-ft production panel room span), each bolt would carry a

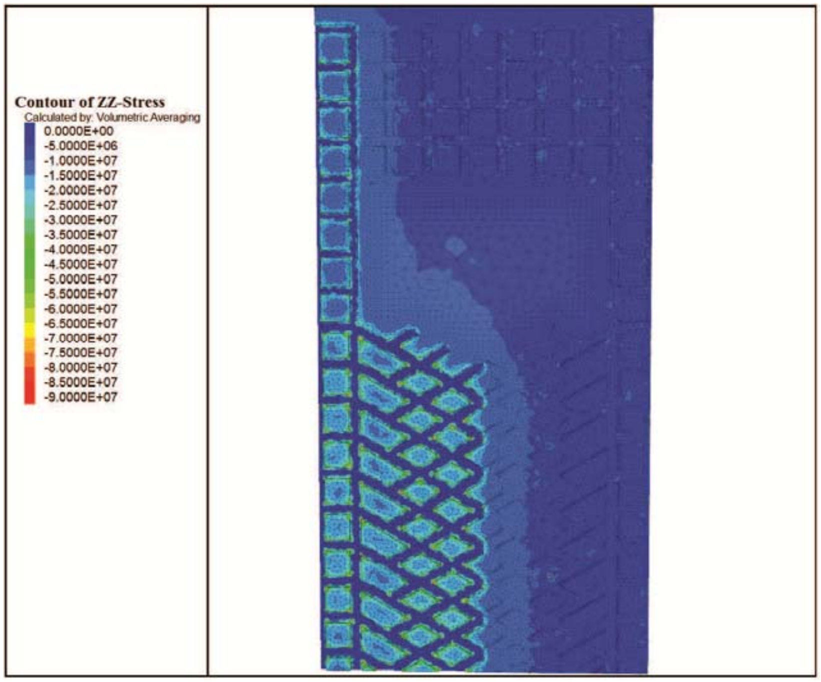
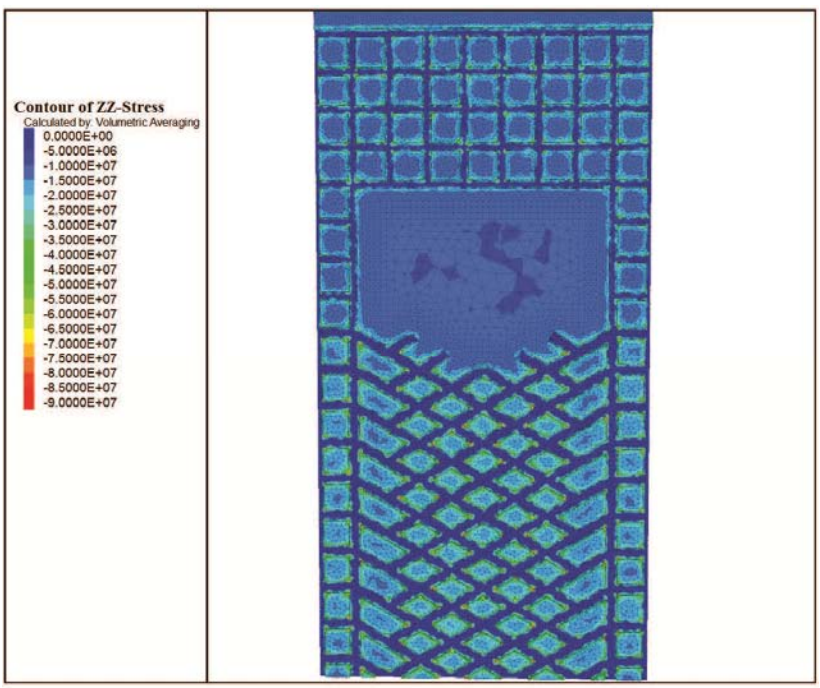
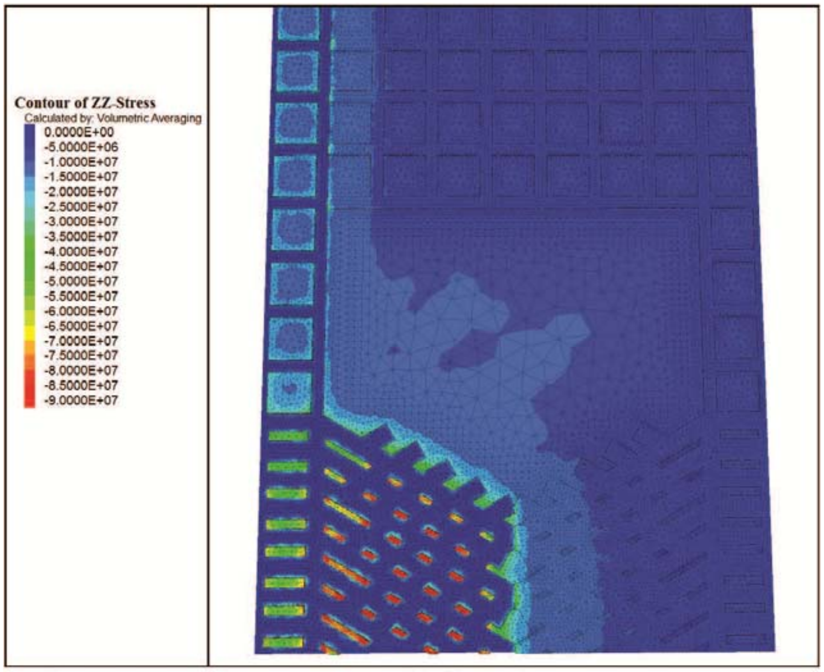


Figure 16-12. Vertical Stress (MPa) **in the Low-Extraction** (60%) **Panel, Barrier and Mains Pillars 6 Months after Completion of First Panel**



800-01 ICP [Fig11.2.2-9_11.2.2-17_FLAC3D Model Post.cdr; Page 10]:sm/gl/smvf (10-10-2013)

Figure 16-13. Vertical Stress (MPa) **in the Low-Extraction** (60%) **Panel, Barrier and Mains Pillars 6 Months after Completion of Second Panel**



800-01 ICP [Fig11.2.2-9_11.2.2-17_FLAC3D Model Post.cdr; Page 11]:sm/rjl/smvf (10-10-2013)

Figure 16-14. Vertical Stress (MPa) **in the High-Extraction** (90%) **Panel, Barrier and Mains Pillars** (before advancing mains past panel neck)

Table 16-18. Results of the ARBS Analysis

Roof Diagonal Span (ft)	Coal Mine Roof Rating	Bolt Length (ft)	Bolt Grade (ksi)	Bolt Number*	Bolt Capacity (t)	Bolts per Row	Row Spacing (ft)
33	58	4	75	6	16.6	4	5.0

*Corresponds to eighths of an inch (i.e., a #7 bolt is nominally ⅞ inch diameter).
Note: ksi = thousand pounds per square inch.

dead weight load of approximately 3 t, assuming a 1.5-ft-thick anhydrite layer below the clay-filled joint. Given the bolt capacity of 16.6 t, the resulting SF in suspension is 5.5.

Additional analysis was performed based on Merrill's Roof Assessment Methodology (Merrill 1954), which analyzes the roof stability as an elastic beam, and the Equivalent Support Pressure Method (Cording, Hendron, and Deere 1971) which calculates the equivalent uniform support pressure to achieve stability based upon large-span underground openings in rock. The results of these analyses are summarized in Table 16-19.

Comparing the results in Tables 16-18 and 16-19, both the ARBS and Merrill/Cording methods point to a system of No. 6, grade 75 bolts, with approximately 5-ft spacing both within and between rows of bolts (four bolts across the 23-ft main/submain entries, and six bolts across the production panel entries and rooms). This system with a 4 ft length is considered a reasonable assumption for the FS, as it is applicable for suspending the anhydrite in the production panels and as reinforcement of the halite in both the mains/submains and production panels.

Table 16-19. Required Roof Beam Thickness and Bolt Spacing

Parameter	Location: Mains	Location: Rooms
Diagonal span (ft)	33	53
Unconfined compressive strength (UCS) (psi)	3,970	3,970
Estimated modulus of rupture (R) based on UCS (psi)	473	473
Density (pcf)	133	133
Unsupported SF by the Merrill method	1.50	1.50
Bar size (⅛ inch)	6	6
Bar grade (ksi)	75	75
Required roof beam thickness (ft)	1.60	4.11
Spacing to get n= 0.20 equivalent support pressure (ft)	6.1	4.8

The PFS mine design was based on selectively mining the roof anhydrite in the production panel entries and main entries, gobbing it in outby crosscuts, and loading it out during a shift change window. This approach was modeled in the FS to test its viability. If the anhydrite can be selectively mined to segregate it from the polyhalite, the proposed mining approach in the production panels would be to take a 20-ft-deep miner cut in the polyhalite, load it out, then cut down and gob the exposed anhydrite roof layer. The anhydrite would be gobbed in adjacent inby rooms as they are mined on panel retreat. This would be repeated laterally (in passes) to create a room three continuous miner cutter head-widths wide before advancing the next 20-ft-deep cut to that width. The ability of the anhydrite to remain stable temporarily to allow this process was analyzed with FLAC3D using the 60% extraction plan. Figure 16-15 depicts a step in the sequence. The sequence of passes was modeled from panel pillar to solid rib (pillar-to-solid) and the reverse sequence was also analyzed. Creep of the halite was included in the modeling scenario, with time simulated to correspond to the mining cycle.

Overall, the results indicate that a thicker anhydrite layer is likely to provide longer stand time than the thinner layer. A solid-to-pillar sequence of passes is recommended to reduce tensile and shear stress concentrations. Even though results suggest that cuts up to 20 ft are likely to be stable prior to removal of the anhydrite in the first pass, it is recommended that a cut length be no longer than 10 ft in the first pass to account for geological anomalies such as discontinuities, pinch-outs, etc. The second passes should be limited to 5 ft and 7.5 ft depths and finally 5-ft cut depth increments in the third pass prior to removal of the anhydrite.

To minimize productivity losses due to anhydrite gobbing time, therefore impacting the required number of continuous miners and crews required to produce the required annual ROM ore tonnage, the scenario of leaving the immediate roof anhydrite layer and supporting it with roof bolts was evaluated. This scenario would be utilized in the production panels, with the former scenario utilized in the mains and submains, as the extra height and a halite roof is desired in the main entries for ventilation and long-term entry stability. Therefore, an additional analysis was conducted to determine the overall full-width cut depth that could be mined before the anhydrite became unstable and fell during mining of the ore under the anhydrite. The cost of the additional bolting would likely be offset by gains in productivity realized by eliminating anhydrite cutting and hauling time.

A FLAC3D simulation of this scenario was performed with the assumption that the anhydrite is bolted after each 20 ft advance (three cuts the width of the miner head). The evaluation was performed with three 20-ft-deep cuts for both 1.5- and 0.5-ft-thick anhydrite layers in the roof.

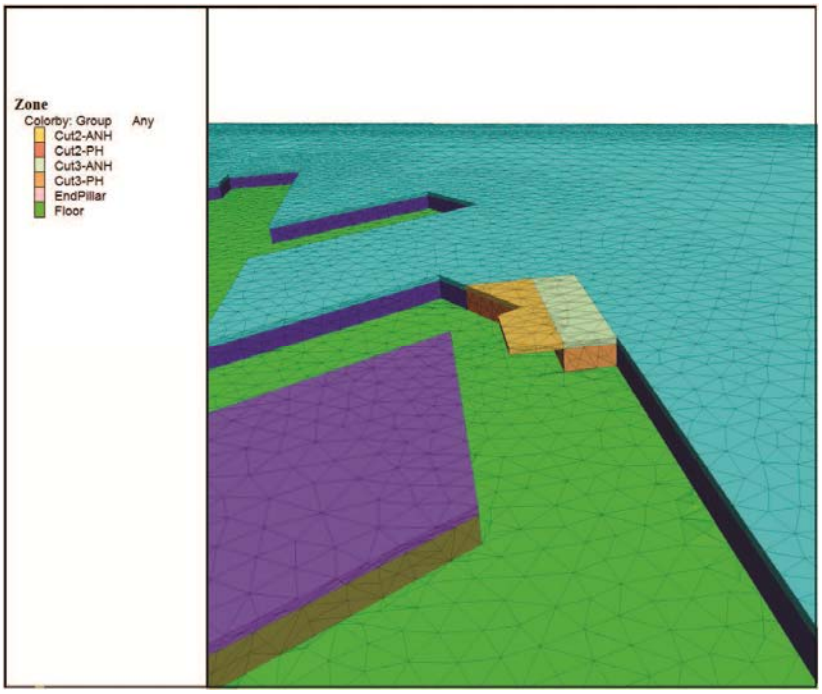


Figure 16-15. Polyhalite Mined in Pass 2: Anhydrite Removed from Pass 1

The results indicate that that the anhydrite is likely to stand long enough to allow three 15-ft cuts, three passes wide, to be taken before it is bolted. To provide for discontinuities, the cut depth would be 15 ft from the last row of roof bolts, or an average depth of cut of 13.5 ft from the face.

Shaft Design—FLAC3D was used to study long-term displacement of the halite portions of the ground through which the main mine shaft passes. The main shaft is located at the center-south of the mine and is proposed to have a finished diameter of 25 ft. It is anticipated the portions of the shaft whose rock walls are composed of halite will experience increased deformation due to creep over the operational life of the mine. To mitigate this movement, portions of the shaft excavated through halite are proposed to have a 6-inch-thick expanded polyethylene foam board and 4-inch-thick corrugated metal sheeting installed between the shaft walls and the 15-inch-thick reinforced concrete lining. The polyethylene and corrugated metal is designed as a soft inclusion that can deform without transferring large loads to the concrete lining. The numerical modeling analysis assessed the magnitude of deformation that the shaft walls may experience due to both halite creep and deformation arising from mining activity near the shaft.

A FLAC3D model was developed to simulate the shaft and surrounding mine workings. A sump area below the ore zone was included, extending 100 ft into the mine floor. The 60% extraction plan was used as the basis for the nearby production panels. To account for the worst case of floor lithology, a 48-ft-thick halite member was assumed immediately below the polyhalite. Halite was also assumed for 100 ft above the polyhalite. Anhydrite was assumed both above and below the halite.

The maximum deflection of the ground was observed to be 1.6 inches at 10 years, 1.9 inches at 20 years, 2.2 inches at 30 years, 2.5 inches at 40 years, and 2.7 inches at

50 years. The results indicate that maximum predicted shaft closure is unlikely to impart large stresses to the concrete lining, as the 10 inches of crushable lining has sufficient space to absorb this amount of movement without building significant stress that would be transmitted to the 15-inch reinforced concrete inner liner.

Subsidence—To model surface subsidence, the FLAC3D shaft model was expanded to simulate deformation between the mining horizon and the surface. This subsidence model was simulated in three stages: first, material stresses were equilibrated, followed by excavation of all underground workings, followed by simulation of 50 years of halite creep. Because the 60% extraction plan represents a rigid pillar approach, no pillar failure is expected, and no caving or significant subsidence is expected at the surface. Some surface movement is expected due to elastic deformation of the mining horizon and creep of the halite.

The largest ground movements are immediately above the production panels at mine level. The strata movement gradually decreases higher into the overburden until it reaches a minimum value in the form of surface subsidence. Over time, the creep of the halite layer induces additional strata deformation and surface subsidence.

Model results showing vertical overburden displacement 10 years after mining predicted maximum surface subsidence of 0.8 inches. With time, the peak surface subsidence increases 1.0 inch (20 years), 1.1 inches (30 years), 1.15 inches (40 years), and 1.2 inches (50 years). This level of surface subsidence dispersed over this time span is unlikely to be visually perceptible or to cause structural damage to buildings or surface infrastructure.

16.2.2 Mine Projections

The primary mains (North Mains) are mined to the north from the shaft and slope bottom area. East or West Mains are driven off the North Mains at various intervals to form mining districts. The East and West Mains are spaced about 2 miles apart and divide the mine into distinct mining districts. Production panels are driven off the East and West Mains in a north-south direction. In some limited areas, production panels are driven east-west.

Once mining has been completed within a district and any reusable materials have been recovered, the district can be sealed off from the active portion of the mine, saving ventilation and examination costs.

Entry widths and pillar sizes were evaluated using the physical properties of the strata according to the results of laboratory tests and geotechnical analyses. The dimensions of mains, submains, panel entries, and crosscut centerlines were based primarily on geotechnical analysis with consideration for ventilation requirements, productivity, and equipment operating constraints.

Main entries and crosscuts are mined 23 ft wide, which is two cutting passes with an 11.5-ft-wide continuous miner cutter head. Main entries and crosscuts are mined on 100-ft by 100-ft centerlines. This width provides good productivity and minimizes convergence over time, reducing the frequency of grading entries to maintain ventilation air flow and equipment clearances. All main entries and crosscuts will be mined a minimum of 6 ft high. Mains barrier pillars are a nominal 400 ft wide (centerline distance) on each side of a set of entries. Where double mains are used, the center barrier between the entry sets is 200 ft wide (centerline distance). Long-life main entries will be developed using a selective mining approach to minimize the impacts of OSD and to provide stable, long-term roof conditions. Figures 16-16 and 16-17 illustrate this selective mining concept.

Five-entry production panel entries and crosscuts inby the panel neck are developed to a 32 ft width, which is comprised of three passes with an 11.5-ft-wide continuous miner cutter head. Entries and crosscuts are developed on 100-ft by 100-ft centers. To accommodate the heavy-duty continuous miners needed to cut the polyhalite ore, the minimum required mining height is 62 inches (5.2 ft).

In the production panels, the anhydrite roof will be bolted in place, on cycle. The continuous miner will place-change after each cut that is three passes wide and 13.5 ft deep. Where the ore bed height is lower than the minimum mining height, the extra height required for equipment clearances will be mined from the floor, as it typically has a higher residual ore grade than does the roof. Early installation of roof support will be essential to control peeling of the anhydrite below the mud seam parting. Figure 16-18 illustrates this process.

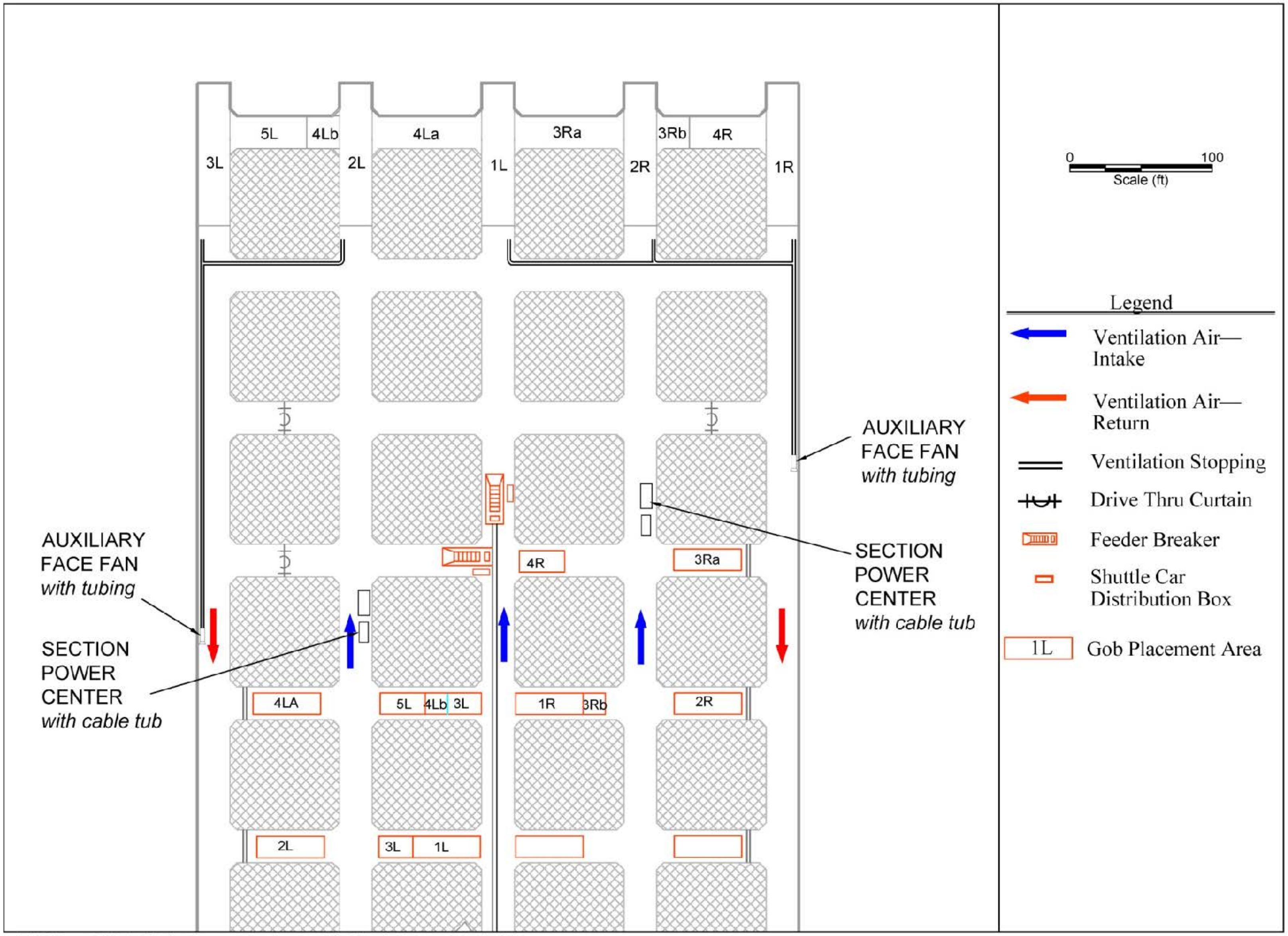
Rooms and room crosscuts are also developed 32 ft wide, which requires three passes with an 11.5-ft-wide continuous miner cutter head. These entries are mined off both sides of the production panel entries as the equipment mines its way out of the panel. Room entries are driven at 60° angles to the production panel entries, approximately 300 ft deep off each side of the production panels as the equipment retreats out of the panel. Room entries are connected by room crosscuts. All room crosscuts that are used as haulageways are bolted on cycle. Figure 16-19 illustrates double mains, panel necks, and production panel entries and rooms, along with the double (dual) split ventilation scenario.

16.2.3 Cuttability

Polyhalite is an evaporite rock, and evaporite rocks are well known to be difficult to cut mechanically, even at low compressive and/or tensile strengths, because they have high “fracture toughness.” “Fracture toughness” is defined as a rock’s resistance to fracturing and the propagation of pre-existing cracks (Joy 2013a).

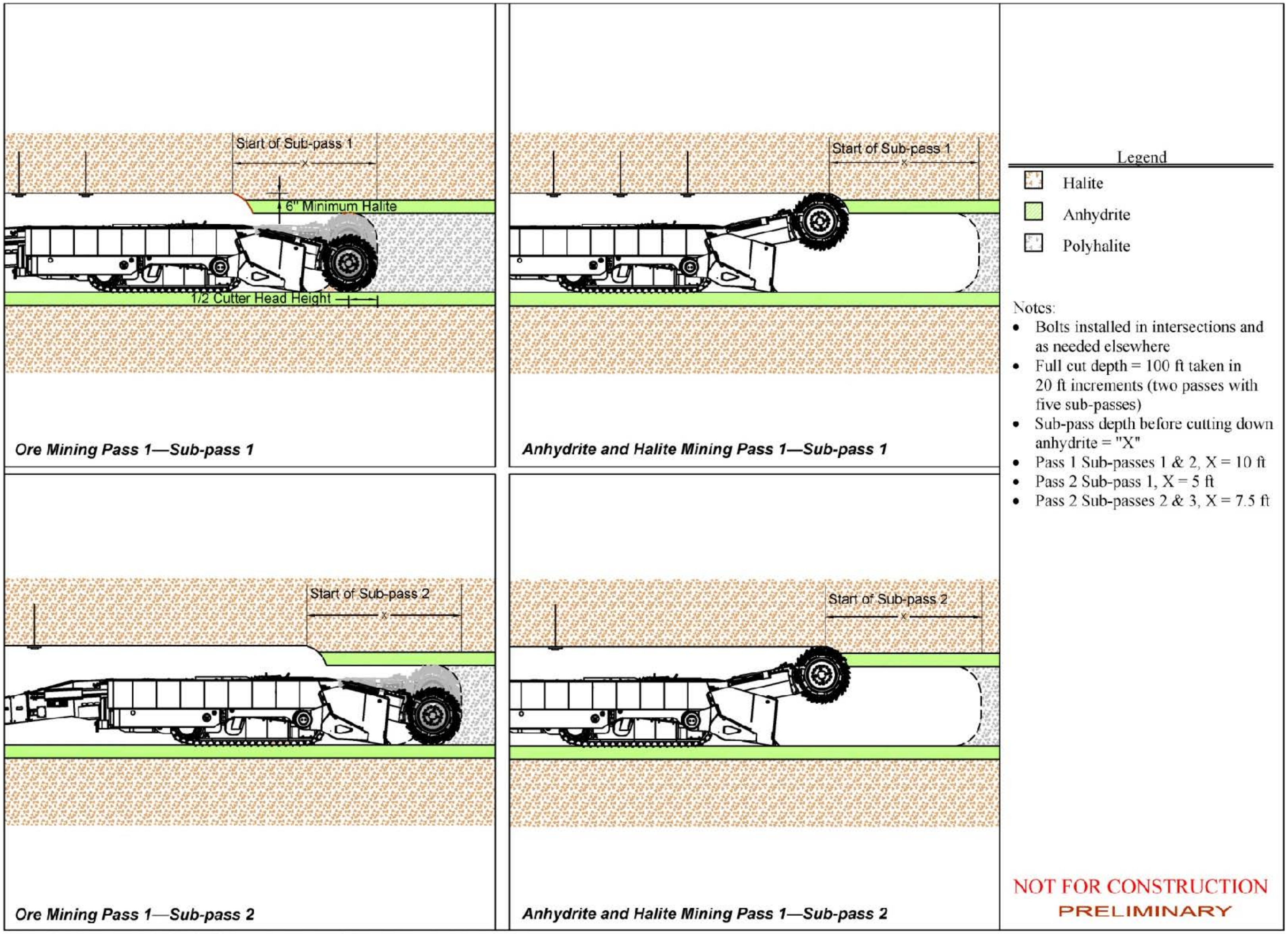
Good correlations have not been reported between UCS values and rock cuttability for evaporites such as potash, anhydrite (gypsum), trona, and salts, although they are sedimentary rocks. Because the UCS values of polyhalite ore within the 50-year mine boundary range are upwards of 15,000 psi and the ore is hard, brittle, and generally fine-grained, the cuttability of the ore with a drum-type continuous miner is a major factor in determining the productivity rate of the equipment. This in turn determines the number of production machines, the necessary support infrastructure, ore haulage requirements, and mining personnel required to meet the annual ROM ore tonnage target. Therefore, determining a feasible mining rate is of paramount importance to the economics of the project. To ascertain the cuttability of the Ochoa polyhalite ore, a series of tests were conducted on ore core samples. The number and type of tests, laboratories, and respective reports and drill holes are listed below:

- J Factor, W Factor, and scleroscope (Sc or shore hardness) tests (19), Joy’s testing lab, Franklin, Pennsylvania, August 30, 2012, drill holes ICP-046 and ICP-053 (no related UCS testing)
- Linear cutting tests (two), Colorado School of Mines (CSM), Department of Mining Engineering, Earth Mechanics Institute (EMI), Golden, Colorado, June 12, 2013, drill holes ICP-083, ICP-088, and ICP-090 (15,089-psi UCS, specific hole location not known but believed to be ICP-083)



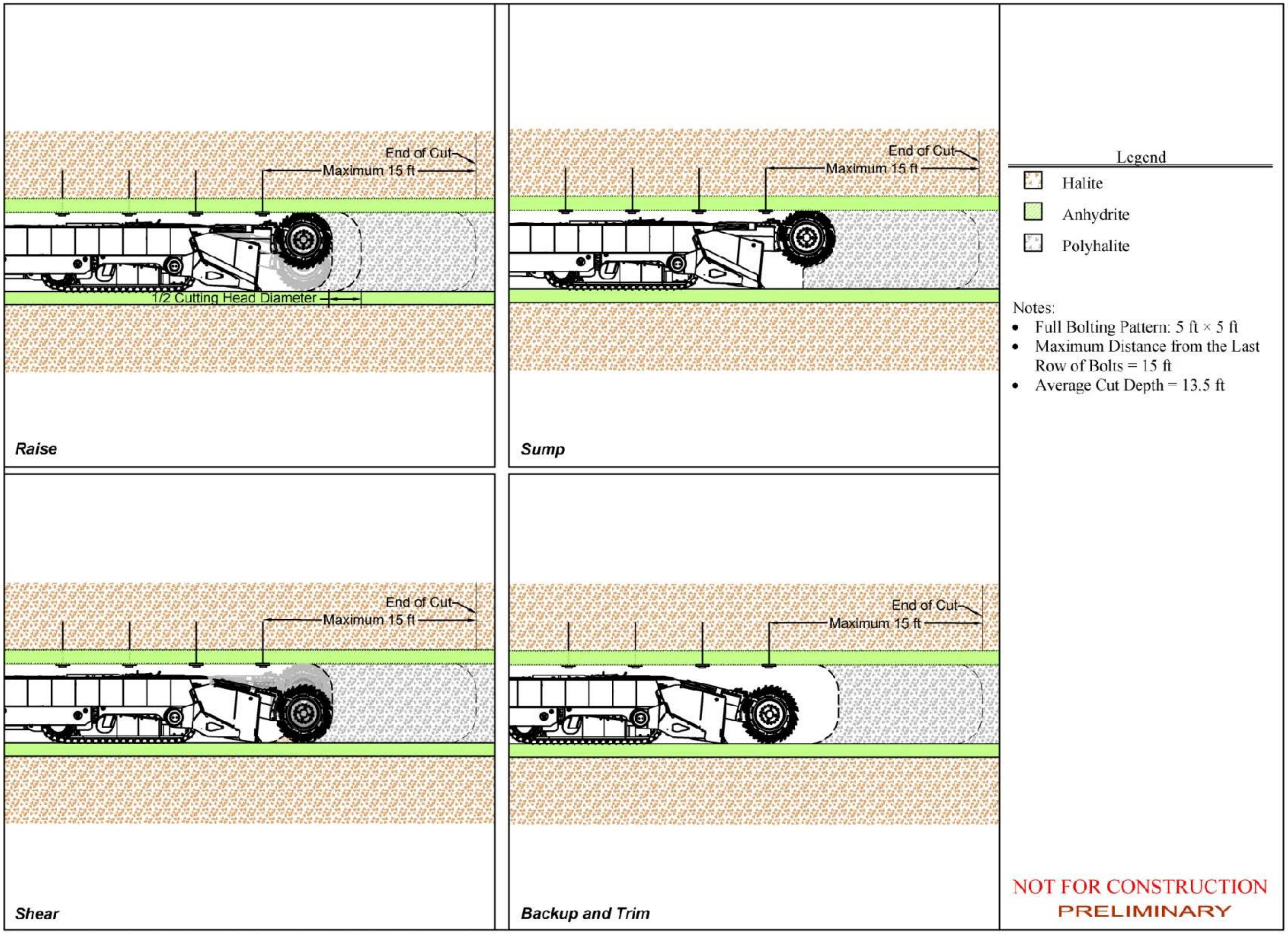
800-01 IC Potash [800-01 ICP_Cut Sequence.dwg: Layout: DSS Mains Gob]:sbp/smvf (8-13-2013)

Figure 16-16. Mains Development—Anhydrite Gob Placement



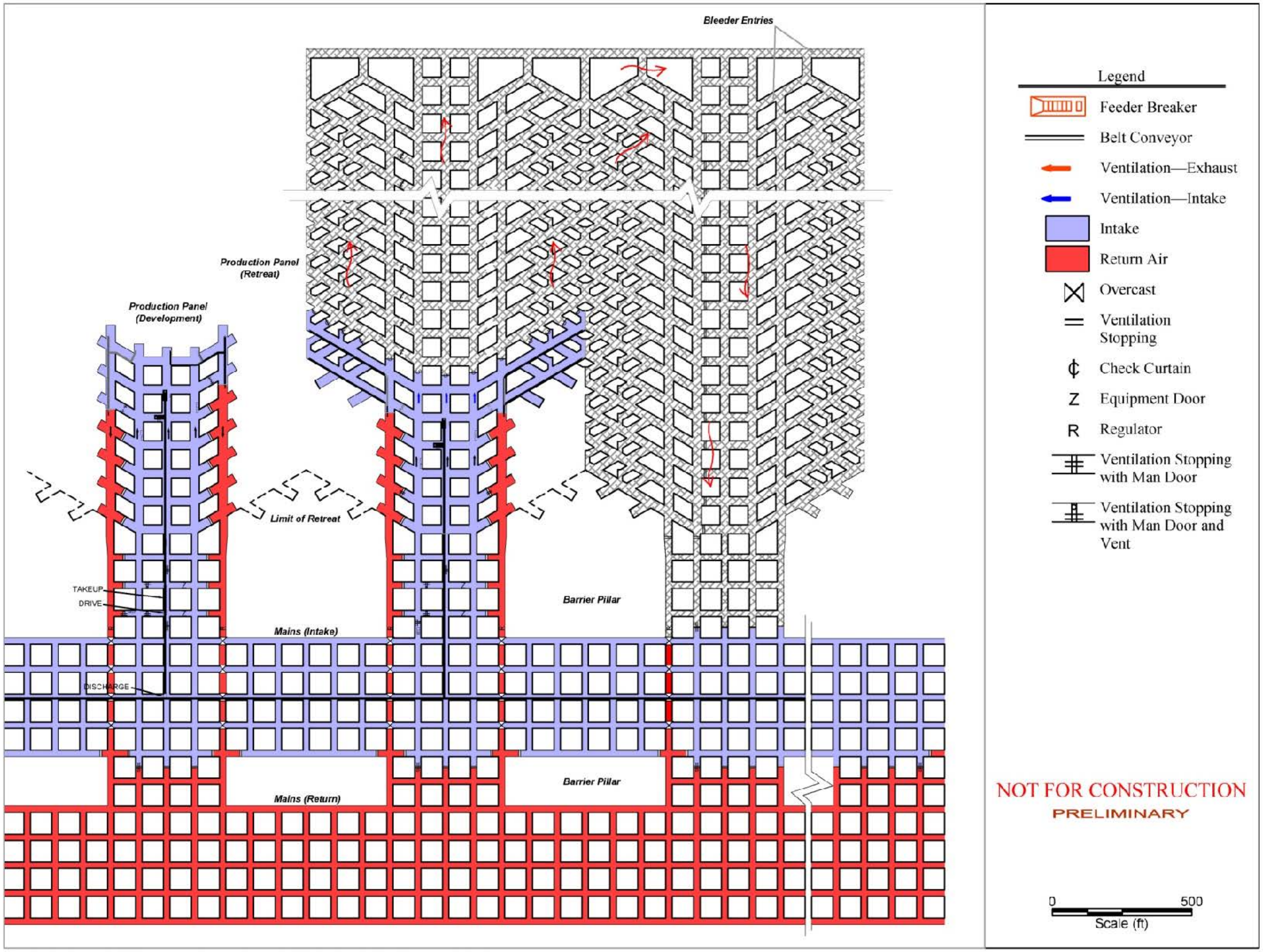
800-01 IC Potash [800-01 ICP_CM Cut Cycle_TR.dwg Layout: Selective Mining].smvf (1-17-2013)

Figure 16-17. Section View of Sub-Pass Showing Selective Mining of Anhydrite



800-01 IC Potash [800-01 ICP_CM Cut Cycle_TR.dwg; Layout: 44inch_Panels]:smvf (1-17-2013)

Figure 16-18. Sump Cycle with 44-inch Cutter Drum



800-01 IC Potash [800-01 ICP_Typical Panel Neck_SS_Ventilcation_TR.dwg]:smvf (1-17-2014)

Figure 16-19. Typical Mine Projections for Production Panels and Double Set of Mains

- J Factor, W Factor, and scleroscope (Sc or shore hardness) tests (20), Joy's testing lab, Franklin, Pennsylvania, June 12, 2013, drill holes ICP-083, ICP-088, or ICP-090 (15,089-psi UCS, unknown hole location)
- *Polyhalite Core Sampling Testing Results*, Joy (2012a) report dated September 13, 2013

To ensure that the testing reflected the more severe cutting conditions expected at the Ochoa Mine, the core samples for the EMI (2013) tests and the Joy (2013b) tests were selected from the denser, finer-grained portions of the ore bed. These areas are generally in the upper one-third to two-thirds of the ore zone. The exact location within the ore bed zone of the core samples taken for the Joy (2012b) tests is unknown.

Table 16-20 provides the results of the two series of J, W, and scleroscope (Sc) factor testing conducted on Ochoa ore samples.

Table 16-20. J Factor, W Factor, and Sc Test Results

Test Date	J Factor (average)	W Factor (average)	Scleroscope (Sc) (average)
August 30, 2012	158.6	0.006	30
June 12, 2013	84.3	0.003	41
Note: J Factor is similar to known polyhalite values.			

The J Factor is an indication of cuttability. The lower the J Factor, the more difficult it is to cut the material relative to materials with higher J Factors. The test results from August 2012 predict that the Ochoa polyhalite ore will be "very difficult" to cut (Joy 2012a). Test results from June 2013, which represent the higher limits of the UCS range for the ore, indicate the ore is "extremely difficult" to cut (Joy Global 2012a and 2013b). Joy, however, has confirmed that it has equipment that could mine the Ochoa ore (Joy 2012a).

The W Factor is a measure of the abrasiveness of the material being mined. The higher the W Factor, the shorter the expected bit and drum life will be, due to wear. W Factors over 0.003 indicate higher than average wear. The polyhalite tested in August 2012 was classified as moderately abrasive. The polyhalite tested in June 2013 was classified as not very abrasive. The average of the two tests is 0.004, indicating a moderate abrasiveness, which may mean accelerated wear to the bits, bit blocks, cutter drums, loading CLAs, and conveyor chain and deck. Joy has encountered material with readings of 0.020 and higher (Joy 2012b).

The Sc tests are an indication of softness. Results indicate that the Ochoa polyhalite is slightly softer than other polyhalite, indicating that it can be cut with a continuous miner.

Table 16-21 compares results of the tests discussed above of Ochoa polyhalite with salt, potash, trona, and gypsum.

Based on the first round of testing in August 2012, Joy recommended that ICP have the EMI's linear cutting machine (LCM) perform linear cutting tests to better define the cuttability of the Ochoa ore. Six-inch core samples were collected and sent to the CSM lab for preparation and testing. Two linear cutting samples were prepared from the cores and tested (EMI 2013). The linear cutting samples were named ICP-01 and ICP-02. The LCM sample ICP-01 was taken from drill hole ICP-083, with core from depths of 1,407.5 to 1,411.1 ft bgs. LCM sample ICP-02

Table 16-21. Comparison of Ochoa Polyhalite to Other Materials

Material	J Factor Range	J Factor Average	W Factor (average)	Scleroscope (Sc) (average)
Ochoa Polyhalite	84 to 188	121	0.004	36
Other Polyhalite	—	higher	lower	47
Salt	200 to 795	421	—	—
Potash	280 to 1,100	495	—	—
Trona	350 to 380	614	—	—
Gypsum	200 to 1,400	686	—	—

Source: Joy Global.

was taken from two core holes: core from ICP-088 was taken from depths of 1,509.5 to 1,511.5 ft bgs, and core from ICP-090 was taken from depths of 1,446.4 to 1,449.1 ft bgs. Figure 16-20 shows the core sections for LCM test samples ICP-01 and ICP-02.

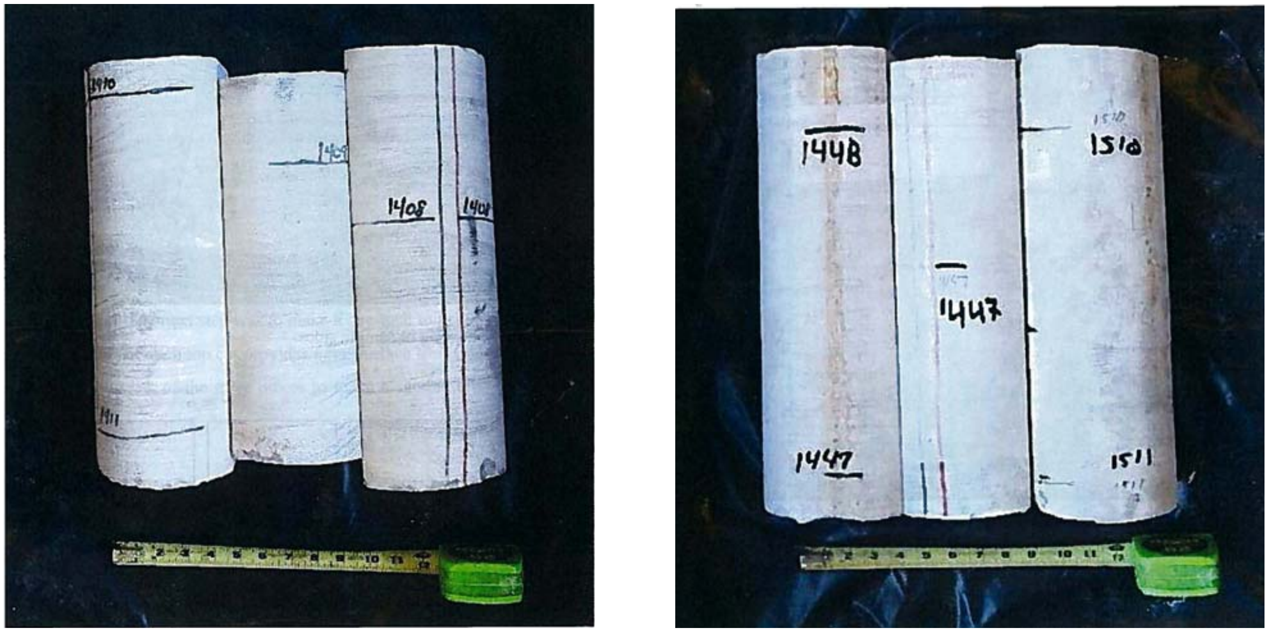
Linear cutting test variables that changed during testing were the spacing between the cuts, representing the spacing between the bits on a continuous miner drum, and the depth of the cut, assumed to be the distance the bit is able to penetrate into the solid material being mined. The variables held constant during testing were cutting speed, bit attack angle, bit type, and rock type (polyhalite). The actual bit type recommended for production continuous miners were Kennametal U92HDL5 or equivalent. Table 16-22 shows the bit spacing and depth for the linear cutting tests.

Table 16-22. LCM Bit Spacing and Penetration

Spacing (inches)	Penetration (inches) 0.3	0.5	0.6	0.7	0.9
2.25	X		X		X
3.00		X		X	

The cutting test runs confirmed the conchoidal fracturing nature of the brittle Ochoa polyhalite (Figure 16-21). The smaller 2.25-inch spacing produced a relatively smooth pattern with no coring (ridges). Table 16-23 shows the summary test results for the cutting test runs.

The test results were used to develop the expected preliminary instantaneous cutting rate (ICR) for a continuous miner operating in the tested material. ICR is defined as the volume of material cut per unit of time. For the FS, it is expressed in tons of polyhalite per cutting hour (tph). The instantaneous cutting rate determined from the CSM linear cutting tests is 422 tph of polyhalite at an *in situ* density of 173 pounds per cubic foot (pcf). This ICR does not take into account any machine availability or utilization and is not the mining rate. Those parameters are incorporated in the productivity analysis, and the mining rate is determined by developing an elemental sump cycle using the ICR as the basis. The CSM linear cutting tests showed preliminarily that cutting polyhalite with a drum-type continuous miner is feasible with bit spacing between 2.25 inches and 3 inches.



a) ICP-01 b) ICP-02

Figure 16-20. Samples Prior to Casting (EMI 2013)

Figure 16-21. Example of Conchoidal Fragments from Cutting Test Run

Table 16-23. Average Linear Cutting Results for Polyhalite (from EMI 2013)

Spacing (inches)	Pene-tration (inches)	Normal Average (lbf)	Normal Max. (lbf)	Drag Average (lbf)	Drag Max. (lbf)	Side Average (lbf)	Side Max. (lbf)	Cutting Coefficient (ratio)	Specific Energy (hp-hr/yd^3)
2¼	0.3	3,926	11,522	1,533	5,569	1,800	5,398	0.39	4.5
2¼	0.6	4,147	13,560	2,089	8,620	2,381	7,262	0.50	3.0
2¼	0.9	4,677	13,581	2,574	9,120	3,612	10,373	0.55	2.5
3	0.5	3,854	16,261	2,028	8,993	537	4,021	0.53	2.7
3	0.7	4,983	19,352	2,563	11,859	2,116	8,582	0.51	2.4

16.2.4 Productivity Analysis

Mine facility design requirements are dependent on the mining unit's projected productivity rate.[7] The productivity rate for a given series of cuts[8] can be reasonably estimated from the section layout, equipment characteristics, and time-study data from similar mining methods. To maintain average unit shift production on a sustained long-term basis, a continuous miner section (unit) must have the capability to produce two to three times the unit shift average (Suboleski 2007; Douglas 1980).[9]

Numerous section equipment configurations are available for continuous miner room-and-pillar mining sections. The arrangement chosen for the FS mine design is a combination of three DSSS and one single-split section.

To determine the theoretical unit shift productivity rate and equipment pairing for development and retreat mining, productivity simulation analyses were conducted using a Microsoft Excel™ spreadsheet developed by Dr. Stanley C. Suboleski, P.E. (Suboleski 2007). This spreadsheet is designed for room-and-pillar mining methods. This deterministic model estimates section productivity from the tons per cut and the cut cycle times under a specified cut sequence using inputs of material characteristics, cut parameters, machinery characteristics for both mining and haulage equipment, shift timing parameters, distances for haulage to the dump point, and tram distances between cuts. Output from the spreadsheet model includes the expected maximum production rate, linear advance rate, and cycle times for the input parameters. This model predicts the capability of the system using the mining equipment's loading and hauling capacities. It does not account for equipment or staffing availability, other managerial factors, or mining conditions, although rating factors can be applied to account for some mining conditions.

Cut sequences to simulate mining were developed for 5-entry mains and 5-entry production panel and room layouts. The ore is overlain by an anhydrite layer 6 to 18 inches thick that will be removed along with an additional 6 inches of halite when mining in the mains. The anhydrite will be bolted in place when mining in the production panels. Figure 16-22 is an example of a typical cut sequence for a 5-entry main setup as a DSSS, where only the intersections are bolted.

Table 16-24 shows the operating shift parameter assumptions used for the simulation. Shift schedules are portal-to-portal or collar-to-collar. Table 16-25 shows the equipment parameters. Table 16-26 provides the continuous miner cut cycle analysis developed using the ICR. The model output for mains and panel productivity analysis is summarized in Table 16-27. Results include the estimated tons per shift productivity for the cut cycle. These tons per shift represent the theoretical optimal productivity possible when operating under the assumptions of the input parameters and in ideal conditions. Average productivity will be less by a factor of at least two.

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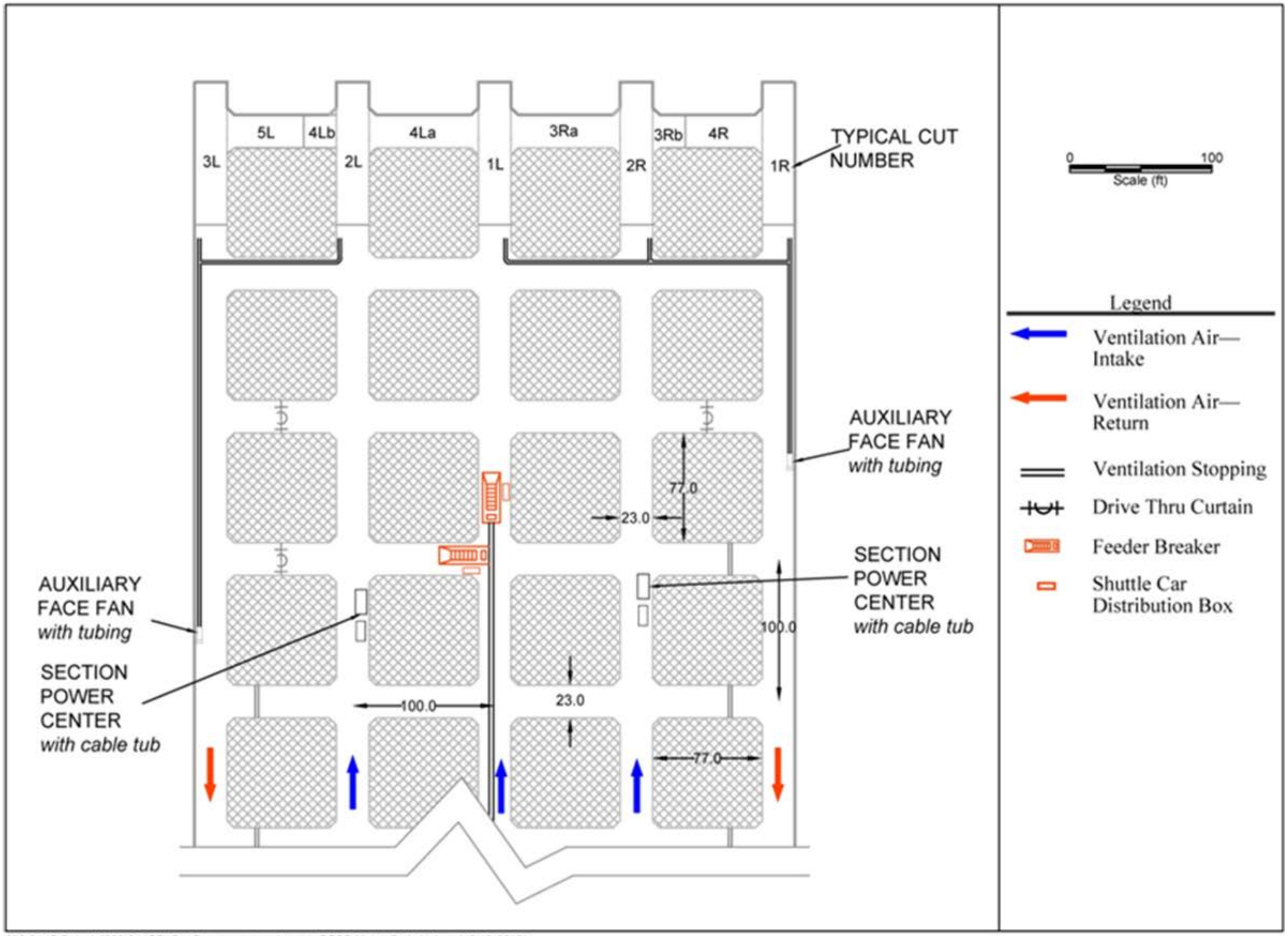
[7] Productivity rates are commonly expressed in tons per (scheduled) unit shift or feet of entry advance (equivalent single entry). For the FS, productivity is expressed as tons per unit shift.
[8] A "cut" is defined as the block of ore the full width of the entry by the depth of cut that, when mined, place-changing the continuous miner is required.
[9] Confidential operations audits for mining companies.



800-01 IC Potash [800-01 ICP_Cut Sequence.dwg; Layout: DSSS Mains Cut] sbp/smvf (9-13-2013)

Figure 16-22. Mains Development DSSS Cut Sequence

Table 16-24. Shift Parameters

Parameter	Value	Units	Notes
Number of cuts	Varies	cuts	Number of cuts being analyzed, usually belt move to belt move
Shift duration	720	minutes	Shift duration is portal-to-portal shift time
Travel of persons (in + out)	30	minutes	Travel time from the mine dry to the working section, both directions, hot seat change-out
Prepare to start	20	minutes	Safety talk, etc.
Lunch	0	minutes	Float lunch
Service miner	45	minutes	Service equipment
Delays	75	minutes	Necessary and unnecessary delays

Table 16-25. Equipment Parameters

Equipment Characteristics	Value	Units	Notes
Shuttle car capacity	15.5	t	Joy 10SC32AB shuttle car capacity 18 t, 15.5-t average load
Loading rate (standard)	5.4	tpm	Sump analysis, cut cycle 123.1 minutes
Loading rate (cleanup)	5.4	tpm	Sump analysis, cut cycle 123.1 minutes
Car speed (change-out to face)	440	fpm	
Car speed (change-out to feeder-breaker)	440	fpm	
Car speed (feeder-breaker to change-out)	440	fpm	
Dump time of car	0.5	minutes	30 seconds to dump load
Miner length	36.3	feet	
Shuttle car length	28.5	feet	
Miner tram speed	30	feet	
Switch-in time	0.5	minutes	Switch-in time is turning hauler at the feeder-breaker and prior to change-out point (articulated hauler only)
Switch-out time	0	minutes	Switch-out is turning an articulated haulage vehicle at the change-out point

Note: fpm = frequency per minute, tpm = tons per minute

Table 16-26. Miner Cut Cycle Analysis

Cycle	Rate (inches/sec)	Distance (inches)	Time (seconds)	Percent (%)	Material Cut (t)	Rate* (tpm)
Sump	0.29	29.64	104.0	87.3%	12.1	7.0
Shear	0.57	2.72	4.8	4.0%	0.6	7.0
Backup	3.00	29.64	9.9	8.3%	0.0	0.0
Raise	6.00	2.72	0.5	0.4%	0.0	0.0
Summation			119.1		12.6	6.4

Note: Cutting drum diameter is 59.28 inches (includes 4-inch bits at 75° attack angle). Sump depth is one-half the drum diameter.

* Based on CSM linear cutting test.

Table 16-27. Productivity Analysis Summary

Sequence	Entry Centers (ft)	Number of Entries	Number of Cuts	550-minute/720-minute Mining/Shift: Maximum Polyhalite Productivity (t/shift)	Average Polyhalite Productivity (t/shift)	Anhydrite Gobbed (t/shift)	Average Anhydrite Gobbed (t/shift)
Gob Anhydrite (cut depth 20 ft with sub-cuts)							
Mains	100	5	9	1,674	835	504	252
Bolt Anhydrite (cut depth limited to 13.5 ft)							
Panel development with place change	100	3	40	1,915	955		
Panel retreat with place change	100	2	14	1,985	990		

Notes: Based on a sump cycle time of 123.1 seconds.
Articulated haulers are used in the mains (gob handling).
Shuttle cars are used in the panels.
Standard shift = 400 minutes, mining of 600-minute shift.
Hot seat change-out increases mining time by 30 minutes.
Hot seat change-out and float lunch increases mining time by 60 minutes.
It takes 36 minutes to bolt 13.5 ft deep in a 32-ft-wide entry.
Panel shuttle car change point kept at 100 ft.

The anhydrite gobbed per shift is calculated by estimating the tonnage of anhydrite (gob) at an average thickness of 1.5 ft over the cut cycle area. The gob tonnage is divided by the shifts required to mine the cut cycle. To operate the number of required units to meet the annual tonnage targets and account for the belt moves, panel-to-panel moves, mine maintenance delays, regulatory inspection interruptions and other major delays, an additional unit above the theoretical number of units is required to be installed and operated with the other units on a rotating basis. The estimated number of mining units required to produce 3.7 Mtpy is seven continuous miner units set up as three DSSS and one single section.

16.3 Production Scheduling, Mine Modeling, and Ore Grade Control

Polyhalite ore production is on a 7-day per week, two 12-hour shifts per day schedule, with one 8-hour overlap shift per day on a 5-day per week schedule for utility and maintenance work. The 8-hour shift has crews that overlap from one day to the next so that all 7 days in the week have utility and maintenance coverage. This schedule is typical of current practice at the Carlsbad potash mines. Crews work four 12-hour days the first week, and three 12-hour days the second week, averaging 84 hours for 2 weeks.

With the production shifts using “hot seat change-out,” a window is created between one set of production shift crews and the other shift’s production crews for the utility and maintenance crews to perform limited daily scheduled utility and maintenance activities. This window is approximately 2.5 hours long. For sections developing the main entries where gob has to be loaded out each day, this 2.5-hour window is used for that activity, as no ore can be loaded on the belt conveyors during that time.

Each DSSS or single section is scheduled to produce the ore equivalent of 279 days a year out of a scheduled 351 mine operating days a year, requiring each production section to be available for production 79% of its available time.

Mine planning models for production scheduling and ore grade determination were developed using Carlson Mining 2013's *Underground Mining Module* (Carlson) (Carlson 2013). Carlson, historically known as SurvCADD™, is the predominant mine planning software used by US underground mine operators in bedded seam deposits, including coal, trona, and potash, and is well-suited for mine planning of the Ochoa Project. It uses a gridded seam model instead of a block model.

Detailed mine projections were developed in AutoCAD 2013™ (Autodesk Inc. 2013) based on the Ochoa polyhalite resource model grids discussed in Item 14. The resource model grids describe true bed thickness, elevation, depth of cover, dip, and the following quality parameters: polyhalite grade, equivalent potassium sulfate (K_2SO_4), anhydrite grade, halite grade, and magnesite grade.

The mine projection layout is limited to Measured and Indicated Resources in accordance with the definition of Mineral Reserves under CIMDS/NI 43-101. Mining projections are additionally constrained by Property boundary and gas and oil well barrier pillars.

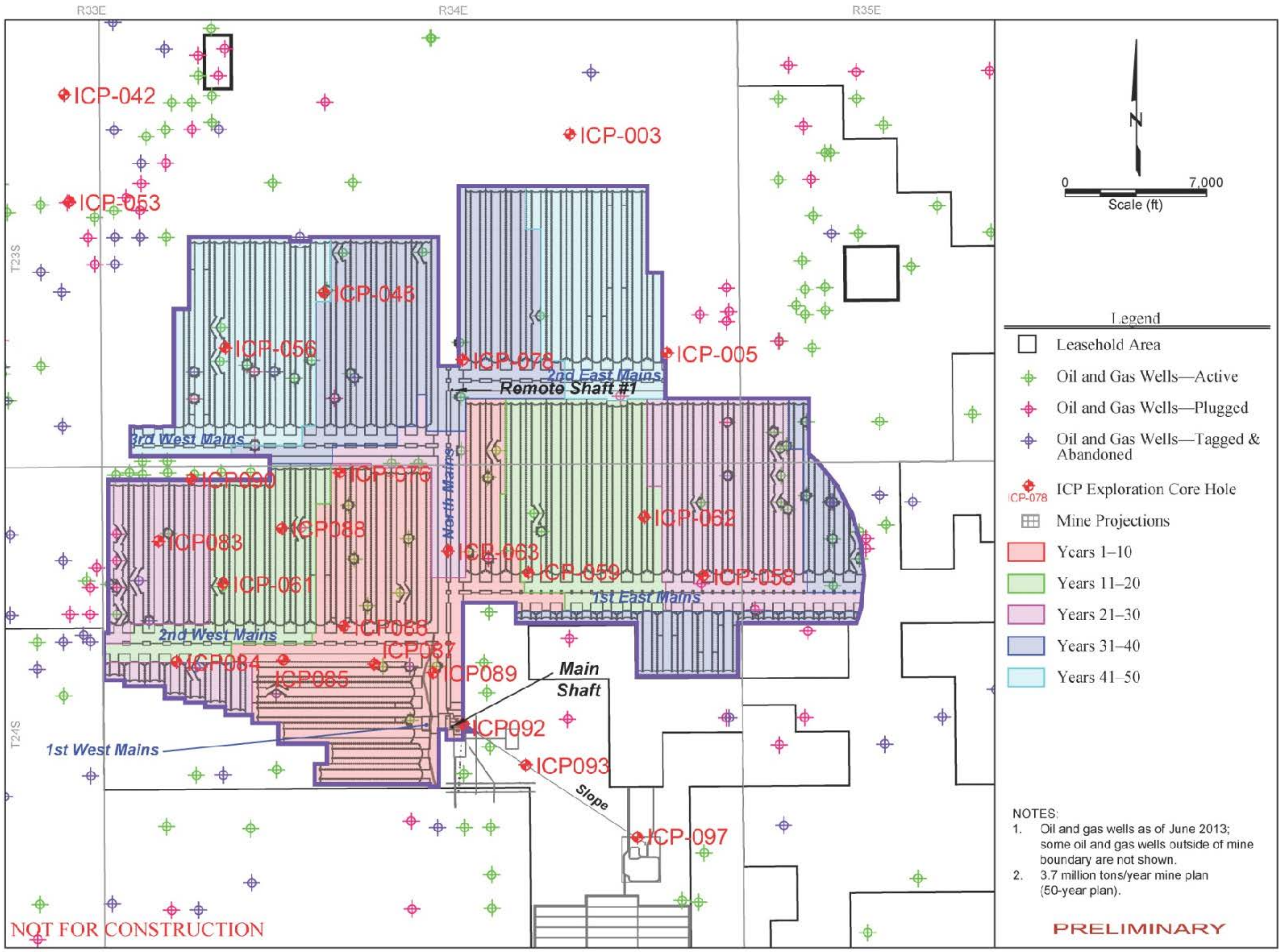
Mine scheduling (timing) is developed in Carlson based on bed volumetrics, productivity rates for each continuous miner section or super section, work schedules, and recovery factors for advance and retreat mining. Each continuous miner section is scheduled by month, quarter, or year for the life of the mine, with the two former timing segments used for the initial few years of operation. Mine production starts in July 2016 and runs through 2065. Figure 16-23 illustrates the 50-Year Mine Plan subdivided into 10-year increments.

Production data from the Carlson model is compiled and post-processed in a Microsoft Excel™ spreadsheet. OSD is calculated in the mine modeling volumetrics and is sourced from the resource model grids. It is calculated on a minimum mining height of 5.2 ft plus 6 inches of OSD. Two inches of roof and 4 inches of floor dilution are allocated for OSD on a mine-wide basis. Where the ore bed height is less than 5.2 ft, the additional height needed for equipment clearances is mined from the floor. The final ROM ore head grade and mined tons are calculated as the mixture of rock mined in-seam and out-of-seam.

The Carlson software generates production tons and timing maps based on the geologic model grids and mine projections. For the FS, the Carlson models are configured to provide annual production tons and ore grade parameters as defined in the geologic models. Results are summarized for economic analysis. Table 16-28 summarizes the mine modeling production and quality results by year. Figures 16-24 and 16-25 graphically illustrate the tons and polyhalite ROM ore grade by year.

Ore grade control may be challenging at the Ochoa Mine for several reasons. The polyhalite does not always exhibit a clean contact between the ore and the anhydrite or other floor material. The polyhalite grades downward into the anhydrite and floor. In addition, the floor has areas of mudstone close to the polyhalite, and the heavy mining equipment may tear up the floor material which then has to be loaded out as the ore is mined.

Mining the ore will generate significant dust which must be combated with water sprays and the exhausting face ventilation system. Dust and water spray will create difficulty for the continuous miner operator, who will be some 25 to 30 ft outby the cutter head, to see the face and ribs near the cutter head.



800-01 IC Potash [800-01 ICP_Mine Projection Maps_TR.dwg: Layout: Mine Projections 10 Yr Increments]:smvf (12-20-2013)

Figure 16-23. 50-Year Mine Plan in 10-Year Increments

Table 16-28. Annual Polyhalite Ore Production and Quality Summary

Mine Plan Year	Production (t)	Bed Thickness (ft)	Mined Thickness (ft)	Grade (%)	Equivalent K_2SO_4 (%)	Anhydrite (%)	Halite (%)	Magnesite (%)
2016	377,697	5.24	5.74	83.38	24.09	10.61	3.22	5.71
2017	3,247,915	5.22	5.72	78.71	22.74	13.49	3.61	6.51
2018	3,749,846	5.18	5.68	75.92	21.93	16.35	4.17	7.03
2019	3,655,576	5.20	5.70	78.10	22.56	13.74	4.04	6.99
2020	3,733,779	5.17	5.67	77.27	22.32	16.65	4.05	6.90
2021	3,871,664	5.20	5.70	73.24	21.16	17.24	4.33	9.09
2022	3,589,576	5.21	5.71	78.34	22.63	14.80	4.20	6.81
2023	3,723,839	5.22	5.72	77.23	22.31	11.01	3.89	8.00
2024	3,679,910	5.29	5.79	76.23	22.02	11.06	4.04	8.59
2025	3,678,954	5.27	5.77	76.83	22.20	12.70	3.97	8.37
2026	3,811,788	5.23	5.73	76.05	21.97	11.40	3.94	8.88
2027	3,778,849	5.41	5.91	77.68	22.44	10.01	3.85	8.35
2028	3,825,235	5.29	5.79	78.36	22.64	10.15	3.51	8.19
2029	3,582,173	5.38	5.88	78.85	22.78	10.52	3.43	7.92
2030	3,810,703	5.34	5.84	79.29	22.91	10.05	3.28	7.69
2031	3,689,640	5.53	6.03	78.90	22.80	10.28	3.22	7.75
2032	3,716,090	5.48	5.98	79.46	22.96	9.93	3.28	7.56
2033	3,662,464	5.52	6.02	80.01	23.12	10.23	3.22	7.07
2034	3,762,705	5.56	6.06	79.92	23.09	10.37	3.03	6.97
2035	3,821,148	5.55	6.05	78.91	22.80	10.65	2.97	7.52
2036	3,685,926	5.51	6.01	78.45	22.66	11.09	2.89	7.47
2037	3,822,409	5.53	6.03	78.79	22.76	10.62	3.41	7.79
2038	3,755,743	5.62	6.12	80.62	23.29	9.94	3.22	6.82
2039	3,823,171	5.57	6.07	81.93	23.67	9.82	3.33	6.28
2040	3,675,390	5.53	6.03	82.19	23.75	9.77	3.58	6.12
2041	3,915,148	5.62	6.12	81.01	23.40	10.01	3.37	7.00
2042	3,571,953	5.42	5.92	77.41	22.36	10.58	3.94	8.15
2043	3,521,775	5.36	5.86	76.57	22.12	12.20	4.22	8.04
2044	3,612,237	5.32	5.82	76.00	21.96	11.48	3.93	8.83
2045	3,515,052	5.34	5.84	77.46	22.38	11.34	3.71	8.40
2046	3,679,814	5.34	5.84	75.61	21.84	10.75	3.54	9.49
2047	3,750,132	5.23	5.73	75.74	21.88	9.97	4.04	9.81
2048	3,455,728	5.25	5.75	74.98	21.66	12.85	3.99	9.86
2049	3,486,829	5.23	5.73	74.93	21.65	10.65	3.85	10.15
2050	3,728,815	5.24	5.74	74.87	21.63	10.80	3.92	10.03
2051	3,558,311	5.22	5.72	75.82	21.91	10.66	3.72	9.53
2052	3,805,017	5.27	5.77	76.04	21.97	12.57	3.78	9.82
2053	3,892,248	5.27	5.77	76.56	22.12	11.29	3.74	9.42
2054	3,706,266	5.32	5.82	76.98	22.24	10.43	3.69	8.99
2055	3,781,651	5.29	5.79	77.19	22.30	10.25	3.75	8.88
2056	3,865,193	5.33	5.83	77.11	22.28	12.59	3.76	9.31
2057	3,772,379	5.33	5.83	77.62	22.42	11.21	3.68	8.93
2058	3,773,883	5.34	5.84	78.16	22.58	11.11	3.74	8.45
2059	3,689,768	5.51	6.01	81.29	23.48	9.46	3.25	6.83
2060	3,726,614	5.39	5.89	80.35	23.21	10.21	3.35	7.36
2061	3,692,711	5.39	5.89	81.41	23.52	9.66	3.37	6.83
2062	3,830,491	5.40	5.90	78.71	22.74	12.09	3.76	8.28
2063	3,860,593	5.35	5.85	78.72	22.74	12.21	3.84	8.32
2064	3,786,599	5.49	5.99	81.84	23.64	9.69	3.47	6.68
2065	3,825,541	5.35	5.85	79.44	22.95	12.57	3.62	7.96
Average	3,723,153	5.36	5.86	78.05	22.55	11.39	3.66	8.08
Total 50-Year Mine Plan Production (rounded up to next 100,000 t)								182,400.00

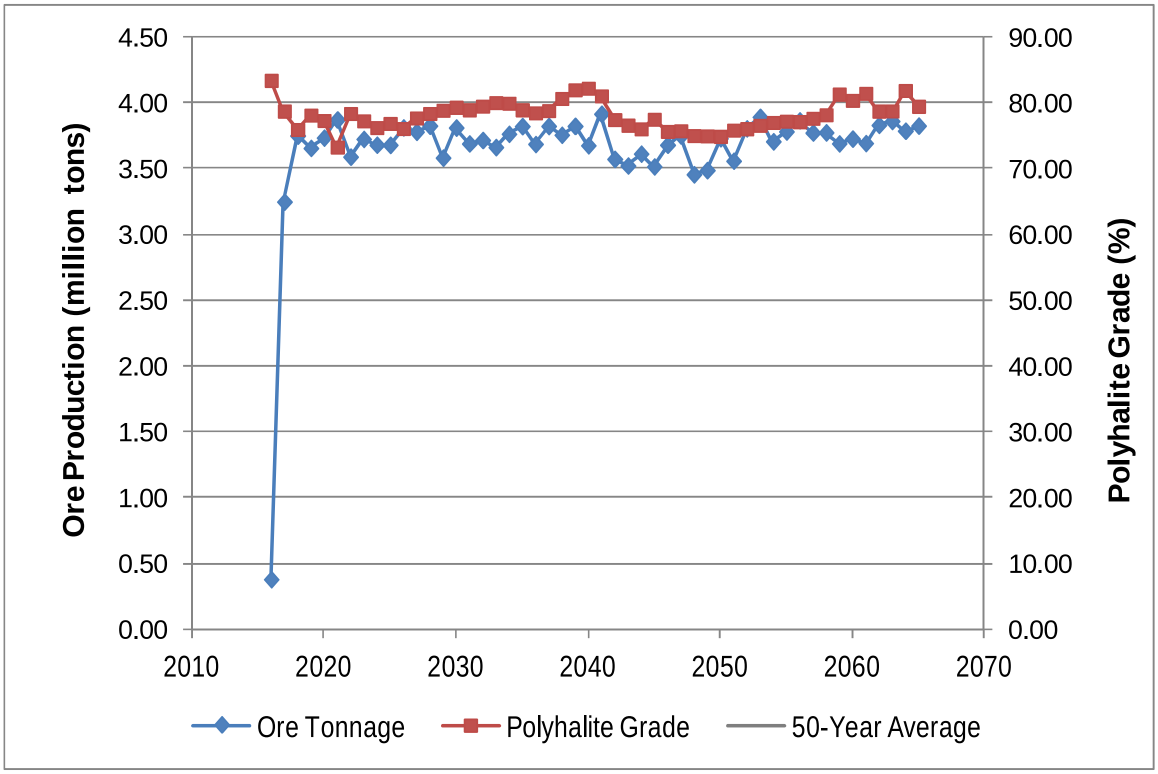


Figure 16-24. Annual Tonnage and Polyhalite

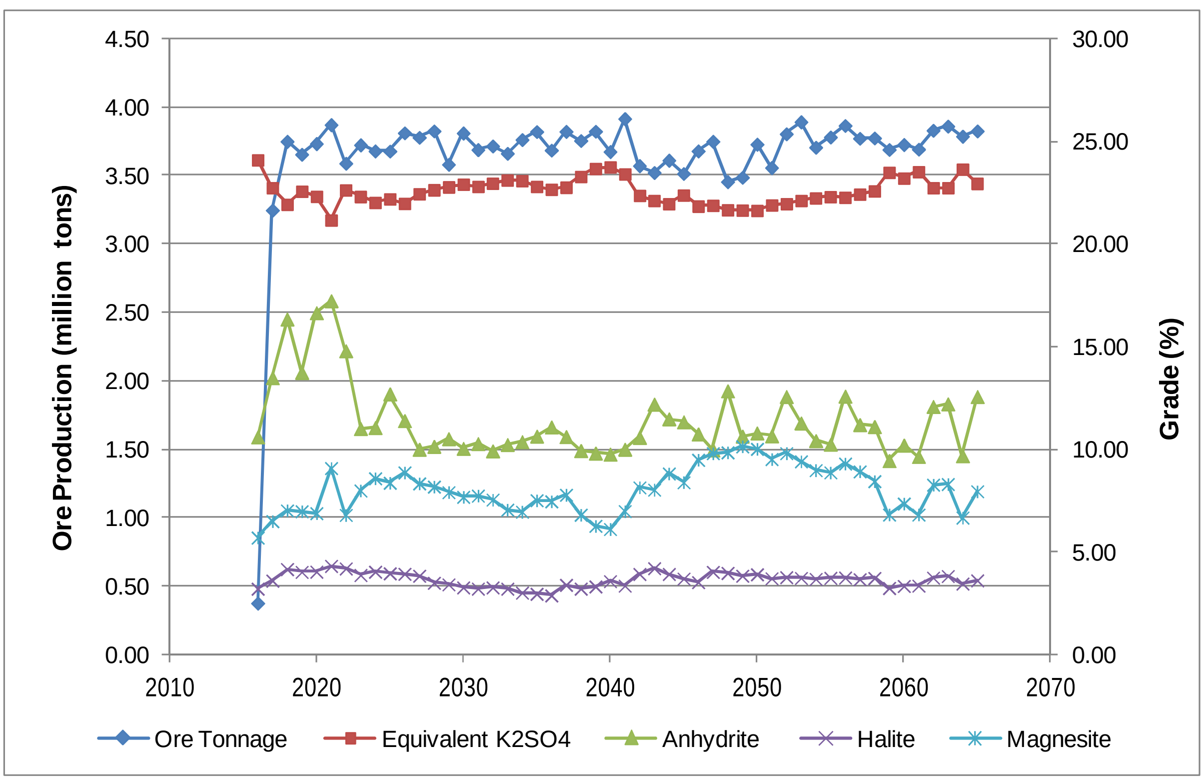


Figure 16-25. Annual Tonnage and Quality Parameters

Where the ore bed thickness (height) is less than the minimum mining height, the floor will be mined to provide the necessary clearance. Figure 15-5 shows the projected minimum theoretical additional floor thickness that must be cut for equipment clearance. This thickness is in addition to the 4 inches of floor OSD that is allocated mine-wide. The roof and floor materials were assayed and included in the mine models. This OSD effectively increases the minimum mining height to 5.7 ft.

Another potential detriment to ore grade quality is the portion of the anhydrite roof layer that lies between the top of the polyhalite and the thin mud seam parting. For panels where roof bolting will be used to minimize any anhydrite roof material being mined with the ore, geotechnical modeling indicates that a cut depth up to 13.5 ft, or 15 ft from the last row of roof bolts, should stand for sufficient time to prevent the anhydrite from falling prior to it being bolted, provided the roof is bolted shortly after the cut is mined. Fractures or other discontinuities may allow the anhydrite to fall during the ore mining phase of the cut. The thickness of the anhydrite between the mud seam and the mined roof will impact the standup time.

For ore grade control in the main entries, where extra height is desired for machinery clearances and long-term convergence, the geotechnical modeling provides the indicated standup time and the cutting depths as illustrated in the productivity analysis required for OSD control. Actual mining experience will be needed to ascertain the validity of these guidelines.

16.4 Mining Equipment and Supporting Infrastructure

Each underground mining production DSSS' equipment consists of:

- Two drum-type continuous miners (2,300 V)
- Four shuttle cars (950 V), or
- Four articulated haulers (battery or diesel)
- One section scoop tractor (low-seam load haul dump [LHD], battery or diesel)
- One section fork lift (battery or diesel)
- Two dual-boom roof bolters (950 V)
- Two section power centers (12.47 kV by 2,300, 950, 480, 220, and 110 V)
- One section switchhouse (12.47 kV)
- Two auxiliary face fans (480 V)

Each underground mining production single section's equipment consists of:

- One drum-type continuous miner (2,300 V)
- Three shuttle cars (950 V), or
- Three articulated haulers (battery or diesel)
- One section scoop tractor (low-seam LHD, battery or diesel)
- One section fork lift (battery or diesel)
- One dual-boom roof bolter (950 V)
- One section power center (12.47 kV by 2,300, 950, 480, 220, and 110 V)
- One section switchhouses (12.47 kV)
- Two auxiliary face fans (480 V)

All electrical- and diesel-powered equipment used in or inby the last open crosscut or in return air will be Mine Safety & Health Administration (MSHA) approved permissible.

Various types of underground stationary and mobile equipment will be provided for supply materials transport, underground belt conveyor ore haulage, equipment transport, mine dust suppression and firefighting, electrical high-voltage distribution, communications and monitoring, and maintenance. Typical underground support equipment includes but is not limited to:

- Outby scoop tractors (low-seam LHD, diesel)
- Supply tractors with grading and lifting attachments (diesel)
- Personnel vehicles (battery and diesel)
- Maintenance vehicles (diesel)
- Supply trailers
- Specialty trailers, such as belt structure material carriers, pipe trailers, high-voltage table tubs
- 60-inch slope and main line belt conveyors for ore haulage
- 42-inch production panel belt conveyors for ore haulage
- Belt conveyor fire detection systems
- Miner personnel tracking system
- Mine monitoring and control system (atmospheric and equipment)
- Communications systems
- Mobile diesel-powered generators for moving self-propelled electrically powered equipment
- Firefighting equipment
- High-voltage distributing equipment (12.47 kV)
- Mine firefighting and dust suppression pipelines (6-inch mains and 4-inch panels)

16.5 Gas and Oil Wells

The Ochoa Project area is an active production area for gas and oil, and there are numerous active gas and oil wells within the mine plan area. The polyhalite bed to be mined is located approximately 1,300-ft to 1,635-ft bgs in the 50-Year Mine Plan boundary, while the gas and oil formations (and wells) are reported to be between 5,000-ft and 16,000-ft bgs. The polyhalite ore zone in the mine area is approximately flat-lying and is from 4.5 to 6.5 ft thick. The minimum mining height is currently planned to be 5.2 ft. Newer wells are reported to have employed horizontal drilling techniques, and the horizontal well bores are expected to extend from 4,000 ft to a 2-mile radius from the vertical well in the future. Hydraulic fracturing of the horizontal well bores is reportedly practiced in the area. Gas well reservoirs in the basin have bottom-hole pressures up to 6,000 psi (Angelo 2013).

Historically, co-located gas and oil well operations and potash mining have been a contentious issue. ICP has endeavored to form working relationships with the area's gas and oil companies in hopes of changing the two industries' relationship to one of mutual cooperation. To that end, ICP has signed MOUs with several local gas and oil lease holders and conducts meetings with gas and oil producers on a continuing basis.

Safely mining near gas and oil wells has been a much studied topic for over a century. As the knowledge base of experience and technology has developed over time, numerous regulatory bodies have enacted codes to permit resource conservation while maintaining the overarching objective of mine worker safety. From a regulatory viewpoint, miner safety has always been the preeminent goal of any regulation promulgated.

A common approach to safety is the use of “barrier analysis” to place one or more preventive measures (“barriers”) between the hazard and the person or asset being protected. This approach is applicable to designing and operating mines near gas and oil wells. The creation of redundant or multiple measures and warning systems reduce the exposure risk considerably over single measures.

Typical “barriers” to protect the safety of the miners are:

- Mine design and operational procedures for gassy mining conditions
- Adequate mine ventilation system design
- Methane monitoring systems
- Well protection pillars
- MSHA mine safety regulations for gassy mines
- MSHA plug and mine-through standards, used when mining is near or through a well (101[c] petitions)
- For new wells, well designs with multiple casing strings to allow venting of any migrating gas from below the ore zone

It has long been understood by the coal mining industry that no barrier pillar of any practical size would prevent migration of gasses from a corrupted well bore into the mine workings. Gassy coal mines are, by their nature, considerably more likely to create hazardous conditions for the miners than potash, polyhalite, and trona mines. If coal mines can safely work adjacent to gas and oil wells, it stands to reason that methods are available to accomplish safe co-tenancy with polyhalite mines, especially because the ore is non-combustible.

The existing Carlsbad potash mines operate under MSHA’s (2013a) Title 30, *Code of Federal Regulations*, Part 57.22003 (30 CFR 57.22003) for metal/non-metal regulations for ventilation Category IV mines, i.e., non-combustible ore and no detectable methane. ICP proposes to design and operate the Ochoa Mine under Part 57.22003 regulations for ventilation Category III mines, i.e., mines in which non-combustible ore is extracted and which liberate a concentration of methane, or is capable of forming explosive mixtures with air, or have the potential to do so based on the history of the mine or the geological area in which the mine is located. Based on the known geologic setting for the project area, the only plausible avenues for methane to enter the mine workings would be associated with gas or oil wells, fault planes, or geologic collapse features that are sufficiently deep enough for hydrocarbons to be forced upward through them to the ore bed and surrounding strata.

Adequate mine ventilation is a critical component to preventing the accumulation of potentially harmful quantities of methane within the mine atmosphere. Other regulatory (and prudent) lines of defense are standards for workplace examinations for regular monitoring of gas concentrations, the elimination of ignition sources (“permissible” or “intrinsically safe” electrical equipment requirements, no smoking underground, and controlled cutting and welding procedures), and methane monitoring systems.

There are no set methodologies to calculate sizes for well protection pillars. Several methods have been used by various groups over the years, both empirical and numerical. Empirical geotechnical analysis for sizing well protection pillars for trona and potash have traditionally focused on angle-of-draw[10] relationships and potential theoretical shear plane

[10] The angle subtended between a point on the surface directly above the edge of mine workings and the farthest point of ground disturbance attributable to mining.

failure of well casings due to mine subsidence effects on ground movement (Sandia National Laboratories 2009; Craig 1998). In 1957, however, Pennsylvania conducted a study (Commonwealth of Pennsylvania 1957) which concluded that well failure was occurring near the mine horizon due to inadequate pillar strength and not from being within the angle of draw. This study has recently been revisited with similar conclusions.

On the basis of the 1957 study, Pennsylvania adopted regulations for well protection pillars. The maximum pillar size is 200 feet square, or 40,000 square feet (ft^2), regardless of depth. Larger pillars may be split into smaller pillars and still maintain the total equivalent pillar support area. In 1997, Pennsylvania changed its code to require approval prior to mining within 150 ft of a well (this is the same distance MSHA requires).

In 2003, Peng et al. (2003) conducted a study that reviewed historical gas well pillar failures and used 3D finite-element numerical modeling techniques to better identify the factors that affect gas well stability. Peng used the finite-element code ABAQUS (Berg 2010) to numerically simulate the mechanics of mining-induced ground deformation previously predicted by his empirical Comprehensive Integrated Subsidence Prediction Model (CISPM) (Peng and Luo 1992) which generated only surface subsidence profiles. Efforts were made by Peng to improve the representation of actual field conditions in the model and to account for rock mass effects (Voight and Pariseau 1970).

The Peng et al. (2003) analysis found that 90% of the gas well failures occurred within the coal seam or 34 ft above and below the coal seam being mined, and that the angle of draw concept did not apply to gas well failure. Of the 10% of failures that were located beyond 34 ft of the roof and floor, the maximum distance was 34 ft in the roof and 100 ft in the floor.

For ICP, there are two primary concerns regarding subsidence:

- The possibility of gas and oil extraction-related disturbance and subsidence by various gas wells below the ICP polyhalite mining horizon disturbing the planned overlying ICP polyhalite-extraction operations
- The potential for gas production-induced ground subsidence opening fluid pathways up the well bores (external to the casing strings), providing gas and/or fluids access to the polyhalite ore body and into mine workings

Although some analysis tying well protection pillar size to preventing gas migration into mine workings has been attempted, the coal industry, gas and oil industry, and coal mine regulatory agencies have long recognized that it is not a practical approach.

Potential pathways for hydrocarbon migration to the Ochoa polyhalite ore bed are primarily via geologic features or gas and oil well bores. Geologic features would include fault planes, major jointing running continuously for thousands of feet vertically, or collapse features extending from the ore zone to the gas reservoir horizons. With the presence of the salt zones between the gas and oil reservoir horizons and the polyhalite ore bed, it is unlikely that major joints would provide a continuous pathway from the gas and oil reservoirs to the ore zone. There do not appear to be any major fault traces identified in the area designated for the Ochoa Project. In light of these findings, the most frequently occurring pathway for hydrocarbons to reach the mine workings would be through gas and oil wells.

Although MSHA and individual states regulate coal mines differently than metal/non-metal mines, coal mine regulatory and industry practices regarding gas and oil wells are applicable to the Ochoa Mine.

For the FS, a 200-ft-radius well protection pillar is planned around each gas or oil well, and the mining extraction ratio was limited to 60%. Individual well pillar analysis using appropriate geotechnical modeling techniques can be applied on a site-specific basis for ongoing operations.

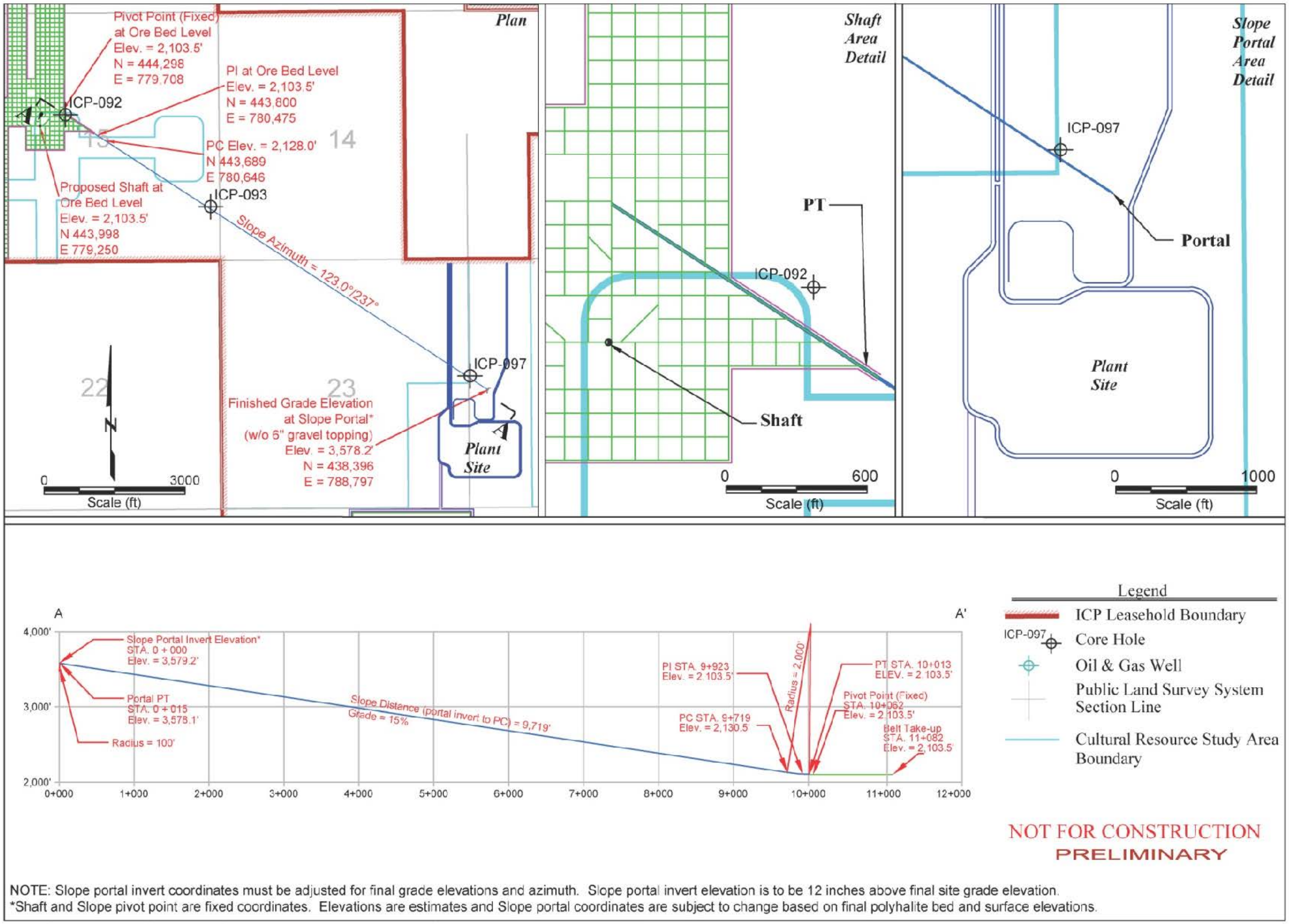
16.6 Mine Access

Mine access is via a 25-ft-diameter, two-compartment, 1,525.5-ft-deep, concrete-lined air shaft and an 8.5° mine slope. The shaft will be used for intake and return air, and the slope will be used for intake air and vehicle travelway, and it will be equipped with a 60-inch-wide belt conveyor for ore haulage to the surface. Figure 16-26 shows the relative locations of the shaft, slope, and plant site and the profile of the slope. The shaft design was based on geotechnical and hydrological data collected from drill hole ICP-092, which is located approximately 880 ft east-northeast of the shaft site and adjacent to the base of the slope. The slope design was based on data from ICP-092 and on data from additional geotechnical testing conducted on cores from drill holes ICP-093 and ICP-097.

The mine's main air shaft has one compartment for return air and one compartment for intake air. The second compartment, for intake air, is equipped with an emergency escape hoist to provide a secondary means of escape for miners. Federal mine safety regulations require two separate means of escape from a mine before more than 20 persons can be underground at any one time. The return air compartment of the shaft will be furnished with two 1,500-hp fans located on the surface and offset by at least 15 ft from the nearest edge of the air shaft. At full mine production, both fans will be in service. The shaft's compartments are separated by a reinforced concrete curtain (divider) wall. Considering the volume of air that will be flowing into the intake side of the shaft, control of groundwater seepage will be important.

A mine slope that is 8.5° from horizontal (also called a "ramp" or a "decline") will be constructed from a location near the plant site to a location near the shaft (Figure 16-26). The slope will serve as the ore haulage corridor and as the mine's personnel and supply/equipment corridor, and it will also provide additional intake air to the mine. The mine slope is approximately 11,082 ft long overall, from the slope portal face to the slope belt conveyor take-up. The slope invert is paved with concrete to permit efficient and safe vehicle travel and easy cleanup of any ore spillage from the 60-inch slope belt conveyor. The design allows for MSHA-required clearances on the "tight" side of the slope conveyor. The slope width is planned for a minimum inside clearance of 21 ft and a minimum vertical clearance at the rib line (spring line) of 10 ft.

The slope design for the FS is a side-by-side, conveyor/travelway (roadway) configuration. The main slope excavation will be by roadheader or continuous miner. Ground support will be conventional resin-anchored rock bolts for the entire slope length and shotcrete in areas of poorer ground conditions. Some ground conditions are likely to require steel sets/arches/lattice girders and lagging for adequate support. Further slope ground support in the form of additional bolting and shotcreting in unshotcreted areas is budgeted on a periodic (8-year) basis in the sustaining capital cost with the center of the slope height set at 13 ft.



800-01 IC Potash [800-01 ICP Mine Plan Slope Plan and Profile_TR.dwg; Layout: Slope Plan and Profile]:smvf (1-17-2013)

Figure 16-26. Shaft and Slope Relative Location and Slope Profile

The shaft and slope bottom area will be developed in the ore bed using the mine's continuous miner fleet. Development will commence from the slope bottom using a continuous miner section ventilated with a 5-ft-diameter exhausting ventilation tube located in the slope. This temporary ventilation tubing is connected to temporary fans which are located on the surface at the slope portal. In addition to the surface fans, three in-line MSHA-approved permissible fans will be installed in the tubing downslope to provide a sufficient quantity of air for mining. Ore haulage will be up the slope using the permanent slope belt conveyor.

Figure 16-27 shows the permanent shaft and slope bottom area fully developed, with the underground shop, warehouse, offices, high-voltage switchhouses, equipment diesel fueling areas, main belt conveyor drives, and other facilities. Should the shop and warehouse need additional space, the ore bed can be developed to the south and east of the shaft to expand these facilities.

The mine will have a dedicated 30/40-MVA electrical substation located at the main shaft site. The substation will receive incoming power at 115 kV via an overhead transmission line from the main facility substation and transform it down to 12.47 kV for underground distribution throughout the mine.

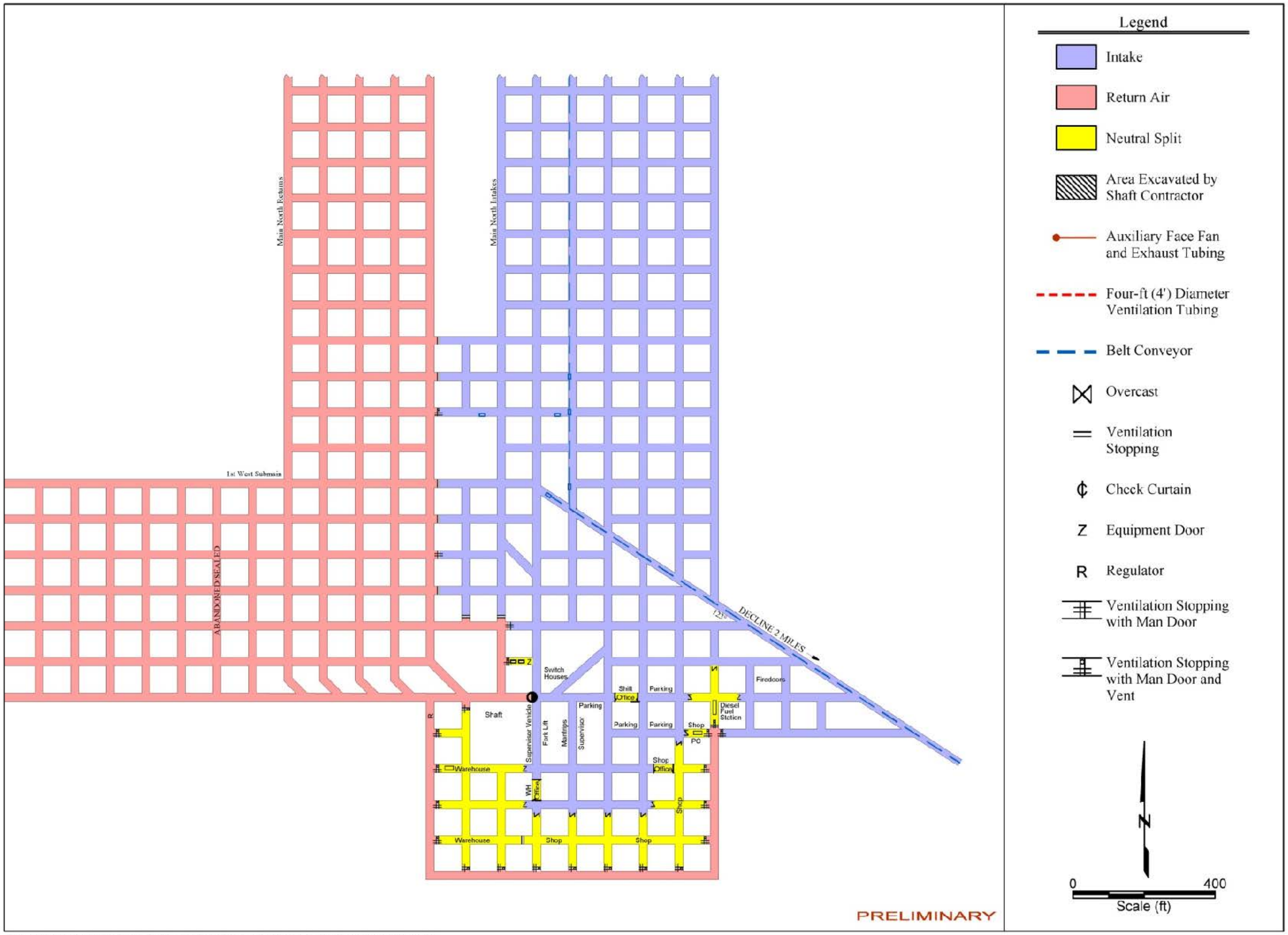
The mine will have a mine-wide monitoring and control system used for control and monitoring of all belt conveyors, pump stations, compressor stations, major electrical installations, ventilation fans, and other crucial support systems. The system will also use appropriate intrinsically safe barriers and sensors to accommodate environmental monitoring for methane, carbon monoxide, hydrogen sulfide (H_2S), air velocity, ventilation pressure, DPM, and other atmospheric conditions (atmospheric monitoring system [AMS] portion). All networked programmable logic controllers (PLC's) and downstream devices are interconnected by means of Ethernet communication over fiber-optic cable.

A two-way radio communication system that can be used on the surface and underground, along with a "tunnel" radio ultra-high frequency (UHF) system with surface antennas and leaky feeder cable underground that can track miners underground will be installed. In addition, a battery-powered, MSHA-permissible phone and cable system will be installed as a separate and isolated backup communication system.

16.7 Mine Ventilation

The main mine ventilation air will be provided by two 1,500-hp fans located on the surface adjacent to the return side of the main air shaft. A mine branch network diagram was developed from the life-of-mine projections. The mine entry centerline projections were imported into Ventsim™ as mine ventilation branches and the system was modeled. Characteristics assigned to branches include the area, perimeter, length, and friction factor. The friction factors that were assigned are typical for room-and-pillar mining in metal/non-metal mines (Prosser and Wallace 1999).

As required by gassy mine regulations, the ventilation has been modeled as separate fresh air splits to each mining section. The target quantity at the last permanent stopping at each section is 35,000 cubic feet per minute (35 kcfm). The belt entry is modeled as intake air. The belt drive power centers and battery charging stations are modeled with separate ventilation splits coursing air directly to the returns, of 3 kcfm and 5 kcfm, respectively. The underground shop and warehouse is on a 50-kcfm separate split of intake air with exhaust



800-01 IC Potash [Prelim Shaft and Development Plan_TR.dwg]:smvf (1-17-2014)

Figure 16-27. Phase 8: Shaft/Slope Bottom Area Development

directly to the return airway. Stoppings, belt box checks, and equipment airlock doors are included around belt drive installations to isolate the belt drive area in the event of a fire and to minimize dust entrainment in the intake air.

The total amount of ventilation airflow required for a mine is based on regulatory requirements governing minimum quantity and air quality (methane, oxygen, carbon dioxide, hydrogen sulfide, and DPM concentration) and other factors, such as the number of production units, belt conveyor power centers, battery charging stations, seals, bleeder entries, air compressor stations, underground shops, and similar requirements. MSHA regulations require that the ventilation system maintain methane levels in the airways to less than 1.0% by volume.

Table 16-29 shows the estimated mine ventilation air requirements for the first 25 years of mining. Ventilation modeling indicates an additional shaft is necessary by the 25th year of mining. The branch diagram with air quantities and two fans operating in parallel indicated for the 25th year, just prior to the commissioning of the new shaft, is shown in Figure 16-28. The quantities required for mining years 26 to 50 are shown in Table16-30. The mine air quantity requirements increase due to additional belt power centers and leakage. Figure 16-29 is a 3D view of the mine ventilation system for the 50-Year Mine Plan.

Table 16-29. Estimates for Mine Air Quantity Requirements for the First 25 Years of the 50-Year Mine Plan

Mine Area	No.	Air Quantity Requirement (cfm)	Total Air Quantity Requirement (cfm)	Comments
Producing sections	7	35,000	245,000	Minimum quantity for design basis
Belt power centers	17	3,000	51,000	Allowance, actual to be determined
Battery charging stations	14	5,000	70,000	Allowance, actual to be determined
Setup/belt reclaim areas	1	45,000	45,000	
Mine shop, warehouse, office	1	50,000	50,000	Allowance, actual to be determined
Subtotal			461,000	
Leakage			389,000	Typical range 40% to 60% for room-and-pillar mine
Total Air Quantity Requirements			850,000	

Table 16-30. Estimates for Mine Air Quantity Requirements for Year 26 through 50 of the 50-Year Mine Plan

Mine Area	No.	Air Quantity Requirement (cfm)	Total Air Quantity Requirement (cfm)	Comments
Producing sections	7	35,000	245,000	Minimum quantity for design basis
Belt power centers	23	3,000	69,000	Allowance, actual to be determined
Battery charging stations	14	5,000	70,000	Allowance, actual to be determined
Setup/belt reclaim areas	1	51,000	51,000	
Mine shop, warehouse, office	1	50,000	50,000	Allowance, actual to be determined
Subtotal			485,000	
Leakage			782,000	Typical range 40% to 60% for room-and-pillar mine
Total Air Quantity Requirements			1,267,000	

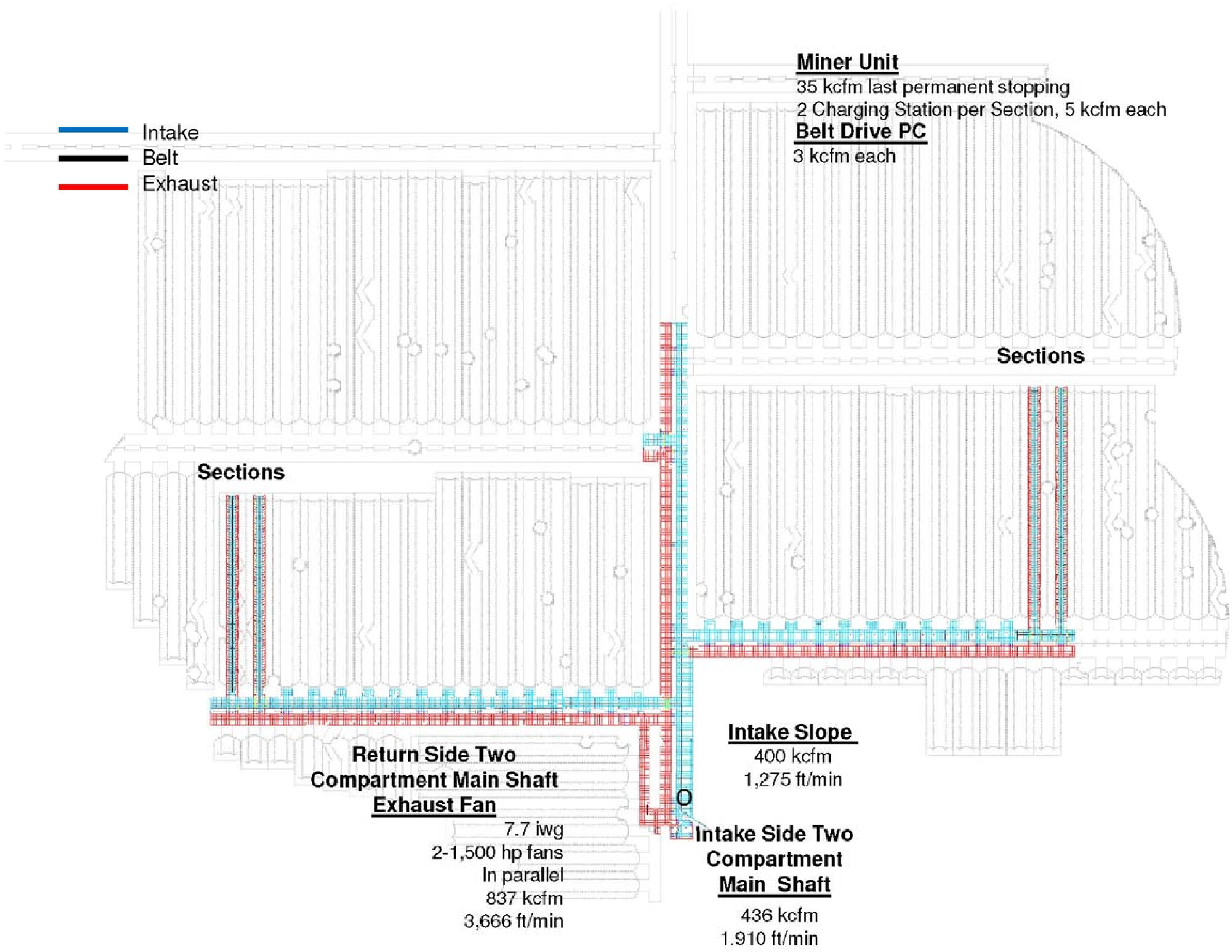


Figure 16-28. Ventilation Model for Mining Years 1 to 25

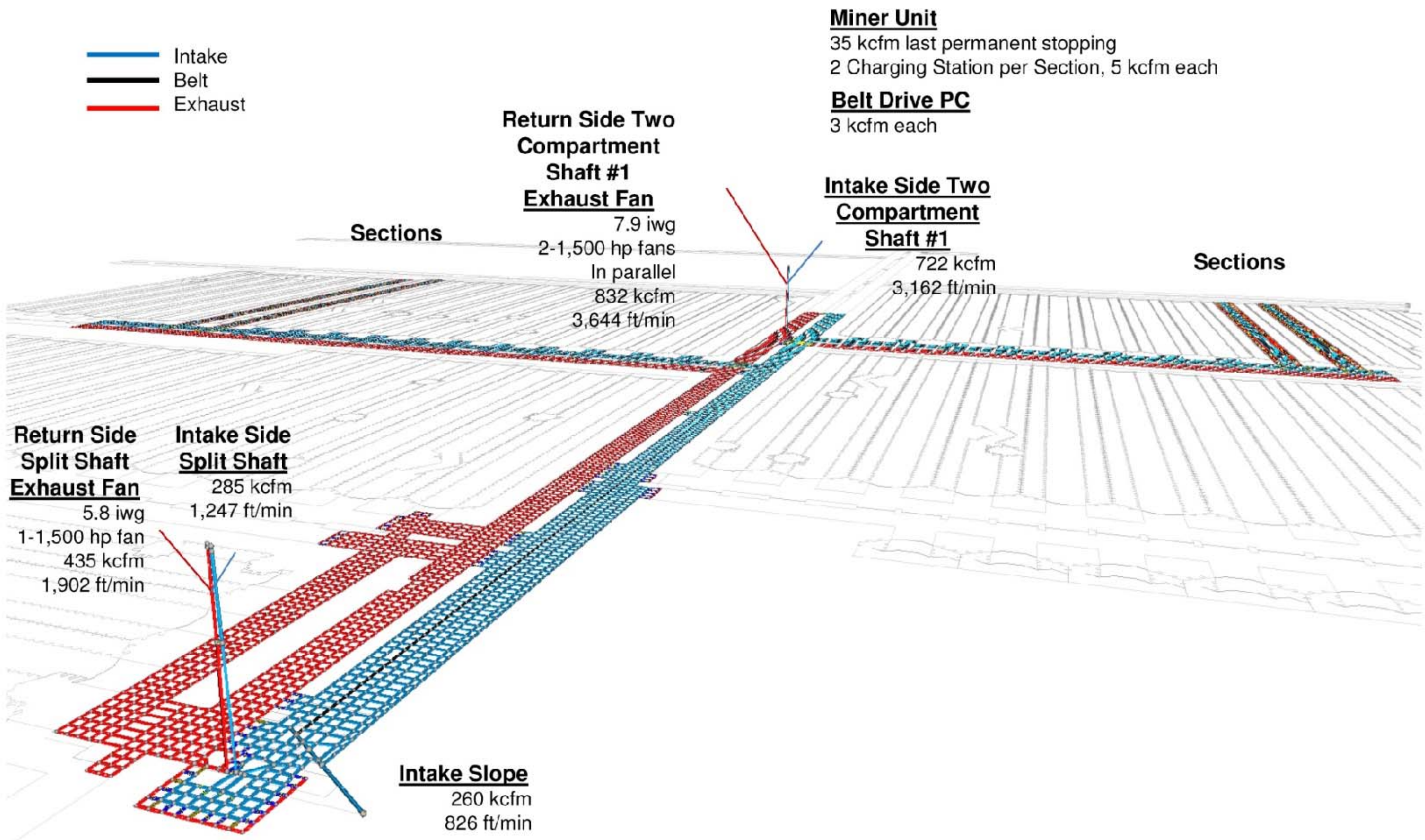


Figure 16-29. Three-Dimensional View of the Ventilation Model for Mining Years 26–50

ITEM 17: RECOVERY METHODS

17.1 Design Criteria

The processing plant design is based on the metallurgical test work conducted by ICP and described in Item 13 and uses the following design parameters:

- Total ROM feed to the plant: 3.69 Mtpy
- Ore composition:
  - o Polyhalite ($K_2Ca_2Mg(SO_4)_4{\cdot}2H_2O$) 80.0%
  - o Magnesite ($MgCO_3$) 7.68%
  - o Anhydrite ($CaSO_4$) 6.9%
  - o Halite (NaCl) 5.42%
- SOP production: 714,400 tpy
- Plant utilization: 90.32%

Table 17-1. Process Recoveries

Area	Recovery	Source
Crushing and Washing	96.5%	Results from both crushing and washing tests
Calcination	99.8%	Results from test work and regulatory exhaust limits
Leaching	95.0%	Results from ICP Pilot program
Crystallization and Evaporation	90.8%	Veolia test work and mass balance
Granulation	99.0%	Industry standards and input from equipment suppliers
Overall Recovery	82.24%	Test and pilot plant results

Table 17-2. Product Output Capacities

Product	Minimum Capacity (tpy)	Maximum Capacity (tpy)	Nominal Production (tpy)
Soluble SOP	0	100,000	90,000
Standard SOP	250,000	503,000	312,500
Granular SOP	250,000	385,000*	312,500

*Note: Granular SOP will be produced in two circuits rated at 192,500 tpy per circuit. The plant will start up with a single circuit with the second added in Year 2 of operation.

17.2 Process Description

The process involves several key unit operations to process conventionally extracted polyhalite ore from the mine to produce the SOP products. The main process circuits include crushing, washing, calcining, leaching, crystallization, drying, and granulation. Several ancillary circuits are included as well. Figure 17-1 illustrates the ICP process block diagram.

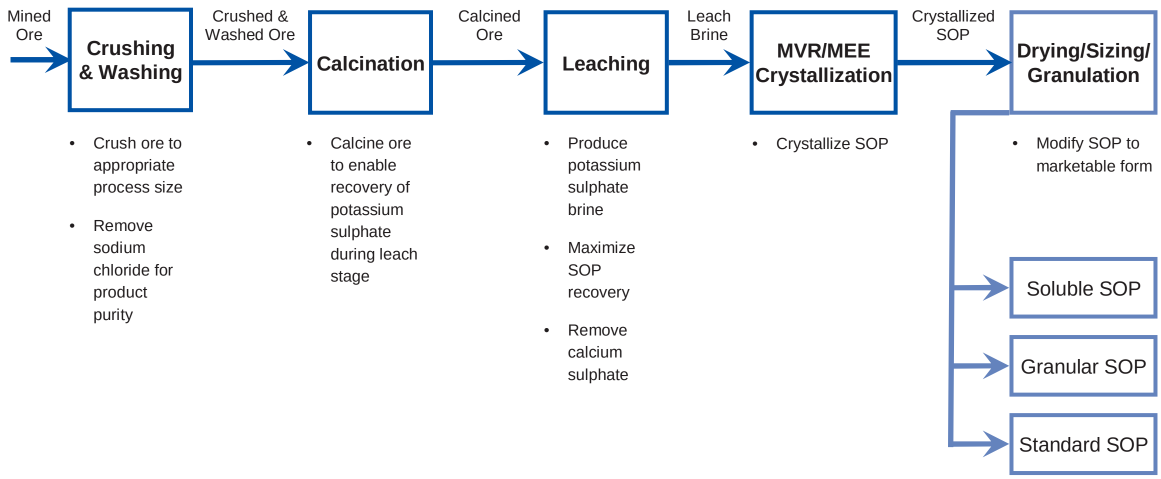


Figure 17-1. ICP Process Block Diagram

17.2.1 Crushing and Washing

ROM ore will be conveyed from the mine to the process plant via a series of belt conveyors. Two raw ore surge bins will be connected to the conveyors between the mine portal and the process plant to allow for surge capacity in the event of any unexpected interruptions in mining operations. The two raw ore storage bins are sized to hold ore from 10 hours of mine operation.

The first step in the comminution process consists of crushing and screening the ROM ore to obtain the particle sizing required by downstream operations. The ore coming from the mine will have a top size of 4 inches.

The ore will be drawn from the ore surge bins and sent to a roll crusher to reduce the maximum particle size from 4 inches to less than 1 inch. The roll crusher discharge will be fed to a pulping tank, where recycled water will be added to produce a slurry. This slurry will pass to a wet sizing screen, where the undersize will move onto the next stage of processing. The over-size will be sent to a Cage-Paktor, which will further reduce the particle size. The crushed particles will be recirculated with the crushed particle slurry from the roll crusher.

The second step in the process consists of removing the NaCl from the ore. Raw ore from the Ochoa site contains approximately 5.4% NaCl by weight. This impurity exists as very thin discrete layers interbedded with the polyhalite rather than forming a mixed bed as with sylvinite. NaCl is an undesired contaminant for two reasons: firstly, in the evaporation and crystallization stages, the concentration will be increased to a level that can cause serious issues with corrosion; secondly, if NaCl is allowed to remain in the system, it can eventually reach a level that constitutes a threat to product grade. Test work confirms that the salt is liberated in crushing and that dissolution is rapid and completed in a salt leach tank.

Salt dissolution begins in the wet portion of the crushing circuit. Additional dissolution occurs in a separate salt leach tank that provides additional residence time and ensures complete dissolution of the salt particles. From the salt leach tank, the washed solids will be separated from the high salt brine using hydrocyclones and vacuum belt filters. A portion of the cyclone overflow and wet end filtrate coming off the belt filters will be purged from the system and sent to a holding pond before being disposed of with deep well injection.

After the initial dewatering stage on the vacuum belt filters, a stream of clean water will be used to wash the filter cake and displace the high-concentration wash brine.

17.2.2 Calcination

The polyhalite ore must be heated to a temperature within the calcination range to result in leachable ore. The calcination reaction occurs in three distinct stages. The first stage involves evaporating the free moisture is evaporated from the polyhalite ore. The second stage occurs at or slightly above 320°F where the water of crystallization is severed from the polyhalite ore and vaporized. The final stage occurs slightly above 860°F, where molecular rearrangement takes place forming two solid solutions, one of which contains all the potassium and magnesium sulfate and is amenable to leaching in an aqueous solution. At temperatures above the target temperature range (i.e., higher than 986°F), additional reactions take place that reduce the leachability of the calcinate.

Fluid-bed thermal processing units have been selected as the preferred equipment for calcining the ore because they allow for excellent control of temperature and residence time, which are the main factors controlling the efficacy of the calcination reaction. After calcination, the ore is fed to a fluid-bed cooling unit. This is required for two reasons: (1) to facilitate handling the material exiting the calciner and conveying it to the leach circuit; and (2) to prevent severe flashing on introduction to the leach circuit. Process condensate, in submerged tubing, and air are used in the product coolers to reduce the temperature of the calcinate. Standard material handling equipment transfers the cooled calcinate to the leaching circuit. A 5-hour surge capacity between calcination and leaching accommodates any imbalances in operating rates.

17.2.3 Leaching and Brine Clarification

The Ochoa polyhalite process uses a two-stage counter-current leach circuit that is designed to produce the highest potassium sulfate concentration brine compatible with high recovery of the potassium contained in the calcinate.

Each of the primary and secondary stages of the leach circuit consists of four agitated vessels connected in series. Because the leaching reaction is endothermic, steam is utilized to maintain the temperature in the vessels at or near atmospheric boiling.

Calcined solids are fed to the first tank in the primary stage and mixed with brine produced from the second-stage circuit to produce the primary leach slurry. Slurry from the final vessel in the primary stage is fed to the SLS stage consisting of hydrocyclones and solid bowl centrifuges. From SLS, the brine proceeds to the next stage in the process, and the solids go to the second stage of leaching, where they are mixed with water to form the second-stage slurry.

The goal of the second-stage leach circuit is to recover essentially all of the potassium sulfate contained in the solids from the first-stage leach circuit and recycle it back to the first-stage leach circuit as brine. The solids produced in the first-stage leach are slurried with recycled condensate. All of the leachable material in the calcinate is either taken into solution in the primary brine or converted back to extremely fine re-crystallized polyhalite, a form that is essentially 100% soluble in low-potassium concentration brine. To ensure that all of the potassium in the feed to the leach circuit ends up in solution, the secondary leach brine concentration. The brine from the secondary centrifuges is returned to the first tank in the primary leach stage and the separated solids are collected and transported by truck to tailings disposal.

During pilot testing, sub-10 micron particles appeared in the first-stage leach brine. These may be eliminated during subsequent processes downstream. To ensure clear brine, extra clarification equipment was added to the circuit. After testing is performed in the detailed and bridge engineering phase, this filtration equipment may be removed or modified, depending on the results.

17.2.4 Crystallization

The crystallization circuit is designed to optimize recovery of SOP from brine produced in the leach circuit. Refer to Figure 17-2 for the crystallization circuit block diagram.

17.2.4.1 Leonite Dissolution—In the first part of the evaporation and crystallization process, leonite (K_2SO_4•$MgSO_4$•$4H_2O$), which is precipitated in the last stage of the process is dissolved in the leach brine. Leonite has the same equimolar ratio of potassium sulfate and magnesium sulfate as the brine produced in the leach circuit. Dissolving this material in the leach brine does two things: (1) it increases the concentration of the brine, thus reducing the amount of evaporation required to reach the SOP crystallization point and (2) it increases the amount of potassium sulfate contained in the feed to the SOP crystallizers, thus increasing the production of the desired end product.

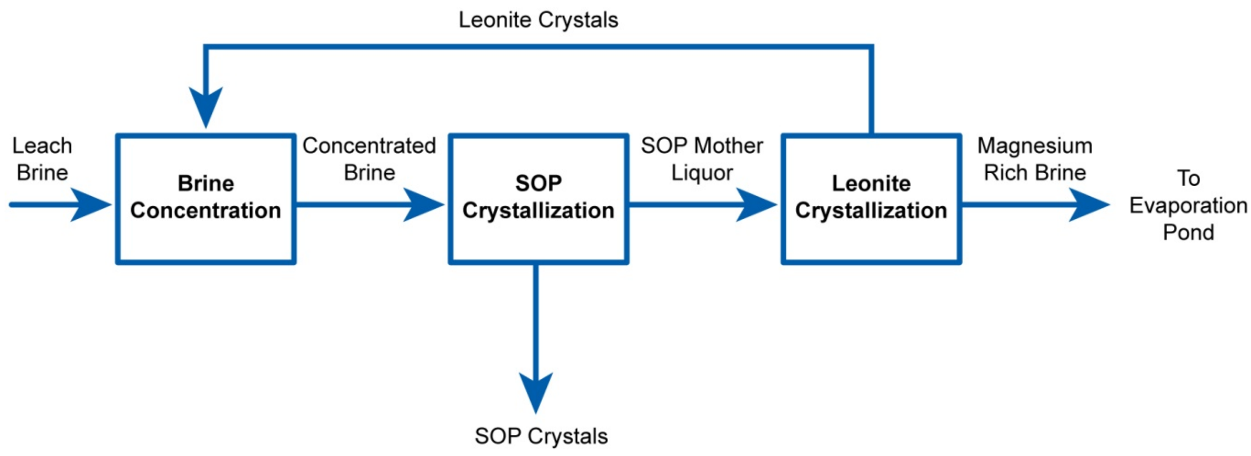


Figure 17-2. ICP Crystallization Block Diagram

Polyhalite seed material is added to the leonite dissolution tanks to aid in the removal of calcium oversaturation in the produced leach brine. This calcium will precipitate as a result of increased concentration of potassium and magnesium. Seeding is used to prevent scaling in the downstream pre-concentration unit. Following initial seeding, a portion of the material precipitated is recycled from the next step in the process as seed. After the leonite is dissolved and polyhalite seed is added, the brine is fed to the pre-concentrator circuit.

17.2.4.2 Pre-Concentration—The pre-concentration circuit further increases potassium and magnesium concentration in the brine by evaporation. This evaporation is accomplished using a single MVR forced circulation vessel. Concentration of the brine by evaporation drives calcium solubility even lower, resulting in additional precipitation of polyhalite, which precipitates as growth on the seed material rather than forming as scale in the heat exchanger tubes. If not controlled by seeding, the polyhalite precipitation could result in tube fouling, necessitating frequent outages for removal. The precipitated polyhalite solids, plus any fine-particulate calcium sulfate contained in the leach brine, are removed in the clarifier following the pre-concentrator.

17.2.4.3 Brine Clarification—The pre-concentrated brine is fed to a clarifier which produces almost clear overflow brine for feed to the SOP crystallizer. The underflow is processed in a solid bowl centrifuge to produce a high percent solids cake suitable for recycle to the calciners. The centrate from this unit is returned to the clarifier.

17.2.4.4 SOP Crystallization—The SOP crystallization circuit uses an arrangement similar to that of the pre-concentrator. In this case, two forced circulation MVR vessels are configured in parallel to evaporate water from the clarified brine, resulting in the precipitation of SOP crystals. Potassium in the feed stream can be precipitated as SOP before the brine composition approaches the leonite phase region. The crystallized SOP solids are removed using small thickener vessels and pusher centrifuges. Pusher-type centrifuges are used as they permit washing of the crystals to remove residual high-magnesium mother liquor, thus producing a drier feed cake that will meet product purity requirements.

17.2.4.5 Leonite Crystallization—Mother liquor from the SOP crystallizer serves as the feed brine for the leonite crystallization circuit, consisting of two parallel triple-effect trains. A portion of the first effect vapors are fed to a thermo-compressor to increase the temperature and pressure and then fed to the first effect heat exchanger; the balance of first effect vapors are sent to the heat exchanger on the second effect. The second-stage evaporator is operated under a lower pressure (and therefore lower temperature) than the preceding unit, allowing the vapor produced in the first effect to be condensed in heating the second effect. The same process occurs between the second effect and the third effect. Evaporation of water and thus concentration of brine in all three effects results in the precipitation of leonite crystals, which are removed using settling vessels and centrifuges and sent back to the leonite dissolvers. Because the production of clean water at the proposed site is expensive, an air-cooled condenser rather than the more commonly used evaporative cooling tower has been incorporated into the design to condense the vapor produced from the evaporator system.

Mother liquor from the third leonite crystallizer is purged from the system. Further evaporation of water from this stream at temperatures reasonably achievable in commercial equipment would result in the precipitation of magnesium sulfate and the contamination of the system. This purge is sent to evaporative waste ponds. Recovery of magnesium sulfate would be possible in the future should that become economically attractive.

17.2.5 Product Drying, Granulation, and Sizing

Following crystallization, SOP will be processed into three different products: soluble, standard, and granular SOP. The circuit is designed to allow flexibility in production for each of the three SOP products. Soluble grade SOP can be varied between zero and 100,000 tpy, granular product can fluctuate between 185,000 and 385,000 tpy, and standard grade product can fluctuate between 250,000 and 503,000 tpy.

The crystal cake from the centrifuges is first dried in a fluid bed dryer to remove any residual moisture and produce a completely dry product. Next, a series of multi-deck screens are used to separate the crystals with respect to the size specifications of the soluble and standard products. Oversize material is passed through a single-stage roll crusher and recycled to the top of the screens.

Screen cuts meeting soluble and standard product size specifications are sent to the product day bins to be loaded into the trucks and transferred to the loadout facility, with the remaining material from the cut being sent to the granulation circuit to produce the granular product.

The process has been designed to meet the commercially available product specifications for all three SOP products. This includes the guaranteed minimum K_2O content as well as the elemental allowances including magnesium, sulfur, chlorine, calcium, sodium, insolubles, and percent moisture.

<u>SOP Granulation</u>

Once the required volumes of soluble and standard product have been separated by screening and sent to day bins, the balance of the dryer discharge is sent to a granulation circuit which produces a product meeting grade and product sizing specifications.

The granulation circuit is designed as two separate trains: one to be installed initially, with the second set of equipment installed at a later date when output requirements dictate.

To produce a granular product with a high strength level, it is necessary to have a distributed crystal size so that smaller crystals may fill the voids as the granule grows to ultimately produce a low-porosity (high-density) product. To ensure this need is met, approximately 30% of the granulation feed is separated for further size reduction through a vertical fine grinding mill. The material is re-combined with the balance of the split and cooled to reduce the temperature of the particles to optimize the granulation process. Granulation is performed using tilting pan granulators and pin mixers for conditioning the feed to the pans. A binding agent is added, as required, to achieve proper granule sizing and strength. Granulator discharge will be dried in a fluid bed dryer and screened. Material meeting specifications is sent to the day bins and while over- and under-size material is re-circulated to the granulation circuit.

17.2.6 Product Loadout

The three SOP products will be conveyed to the site loading area, located east of the granulation and drying process area. Each product (standard, granular, and soluble SOP) will have its own dedicated storage bin with the capacities of 600, 600, and 150 t, respectively. The site loading bins will be elevated and positioned to allow the transfer trucks for each product to enter the loading area beneath the respective loading bin without impacting the traffic of the other product loading trucks. The products are trucked approximately 22 miles over public roadways to the product storage and loadout facility located northwest of the community of Jal.

Dust collected at the site loading area will be directed to the SOP area dust baghouse for collection and recycle.

17.2.7 Reagents

Throughout the process, two reagents will be required: SOP binder and de-dusting agent.

The SOP binder that will be used is a lignin-based organic polymer, which is added in the granulation process to form the granule-like particles. The raw lignin will be mixed in an agitated vessel and diluted with process water. It will then be pumped to a holding tank and added to the process as needed.

Prior to the product leaving the loadout facility, a de-dusting agent will be added to the product on the conveyors to prevent dust release during shipment. Typically, 2 lbs/t will be added for the granular product, while 1 lb/t will be added for standard SOP. The de-dusting agent will be stored in a storage tank and added to the process as needed.

17.2.8 Utilities

17.2.8.1 Standby Power—Grid power will be backed up by a 2,000-kilowatt (kW) standby diesel-fired electrical generator sized to supply critical equipment in the process plant in the event of main power loss. Diesel fuel will be delivered by truck and stored in an above-ground horizontal storage tank, from which it will be pumped as needed to a day tank located adjacent to the generator.

17.2.8.2 Steam Boilers—The process requires approximately 300,000 lbs/h of steam for operation. The major users include the leach tanks, the leonite dissolution tank, and the thermo-compressor on the leonite multiple-effect evaporation.

17.2.8.3 Compressed Air—Four compressed air systems are specified to service the facility as follows:

- Plant air will be supplied by a single compressor with a standby compressor servicing any utility stations in the process area
- The loadout facility air will be supplied by a single compressor with a standby compressor and a heatless desiccant air dryer servicing all utility stations, instruments, and baghouses at the loadout facility
- Raw Ore Area air will be supplied by a single compressor and a heatless desiccant air dryer servicing all utility stations, instruments, and baghouses in the vicinity of the raw ore transfer station
- Instrumentation air will be supplied by three compressors (two in parallel and one standby) and two heatless desiccant air dryers in parallel servicing all instruments and baghouses in the main process site

Each compressed air system is equipped with filters upstream and downstream of the air dryer (one filter for plant air), and wet air receivers upstream and dry air receivers downstream of air filtration/drying. All baghouses will have a dedicated small air receiver located close to the baghouse.

17.2.8.4 Vacuum System—A set of three vacuum pumps in parallel provide the motive force for the vacuum belt filters in the sodium chloride washing circuit. Each pump is equipped with a silencer and is cooled using process water.

17.2.9 Tailings

The plant design incorporates four different systems for handling the tailing materials produced from the process.

17.2.9.1 Gypsum Tailings Storage and Handling—The gypsum tailings are separated from the leach brine in the leaching circuit as the precipitated solids are formed. The solids are analyzed with a K-40 analyzer and scale to determine the amount of gypsum tailings and the corresponding potassium losses, and are then collected in a surge bin. Transfer trucks will be used to transfer the material 1.64 miles to the calcium sulfate storage pile in the TSF.

17.2.9.2 Magnesium Sulfate Bleed Evaporation Ponds—The magnesium sulfate bleed stream from the evaporation/crystallization circuit will be delivered through a pipeline to the magnesium sulfate evaporation ponds. There will be four high-density polyethylene-lined magnesium sulfate evaporation ponds operated in series. Initially, the plant will be constructed with two of the four ponds, while the remaining two will be put in place when increased volume is required. The ponds will be designed to allow solids harvesting via mechanical means.

Solar evaporation combined with the formation of Epsom salt ($MgSO_4$•$6H_2O$) will remove all water from the bleed stream so that there is no liquid discharge from the final pond. The salts formed will be collected and added to the tailings pile.

17.2.9.3 Tailings Evaporation Ponds—The NaCl wash circuit bleed, boiler blowdown, and the RO bleed streams are collected in a tailings pumpbox and then delivered by pipeline to the brine holding pond. Any liquid collected from the calcium sulfate storage pile sump will also be forwarded to this pond. The pond will have a 52-day storage capacity. Because solar evaporation will remove only a small portion of the water, a deep injection well is required to dispose of the brine after the 52-day residence time.

17.2.9.4 Deep Well Injection—Positive displacement injection pumps will be used to pump the excess brine from the evaporation ponds to an injection zone. Five injection wells are included in the design for disposing of 1,200 gpm.

17.3 Plant Water Balance

The water source for the plant operations is the Capitan Reef water well field located approximately 13 miles away. The plant will use both raw and treated water for different operations within the process. The process design requirements for water are provided in Table 17-3. The water requirements from the RO treatment facility are also indicated.

The raw water to the process will be used for leaching the salt contained in the ore. The salt-leached ore will then be sprayed with process water to remove any residual salt and raw water.

Because the Capitan Reef water contains a high level of TDS and undesired impurities, it is necessary to produce clean process water to avoid contamination of the process streams.

The water treatment plant (WTP) will use filtration and RO to produce 1,225 gpm as the nominal requirement. The primary RO water user is the make-up to the leach process. As the leach brine is sent to crystallization, the majority of the water will be evaporated and then

Table 17-3. Process Design Requirements for Water

Water Type	Flow Rate (gpm)
Produced Water from Well Field	2,714
Pretreated Water	1,053
RO Water	1,661
Potable Water	40
Mine	160
Salt Wash	116
Leach Circuit	893
Boiler Makeup	8
Reagents	8
Waste	435

condensed and re-circulated to the leach circuit; however, there are several losses in the system that require a make-up of 900 gpm. The primary losses include crystallization purge and the gypsum tails removal. A portion of the condensate is also used for steam generation which increases the water make-up to the system.

Approximately 40 gpm of RO water is required for potable water consumption. Additionally, 160 gpm of RO water is fed to the mine for dust control and mining operations. Other ancillary point users include reagent mixing, vacuum pumps, and centrifuge cooling.

17.3.1 Process Water System

The water system is designed to pump up to 3,000 gpm of raw Capitan Reef water from five water wells, with each well having a 600-gpm capacity. First, the raw water is stripped of H_2S through packed bed air strippers located in close proximity to the water wells. H_2S is removed from the water because it can cause corrosion of the pipeline and processing equipment, lead to fouling, and affect the process chemistry. Next, the water is pre-treated via pressure filters to remove any particulates to facilitate operation of the RO units downstream. Process water storage tanks after both the air strippers and filters are positioned for surge capacity.

17.3.2 RO System

The RO system is designed to produce 1,225 gpm of process water. The system consists of three parallel double-stage cartridge filters with a 75% permeate recovery. The permeate is then pumped to storage tanks located at the tank farm, while the concentrate (waste) is discarded in the evaporation ponds.

17.3.3 Cooling Water System

Cooling water will be required for the following processes:

- Granular SOP product cooler
- Centrifuge lube system cooling
- Pump seals
- Vacuum pump cooling

RO make-up water, which will be added in the leach tanks as process condensate, will be used as the cooling source for the granular SOP product cooler prior to delivery to the process condensate storage tank. RO make-up water will also be the cooling source for the centrifuge/compactor lube cooling and vacuum pump seal cooling.

17.3.4 Condensate System

The evaporation/crystallization circuit recirculates the vapor produced via the evaporation process to heat the incoming brine. The vapor is condensed, collected, and recycled back into the process. The steam boilers that supply the steam for both the crystallization and leach circuits are fed recycled condensate. The remaining condensate, along with the make-up water from the RO system, is recycled to the second-stage leach circuit as the leachate for the solids produced in the first leach circuit.

The condensate circuit consists of two feed tanks: one solely dedicated to feeding the steam boilers, and the other for eventual recycle to the leach circuit. Different condensate collection points can be routed to either tank depending on operations.

17.4 Process Control and Monitoring

The process control and monitoring system is designed to require a minimum number of operational personnel and provide essential instrumentation. The main Central Control Room will be equipped with the necessary controls and monitoring systems for complete site operations. Auxiliary control rooms will be provided throughout the site to facilitate operator access to the control system.

The level of automation and control will provide normal process and control functions and will communicate efficiently over a fiber-optic and fieldbus cable network with built-in redundancy for no loss of communications between the instruments and the primary Distributed Control System. Provisions will be made to store historical data.

17.5 Process Building

A multi-story steel structure houses the process areas and electrical and utility rooms. Partial roof and wall enclosures have been added in the drying, sizing, and granulation areas for wet weather protection. Electrical rooms accommodating major electrical equipment are enclosed from the remainder of the plant by fire separations.

To help contain spills and facilitate cleanup, solid floors are used under all equipment that has spill potential. Elevated floor areas are checkered plate/open grating, except where the centrifuges are on concrete slabs. The salt washing, crystallization, and de-brining areas are considered wet processing areas and therefore, are provided with epoxy floor painting for additional corrosion protection. Sloped flooring with drains is provided in areas where liquid spills could occur and in areas designated for washing. Sloped floors will also be used at grade in normally dry areas to facilitate some washing.

During the FS, SNCL created a 3D model of the process plant using Autodesk® 3D AutoCAD® software. The model includes mechanical equipment, chute runs, flooring, accesses, and layout steel. The model does not include the evaporation and crystallization plant which is part of Veolia’s design and was not modeled by SNCL. Figures 17-3 to 17-6 show representative screen shot views of the process plant 3D model.

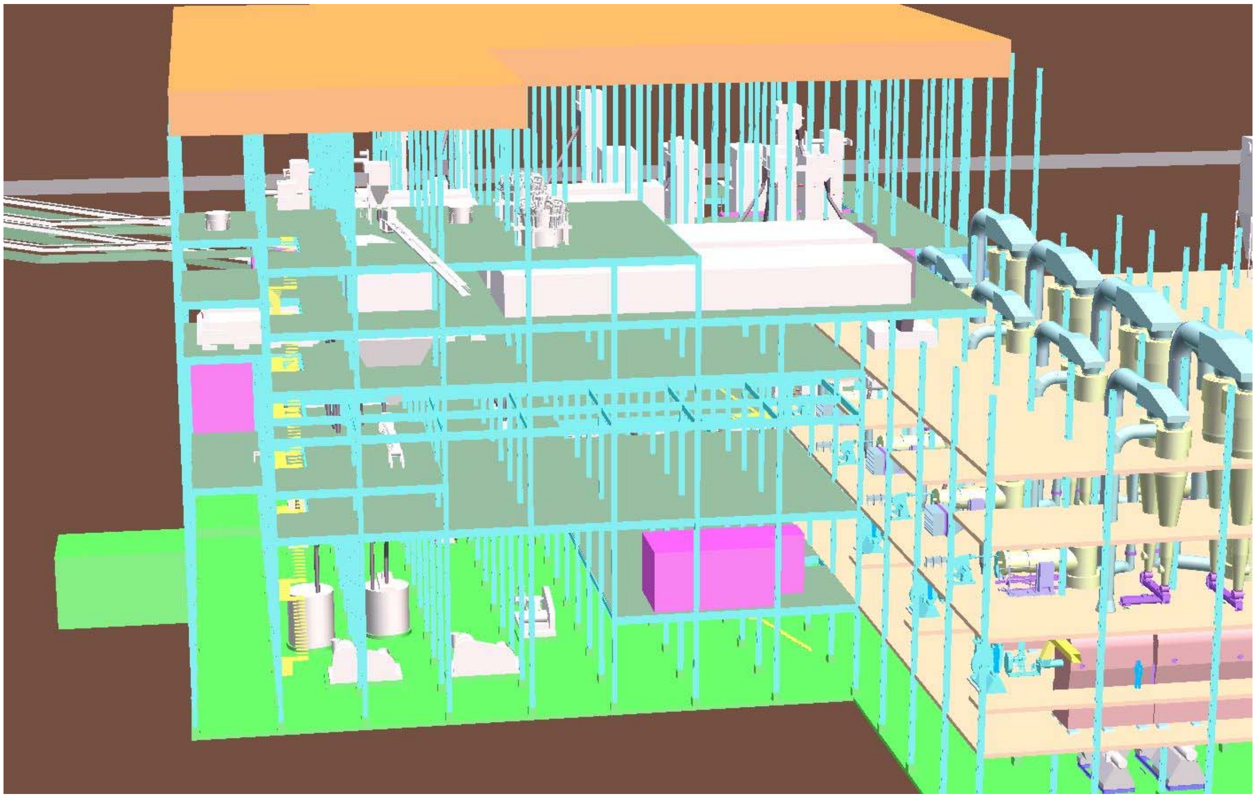


Figure 17-3. Area 24200 Salt Washing (looking south)

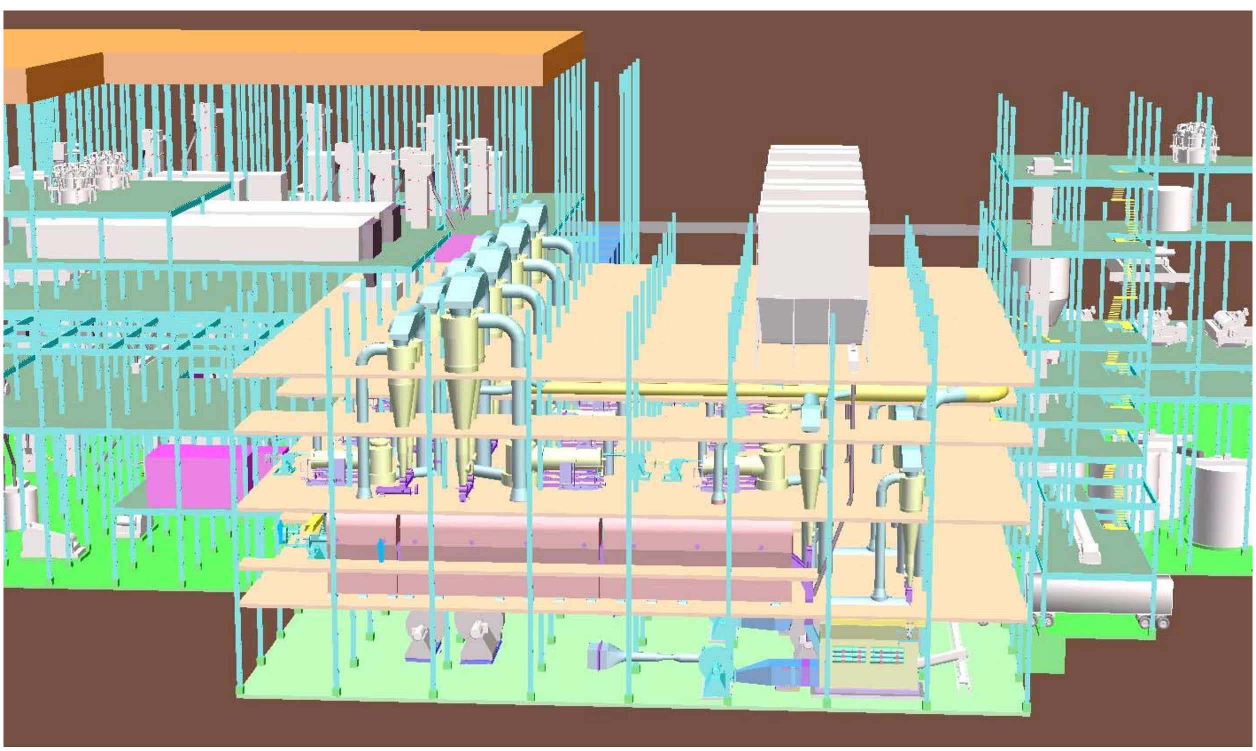


Figure 17-4. Area 24300 Polyhalite Calcining (looking south)

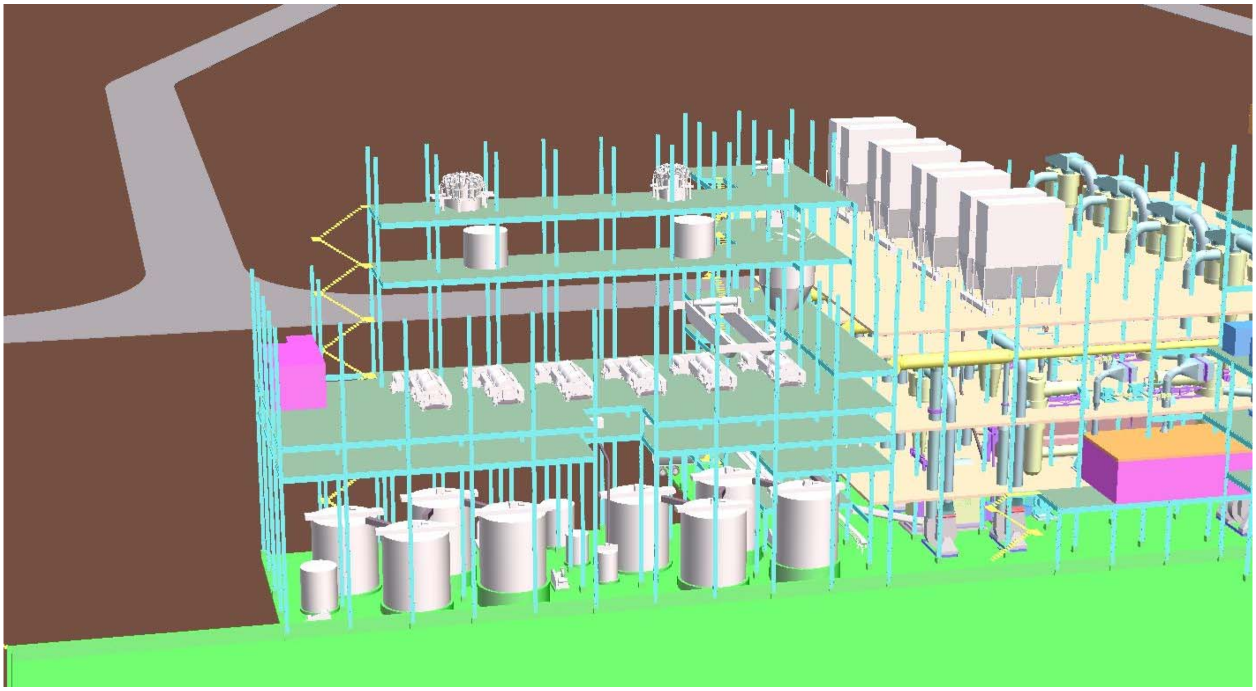


Figure 17-5. Area 24400 Leaching/Brine Clarification (looking north)

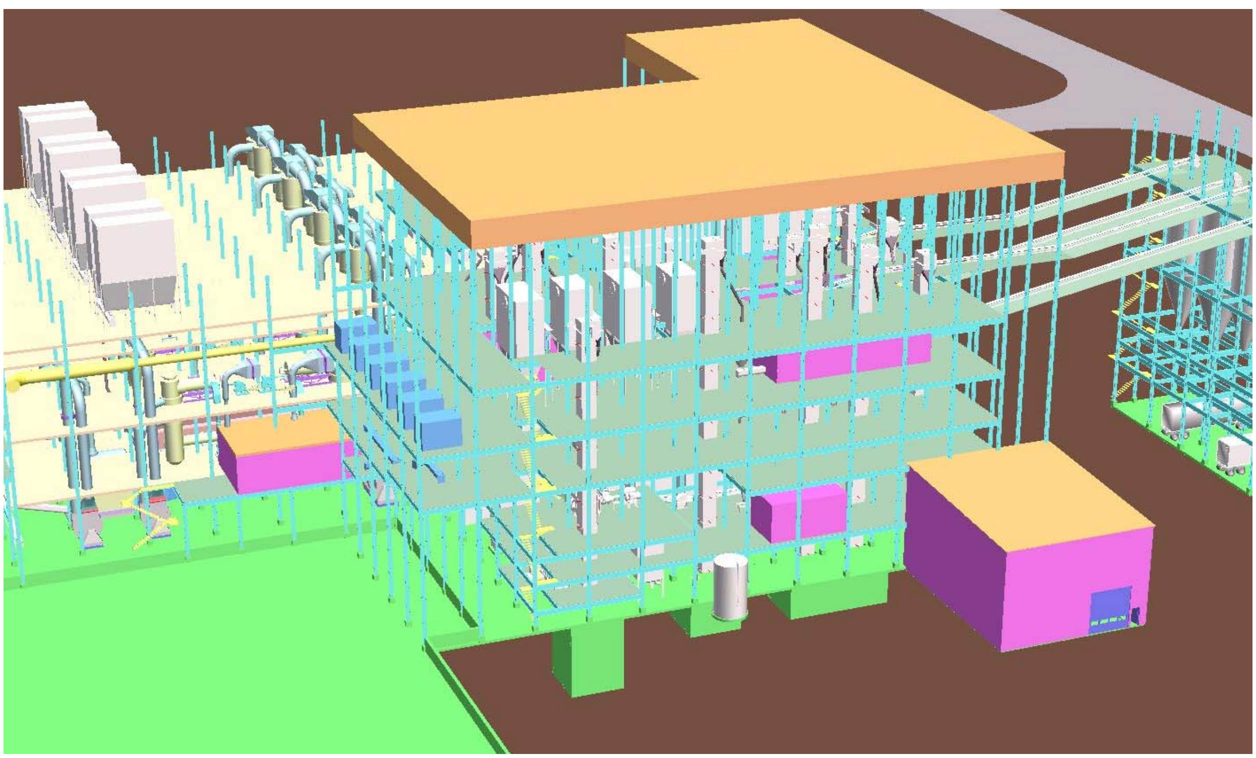


Figure 17-6. Area 24600 Product Drying and Sizing (looking northeast)

ITEM 18: PROJECT INFRASTRUCTURE

The Property is primarily located in Lea County, New Mexico, with a small portion in Eddy Country, New Mexico. The Project site is approximately 60 miles east of Carlsbad, New Mexico, and less than 20 miles west of the Texas/New Mexico state line.

The Project site is readily accessible via public highway SH 128. The principal access to the process plant and mine sites construction and operational activities will be from public highway SH 128, which runs east to west through the Project site.

The loadout facility will be located northwest of the community of Jal, New Mexico which is approximately 22 miles east of the process plant. The loadout facility will be located just north of Phillips Hill Road and west of the existing TNMR mainline.

18.1 Off-Site Facilities

18.1.1 Site Access

The mine shaft site will be accessed via the Brininstool Road from SH 128. Approximately 760 ft of two-lane access roadway will be constructed from Brininstool Road to the mine shaft site. No alterations to the highway (turn lanes, acceleration lanes, etc.) are included in the scope of the project.

The process plant site will be accessed from SH 128. Approximately 2,170 ft of two-lane access roadway will be constructed from SH 128 south to the plant. Plans are to construct an acceleration lane on SH 128 and possibly a left turn lane.

The loadout facility site located north of Jal will be accessed from Phillips Hill Road. Approximately 6,200 ft of two-lane access roadways will be constructed on the loadout facility.

Figure 18-1 shows the overall site plan of the Ochoa Project process plant and mine site. Figure 18-2 provides a plan view of the process plant and surrounding facilities.

18.1.2 Electrical

Xcel Energy will construct a new 345-kV or 230-kV service line to the process plant site with a stepdown to 115 kV. Electrical power will be supplied for construction activities from an existing power transmission line on the Project site. Initial construction activities will temporarily rely on diesel-generated electricity until the construction substation is operational.

18.1.3 Natural Gas

Natural gas required for the process plant operations will be provided by one of several natural gas suppliers in the region. A new underground pipeline adjacent to SH 128 will be installed to service the Ochoa Project. A natural gas regulator station will be installed west of the process plant to provide natural gas for the process plant.

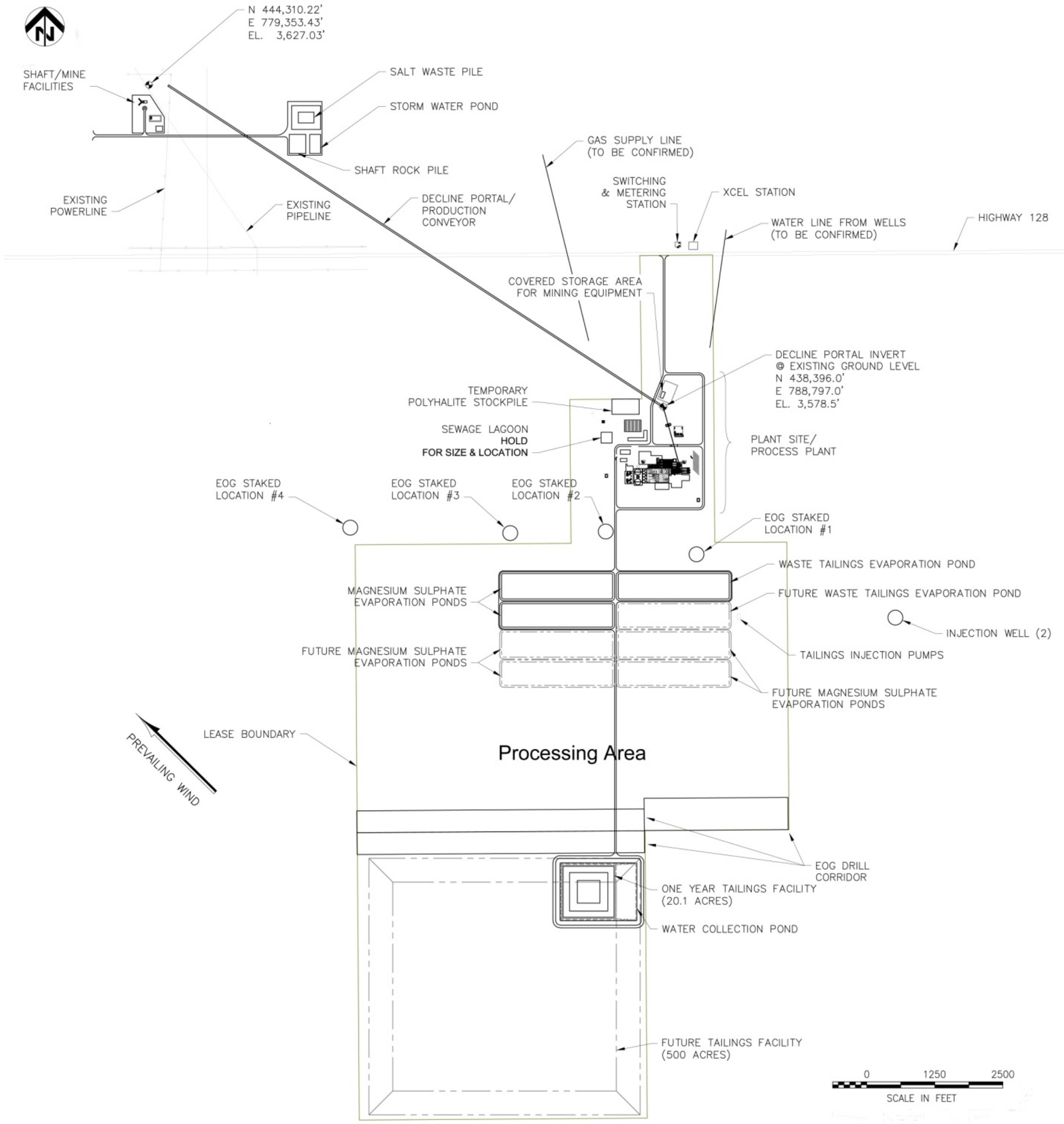


Figure 18-1. Overall Site Plan of Process Plant and Mine Site

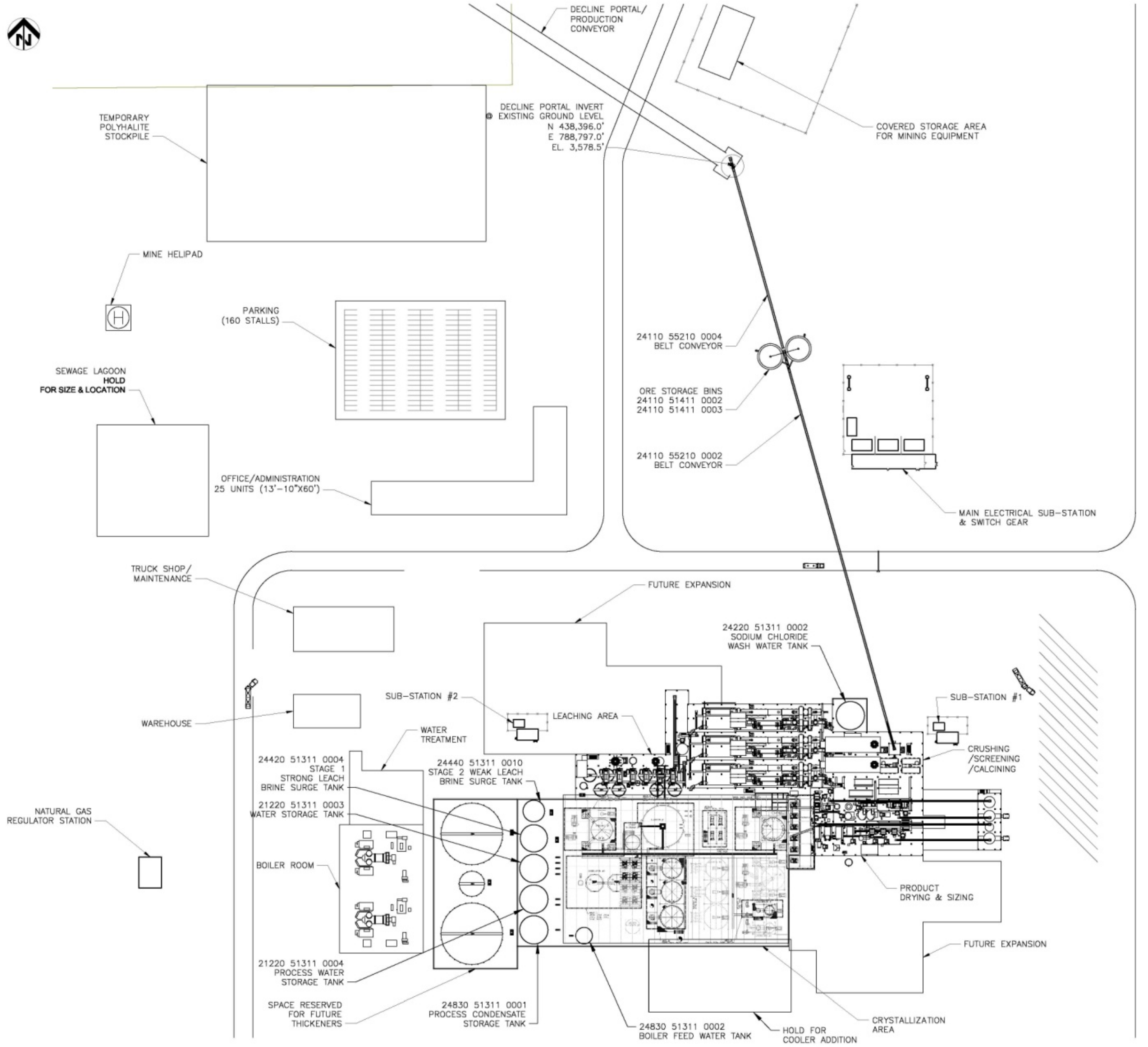


Figure 18-2. Process Plant Area

18.1.4 Water

Water for the project will be sourced from the Capitan aquifer. Water wells, a pumping and distribution system, and a pretreatment system will be located at the well field site approximately 13 miles northeast of the process plant. The WTP will be located adjacent to the process plant.

Potable water requirements for the loadout facility and the mine shaft site will be delivered by truck.

18.1.5 Telecommunications

Backbone links will allow for voice and data communications during construction and operations phases of the Project. Public switched telephone network (PSTN) and internet service links will be provided by the local service provider. It has been assumed that the loadout facility is the nearest ICP location to existing PSTN facilities. Therefore, voice and data services will be connected to the loadout facility office complex and then shared with the process plant and mine site via a microwave connection. A microwave link will support the voice and data transfer between sites.

18.1.6 Rail

The loadout facility will be located adjacent to an existing TNMR rail line. The project will construct new railroads and switch assemblies to connect to the TNMR line, a short-line railroad that runs from Lovington, New Mexico, to Monahans, Texas, and passes through Jal. The TNMR connects to the Union Pacific Railroad at Monahans. Figure 18-3 provides a view of the loadout facility and rail yard.

18.2 Site Preparation, Earthwork, and Roadwork

18.2.1 Site Roads

Temporary and permanent roads have been designed using standard construction practices to minimize surface disturbance, erosion, and visual contrast, and to facilitate reclamation. Roads have been designed following Best Management Practices and BLM road requirements as described in the BLM Road Manual 9113 (BLM 1985).

The access road from SH 128 will be chip seal paved to the product loading facility at the process plant. All other roads will be gravel or caliche. The main access roads and material haul roads will be two lanes with a width of 24 ft with 5-ft-wide shoulders. Tailings and ore haul roads will be one lane with a width of 20 ft with 5-ft-wide shoulders. Minor supporting roadways will be one lane with a width of 12 ft with 3-ft-wide shoulders.

18.2.2 Parking and Walkways

Employee and visitor parking lots have been located adjacent to the main access roadways and within close proximity to the administration buildings at the process plant and loadout facilities. The parking lots have been sized according to the anticipated employee numbers at both sites. Parking stalls and walkways will be gravel or caliche and equipped with lighting for security and safety.

18.2.3 Helipad

A helipad area that will facilitate access by emergency response helicopters in case of a medical evacuation emergency is shown on the site layout drawing Figure 18-2.

18.2.4 Fencing

Two types of fencing will be used for the Project. A perimeter fence, 4 ft high with three strands of barbed wire will be installed at each site. Safety fencing constructed of an 8-ft-high chain-link fence topped with three strands of barbed wire will be installed to protect personnel and property in safety and security sensitive areas.

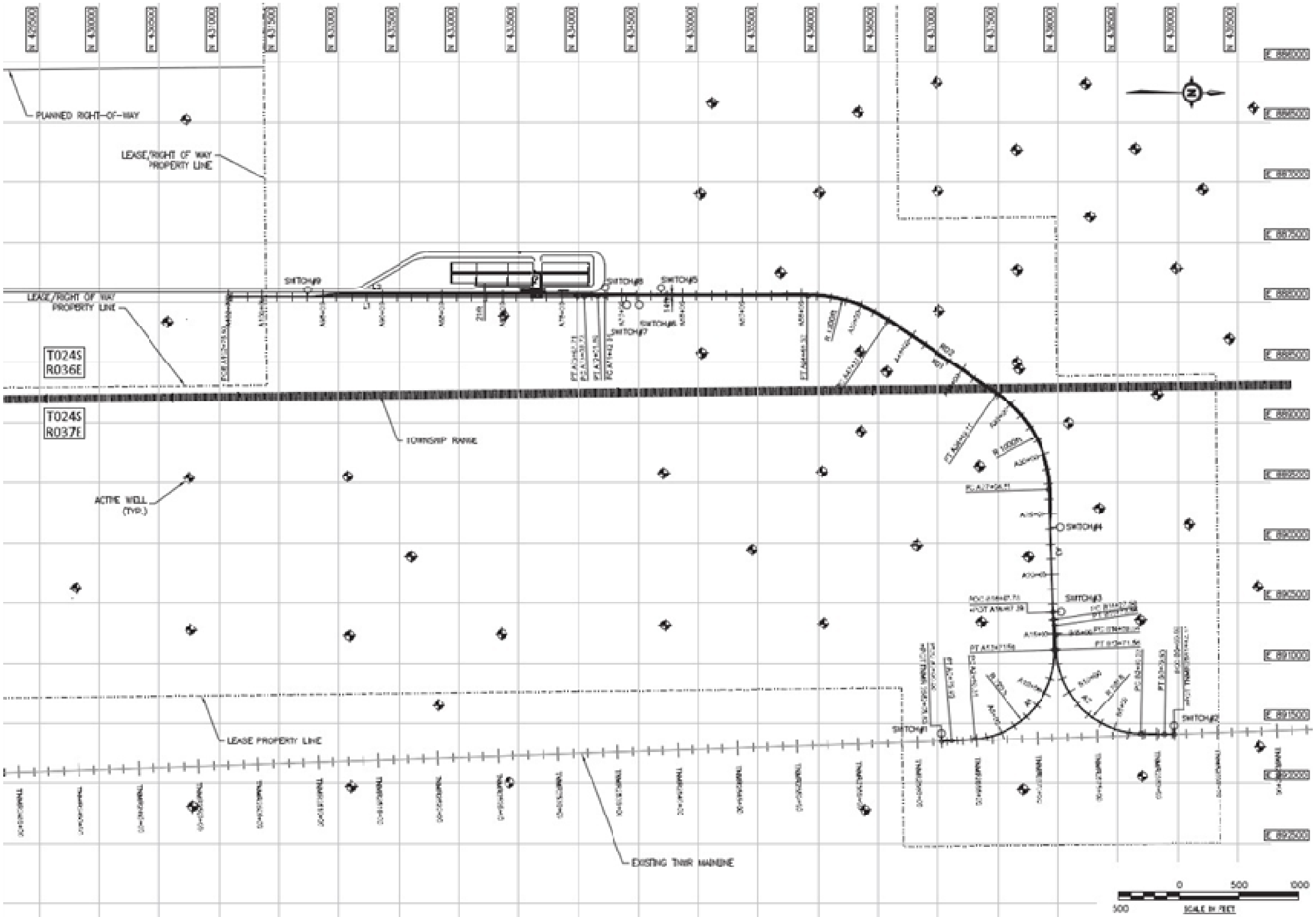


Figure 18-3. Loadout Facility and Rail Access

18.3 Water and Wastewater Systems

18.3.1 Water Treatment System

Raw water from the well field site will be pre-treated at the well field to remove hydrogen sulfide and slime forming bacteria, and will be stored in a water storage tank at the well field for conveyance to the WTP. The WTP will use filtration and RO to treat the pretreated water to defined water quality levels for plant process needs. The clean water output from the WTP will be stored in process water and potable water storage tanks. Water in the potable water storage tank will be further treated with a second-stage RO to meet potable water standards.

18.3.2 Fire Water

Water required for fire suppression within the process plant will be supplied from the raw water storage tank. A dedicated fire pump system will be used to supply fire water throughout the plant. The fire pump system includes two electrically driven pumps with a diesel driven standby pump.

18.3.3 Surface Runoff Water

Rainfall runoff will be managed by constructing protective berms around all disturbed areas and surface facilities at the mine, process plant, and loadout facility sites. Berms will prevent clean water runoff from entering the projects sites. Storm runoff within the sites will be contained and diverted to ditches leading to collection ponds. Sites will be graded and stormwater collection ponds will be located to take advantage of the natural slope of the site. The stormwater collection ponds will be sized to contain runoff from a 100-year, 24-hour storm event.

18.3.4 Waste Disposal Facilities

Sanitary waste from each of the facilities at the process plant site will be piped to a sewage lagoon designed to store domestic sewage for a period of 2 years assuming 1,230 cubic feet of sewage will be produced daily at the process plant site. The lagoon will need to be emptied every 2 years.

Sanitary waste at the loadout facility will be piped to a septic tank sized to meet the waste disposal requirements of the loadout facility for a period of 45 days. The septic tank will be periodically emptied with the aid of a pump truck.

18.4 Steam Boiler Systems

Two natural gas fired steam boilers will be used to generate steam for the crystallization section, and Stage 2 leach tanks in the process plant. The boilers will be designed to comply with the American Society of Mechanical Engineers, Section VIII, Division 1, Boiler and Pressure Vessel Code, and MSHA regulations.

Steam output will be at 356°F and 150 pounds per square inch gauge. Two boiler feed pumps will feed 150°F water from the boiler feed water tank. Recirculation pumps will be provided for each boiler. The steam boiler system will be finalized early in the detailed design stage.

18.5 Fuel Storage and Distribution

Natural gas will be supplied by the utility provider to a regulator/metering station located on the west side of the process plant. Main line pressures will be reduced then distributed through underground piping to the main users such as the calciners and the boiler plant.

Fuel for the mobile equipment and product/tailings trucks will be delivered to site by a local fuel supplier. A fuel storage facility, consisting of a number of double-wall storage tanks will be located on the east side of the plant.

18.6 Electrical Power

Xcel Energy will construct either a 345- or 230-kV service line to the process plant site. A 115-kV line will be provided to the ICP switching/metering substation. From there, an overhead 115-kV line will feed the main substation north of the process plant. A second 115-kV overhead line will feed a substation at the mine shaft site.

The running load for non-mining facilities is estimated at 59 megawatts (MW) (66.6 MVA demand at 0.9 power factor). The design has considered 20% spare power for future expansion. A provision to improve the power factor to 0.98 or better is included in the feasibility budget estimate.

18.6.1 115/12.47-kV Main Substation

SNCL performed an electrical rating calculation study for load flow, short-circuit, and motor starting calculations to obtain criteria that were followed for design calculations and equipment specification and selection.

The electrical system has been designed to minimize the impact of arc-flash hazards and maximize protection as defined in National Fire Protection Association 70E, American National Standards Institute Z535.4, and Institute of Electrical and Electronics Engineers 1584. The potential fault short-circuit level imposed on equipment will be kept as low as possible at each voltage level used. Protection relay trip delays will be set at the shortest time as determined in the electrical rating calculation study. This will be more completely addressed during the detailed engineering phase after the final equipment selections have been made.

At the main substation, three 45/60-MVA oil-immersed transformers will step down the incoming 115-kV feeder to the main 12.47-kV switchgear. From there, feeders of 12.47 kV will provide power to the entire plant area through two substations and seven electrical rooms. Power factor and harmonic improvement units will be installed in the main substation for efficient power and to comply with requirements of the utility provider.

Power control and an automation system will be provided in the main substation and connected to all electrical rooms for protection, monitoring, and controlling the electrical equipment.

18.6.2 Power Distribution

The three main 12.47-kV switchgears (one per main transformer), in the main substation, are connected with tie-breakers intended to keep them open during normal operation. This allows one of the main transformers to be disconnected for preventive maintenance or repairs while maintaining an adequate level of power to the operations.

Two 4.16-kV substations equipped with 12.47/4.16-kV, 20/25-MVA oil-immersed power transformers will be fed from the main substation. These will supply all the 5-kV MV Motor Control Centers (MCCs) in the process plant. A 4.16-kV feeder will supply the RO electrical room from Substation 1. A 4.16-kV feeder will supply the TSF electrical room from Substation 2.

Five low-voltage 480-V MCCs equipped with 12.47/0.480-kV, 2/2.66-MVA dry-type transformers will be fed from the main substation. All non-process low-voltage loads will be fed from these five MCC rooms directly through small 480/208-V dry-type transformers. One 12.47-kV MCC is located in Electrical Room 2 fed directly from the main substation for six motors of more than 4,000 hp each. Three 12.47-kV variable frequency drives of 7,916 hp each will be fed individually from the 12.47-kV main switchgear.

18.7 Telecommunications and Security

Telecommunications will incorporate proven, reliable systems to ensure that related site facilities are well equipped to meet the project requirements. Equipment redundancy is used in critical and main components to ensure reliability. The telecommunication system design consists of the following items:

- Outside plant fiber-optic cabling system
- Building structured cabling system
- Local/wide area network, internet, and intranet services
- Voice over Internet Protocol telephone systems for voice and fax
- Mobile radio systems and antenna towers
- Microwave link between process plant and loadout facility
- Security closed-circuit television and access control system
- Uninterruptible power supplies

18.8 Ancillary Buildings

Ancillary buildings will be open structures where possible. Modular trailer arrangements will be installed to provide space for administrative functions and personnel facilities.

18.8.1 Administration Building (Process Plant Site)

Administrative functions will be located in a central modular trailer complex. The structures will be located in close proximity to employee parking and the entrance to the process plant site. The overall building area will be approximately 20,750 ft^2 and consist of mine and process engineering, administrative areas, change room facilities, first aid station, security area, and training facilities.

The administration office area will include 9 offices, a reception with waiting area, a 14-person conference room and incidental storage. The engineering office area will include 4 offices, 31 workstations, two 12-person meeting rooms, and a printing/library area.

A 48-person lunchroom, with kitchen, will be located at a central location within the administration building. A visitor orientation room will be located in close proximity to the reception area.

Separate change areas with a capacity of 150 personnel each will be provided for the mine and process plant male personnel. A separate change area will be provided for 13 female personnel. Each change area will consist of a dirty and clean side with locker-mounted benches. A single full-height locker will be provided for each employee on both the dirty and on the clean sides. Showers will be provided in each change area.

Modular structures will be constructed and erected in conformance with local codes and regulations and equipped with heating, ventilation, and air conditioning (HVAC), plumbing, sanitary, and fire protection services to meet the functional requirements of each facility. Stormwater will be directed from the roof area and will be surface drained away from the building.

18.8.2 Storage Warehouse

Warehousing of major components will be located outside in lay-down areas with improved graveled surfaces. A fabric-type cold storage warehouse of approximately 9,450 ft^2 will be located at the process plant site in close proximity to the lay-down area and maintenance facility. The warehouse will consist of a white, single-skin, heavy-duty polyethylene fabric on a galvanized steel framework. The floor of the building will be compacted gravel.

18.8.3 Maintenance Facility

The maintenance facility is a single-story building, with a total building area of approximately 15,300 ft^2. The building will be located near the process building and will consist of a flexible, open plan maintenance area configurable to the individual task with mobile equipment and separations (welding area, grinding, etc.). A dedicated electrical and instrumentation shop will be included. The building will be an insulated single-span pre-engineered metal building with sloped roof and external stormwater management.

The building will be equipped with roof exhaust fans to remove contaminated air from maintenance and welding shops. Large capacity make-up air units with natural gas fired heating coils will be used to supply make-up air. Natural gas fired unit heaters will be used to provide heating during winter. The maintenance facility will be equipped with an automatic wet sprinkler system, fire hose cabinets, and fire extinguishers.

18.8.4 Administration Building (Loadout Facility)

All administrative functions of the loadout facility will be located in a central modular trailer complex. The building will consist of office and reception space, a lunch area, and change room and washroom facilities.

Modular structures will be constructed and erected in conformance with local codes and regulations and be equipped with HVAC, plumbing, sanitary, and fire protection services to meet the functional requirements of each facility. Stormwater will be directed from the roof area and will be surface drained away from the building.

ITEM 19: MARKET STUDIES AND CONTRACTS

19.1 Market Studies

CRU International published a “Potassium Sulphate and Potassium Nitrates Market Outlook” in 2012 (CRU 2012) and the information on Market Studies in this TR is summarized from that document. This comprehensive study of the potassium sulfate market is drawn from CRU’s database, which uses a wide variety of sources. SOP is used on chloride-sensitive premium crops around the world including tree nuts, citrus, grapes, and other fruits, tobacco, or high-starch potatoes.

Worldwide capacity in 2011 was approximately 4.4 Mtpy of production. Worldwide demand was approximately 5.1 Mt in 2011.

Farmers and growers generally do not buy SOP as a standalone product. Rather, SOP reaches these users as a component of a balanced fertilizer blend, containing specific amounts of nitrogen, phosphorous, and potassium (NPK) nutrients. These blends are specifically formulated to meet individual crop and soil nutrient requirements.

In most cases, the distributor will have on hand the formulations most suited for the area and crops that distributor serves. Although a somewhat less common route, but still popular in Europe and some other parts of the world, SOP may reach the grower as a component in compound NPK fertilizers.

19.2 Global Markets

There are four major international markets that will be primary users of SOP including:

- Latin America and the Caribbean
- Asia outside of China
- Africa
- Europe

19.2.1 Latin America and the Caribbean

Latin America and the Caribbean will be the most important international market for ICP. In 2012, this region, including Mexico, imported more than 220,462 t of SOP. Demand for SOP in this region is projected to grow at a rate of 5.7% annually.

Mexico imported more than 52,911 t of the total SOP imported into Latin America. A large percentage of SOP imported by Mexico came from the US. Demand for SOP is anticipated to grow significantly as Mexico increases its position as a major supplier of fruits and vegetables to the US. ICP will be the nearest producer of SOP and can deliver by both rail and truck to customers south of the border.

Costa Rica and Colombia import significant quantities of SOP on a regular basis. Venezuela is an important importer of SOP in the Latin America-Caribbean region, but it’s political relationship with the US may impact ICP’s efforts.

Brazil is the largest grower of oranges and other citrus fruits in the world, but Brazil's demand for SOP is low. ICP intends to work with established distributors in Brazil to build a demand for SOP in this potentially very important market.

19.2.2 Asia Outside of China

Japan is a major importer of SOP with imports of about 112,436 t of SOP in 2011. Much of the SOP was supplied from the US. Indonesia and Malaysia are potentially good markets for ICP's SOP because of their production of oil palm, which is a heavy consumer of potassium.

19.2.3 Africa

Demand for SOP in Africa totaled about 171,960 t in 2011. One of the most important SOP consuming countries in Africa is South Africa, where arid soil conditions favor the use of SOP over MOP. ICP intends to work with established distributors in the African continent to build a demand for SOP in this potentially very important market.

19.2.4 Europe

Europe remains the largest consumer of SOP after China. The fertilizer market in Europe is mature, and fertilizer consumption may decline in the future as European economic conditions bring about changes in governmental agricultural support policies. In addition, Europe is also home to K+S KALI GmbH (K+S KALI) and Tessenderlo Chemie, two of the largest producers of SOP in the world. The European market will be difficult for ICP to penetrate because of logistic costs and the historical market dominance of K+S KALI and Tessenderlo Chemie. ICP will work with existing distributors to take advantage of potential Mannheim furnace retirements, deteriorating mining conditions, and increasingly restrictive environmental regulations.

19.3 The North American Market

The principal market for SOP in North America is those areas where high-value and/or chloride-ion sensitive crops are grown. Figure 19-1 illustrates the regions in the US where fruits, vegetables, tree nuts, and tobacco are important crops.

19.4 SOP Demand

Because there is only one producer of SOP in North America, statistical information about consumption of these materials is very limited. The Association of American Plant Food Control Officials (AAPFCO) collects and publishes information about the apparent consumption of certain fertilizer raw materials, NPK mixtures, and single-nutrient fertilizers in individual states. Because of different statistical methods in separate states, the tonnages reported for individual plant nutrients do not necessarily represent the total of each nutrient consumed in each state. The data reported by AAPFCO are the best available and do serve as a good indication of the market for fertilizers like SOP in the individual states. These statistics are reported in the AAPFCO publication *Commercial Fertilizers*. The most recent edition of this publication covers the year 2011.

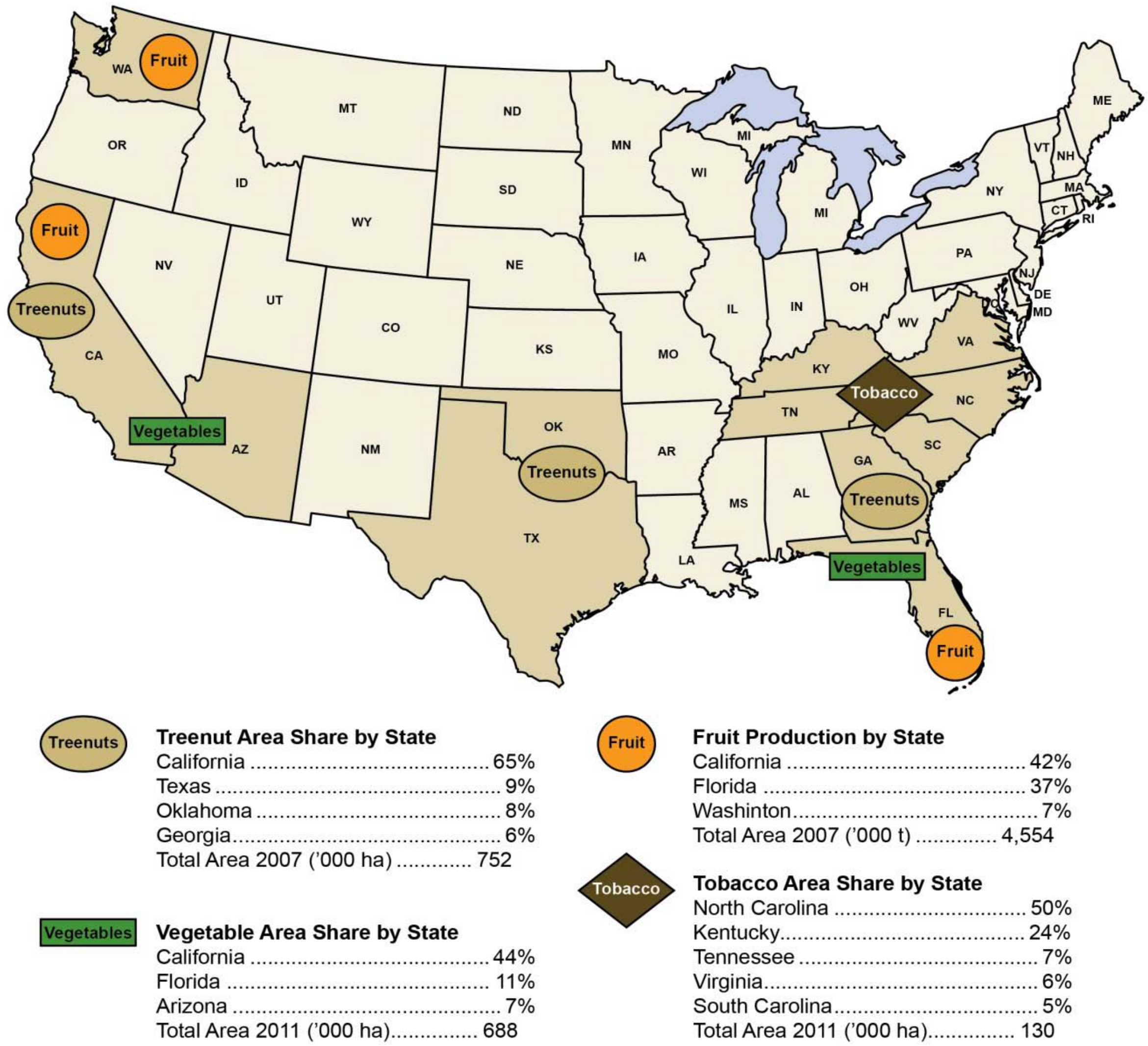


Figure 19-1. Some Areas Where High-Value Crops are Grown in the USA (CRU 2011)

AAPFCO reports that some SOP is used in almost every state in the US. Demand in the top ten states accounted for about 83.5% of US consumption of SOP in 2010, as shown in Figure 19-2. The relative ranking of these states has remained relatively unchanged over the past several years, including the distressed 2009 year, when the worldwide economic crisis impacted fertilizer demand everywhere. Regardless of changes in the other conditions driving the fertilizer market, there is absolutely no substitute for SOP in parts of the US and on some important, high-value crops. While overall consumption of SOP may vary from year to year, farmers in the US will always require a substantial amount of SOP fertilizers.

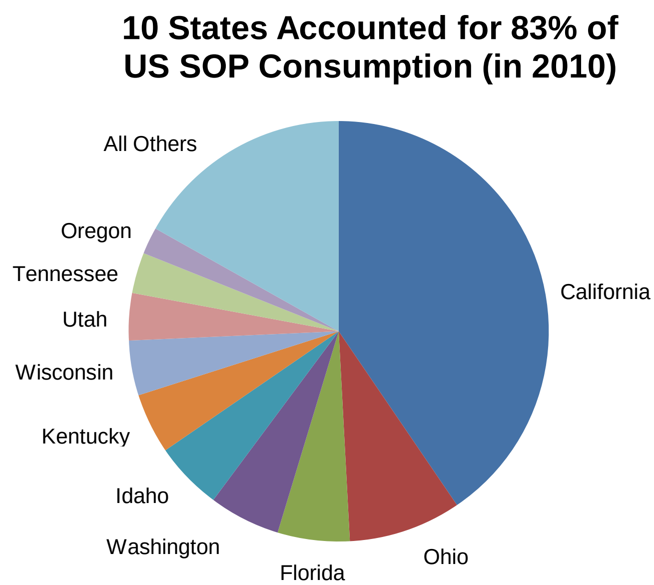


Figure 19-2. US SOP Consumption (AAPFCO Commercial Fertilizers 2011)

SOP use in the US is generally concentrated in those areas where high-value crops, food crops for human consumption, and/or crops with sensitivity to the chloride ion are grown or that have soil conditions that limit the use of fertilizers containing chloride ions. Important SOP consuming states and crops are listed below:

- California accounts for almost one-third of US SOP consumption because of its intensive agriculture, multiple growing seasons, production of a variety of fruits, tree nuts, and vegetables, and its widespread irrigated areas.
- Florida is the next largest SOP consumer, producing fruits and vegetables.
- Washington produces peaches, apples, pears, and cherries.
- Idaho grows potatoes and orchard crops.
- Michigan uses SOP on orchard crops.
- Ohio produces some crops that require SOP, but SOP is also used by fertilizer manufacturers (Scotts Miracle-Gro Company and the Turf & Specialty Group of The Andersons). SOP use on turf for golf courses, parks, and landscaping is an important market.
- Utah uses SOP because of arid soils. The Great Salt Lake Minerals Corporation (GSL), the only producer of SOP in the US, is located in Utah.
- North Carolina, Kentucky, Tennessee, and Virginia use SOP on tobacco crops.

Future SOP consumption in the US is anticipated to be strong, growing at a rate significantly greater than that of MOP. The future growth rate in SOP consumption in the US will be supported by increased consumer demand for more and higher quality fruits, nuts, and vegetables; growth in "fertigation" (the fertilizer application through irrigation systems);

increasing concerns about fertilizer runoff in stormwater; and more stringent environmental regulations for water quality in lakes, rivers, and streams.

19.5 Market Forecast

The market forecast for SOP depends on current and predicted new production, especially by primary producers. Worldwide SOP demand in 2011 (CRU 2012) was 5.1 Mtpy with the greatest demand from China. The rest of the world demand was approximately 2.9 Mtpy. The near-tem worldwide forecast for SOP demand is shown in Table 19-1.

Table 19-1. Near-Term SOP Demand ('000 tons)

	2011	2015	2020
Europe	1,127	1,187	1,159
North America	417	433	455
Central and South America	209	255	295
China	2141	2,314	2,921
Africa	221	248	261
Rest of the World (WOR)	722	896	990
Total	4,835	5,334	6,081

Demand in China is expected to rise, but because it is a closed market, its price influence is mitigated. European and North American demand will stabilize in the short term. Europe's demand is predicted to decrease, while that of the US will slowly increase. Demand in the rest of the world is expected to increase slowly.

19.6 Ochoa SOP Specifications and Value

The value of SOP is tied to MOP and SOP and is priced at a substantial premium (historically 30% to 50%) over the prevailing market price for MOP. SOP prices, based on projected grades, are freight on board (FOB) Jal, New Mexico ("FOB Jal") and net of other sales-related expenses. A.J. Roth and Associates, a US fertilizer consulting company with international expertise in potash and phosphates, provided pricing estimates by grades and receiving locations for the FS. The relevant SOP grades are standard, granular, and soluble. Upon full production of the estimated 714,400 tpy, the product mix is projected to be 229,400 t of standard SOP, 385,000 t of granular SOP, and 100,000 t of soluble SOP. The weighted average FOB Jal SOP price used in the financial model was $636/t. As reported in Green Markets, the average fourth quarter (Q4) 2013 granular SOP price was $680/t for California delivery. Granular SOP prices historically receive an average premium of approximately $50/t above standard SOP. During Q4 2013, ICP estimates the soluble SOP price was $740/t for Florida delivery.

19.7 Marketing Plan

As a new SOP producer, ICP will work strategically to gain a position in markets where its competition is and has been well established. Because there are currently only two sources of SOP supply in the US (Compass Minerals/GSL and product imported from overseas), many buyers of these products will welcome a new domestic source. ICP will develop a strong market position by supplying the very highest-quality SOP products and establishing a system to promptly and reliably deliver these products.

To ensure ICP's products enter the markets in an orderly and efficient manner, ICP has developed a marketing strategy that has two primary objectives:

1. Acquire a significant share of the SOP markets, while maintaining the historical premium price for these materials.
2. Expand the markets for these products beyond the traditional markets for non-chloride potash products and to new customers.

To achieve these objectives, ICP's marketing strategy has been built around three fundamental principles:

1. Provide its customers with the highest-quality SOP products available.
2. Deliver these products reliably and on time.
3. Maintain its customers with outstanding service and support.

There are three important factors that, together, play the most critical role in the buyer's decision to select one producer's SOP product over another's. These factors are as follows:

- Quality—Not only must the product from the supplier meet or hopefully exceed the industry's minimum chemical and physical specification guarantees, but its quality must remain consistent, shipment after shipment. The ability to provide premium quality products will be a particular advantage for ICP in competing with imported SOP, where repeated handling of the material has degraded product quality.
- Reliability—In today's highly competitive fertilizer market, a supplier must always be counted on to deliver product when, and as promised. The supplier must be prepared to promptly resolve any problems with the customer's product or delivery, if complications arise. Because ICP is so strongly committed to providing its customers with the best possible service, part of that commitment will be to promptly and effectively handle any and all potential problems pertaining to quality or delivery without hesitation.
- Service—The supplier must be competitive, and not just in price. As a new source of SOP, ICP will offer its products at prices that are competitive in the market and work diligently with customers to address their specific requirements. ICP's goal is to outperform its competitors in servicing its customers and in developing programs to support its customers. ICP will assist customers' sales efforts through cooperative advertising and/or promotional programs as appropriate.

Because ICP will be the first industrial resource mineral company to use a new raw material and technology to produce its SOP, ICP's strategy will not be to initially attempt to gain a major share of a customer's SOP requirement. Rather, it will be to obtain a portion of each customer's requirement.

In the early years, ICP will need to allay buyer's fears of abandoning historical supply sources and/or relying entirely on a new supplier. Once a customer develops confidence that ICP's products and services are better than traditional sources of supply, ICP will be in a position to capture a major share of a customer's requirement for SOP.

ICP's plan to enter the non-chloride potash market has been designed to work in concert with the company's plans to develop the Ochoa Project. The plan, its timing, and major milestones, may be summarized as follows:

2012

In 2012, ICP joined the most important fertilizer industry associations:

- The Fertilizer Institute (TFI) for the US
- The International Fertilizer Industry Association (IFA)

Senior ICP executives introduced ICP and the Ochoa Project to potential customers at the following key US industry meetings:

- TFI Annual Marketing Conference
- Southwest Fertilizer Conference
- TFI World Fertilizer Conference

ICP began collecting market intelligence, using its own sources.

2013

In 2013, ICP's senior executives attended the most important industry meetings and expanded its contacts with potential customers in the US and overseas.

- TFI Annual Business and Marketing Conference
- Southwest Fertilizer Conference
- TFI World Fertilizer Conference

ICP continued to expand its collection of market intelligence.

2014

In 2014, ICP's senior executives will continue to attend key industry meetings both in the US and overseas.

- ICP will continue collection of market intelligence.
- ICP will begin to look for in-market storage facilities.
- ICP will begin the selection of marketing staff and overseas sales agents with the objective of hiring some key personnel by the end of 2014.
- ICP will initiate regular contacts with potential customers and begin sales contract discussions.

2015

- ICP senior executives and marketing personnel will attend key industry meetings.
- ICP will join state and regional fertilizer and grower associations in its most important marketing areas (e.g., California, Florida, Virginia, the Carolinas, etc.).
- ICP will complete the hiring of marketing staff and, with the cooperation of its partner, Yara, the appointment of overseas sales agents.
- ICP will finalize sales contracts or agreements with customers beginning in 2017.
- ICP will finalize product storage agreements with in-market warehouse operators for the years beginning in 2016.

- ICP will negotiate rail freight contracts for delivery of its products to its in-market storage and/or customer locations.
- ICP will develop its advertising and sales promotional programs for implementation in 2016.
- ICP marketing and production departments will prepare to deliver SOP to its partner, Yara, and its other customers in the US and overseas.

2016

- ICP begins to develop new markets and find new customers for its products.
- ICP begins planning for the production and sale of SOPM products.

2017

- ICP enters the market for SOP

19.8 Contracts

ICP has a committed off-take agreement with Yara. Under this agreement, ICP will sell to Yara and Yara will buy from ICP 30% of all products produced by the Ochoa Project annually. The term will begin upon the commencement of commercial production for a period of 15 years and will automatically extend every 5 years thereafter unless either party elects not to extend. All products will be sold to Yara based on market prices.

19.9 Transportation

19.9.1 Distribution from the Ochoa Mine

The ICP Ochoa mine and processing operation will be located in southeastern New Mexico, away from any export port or domestic fertilizer market area. Product will be shipped from the mine, by rail, to a port on the Gulf Coast for shipment overseas or by rail or truck to the North American market. Although rail service is not available at the Ochoa mine site, a railhead is located about 22 miles away from the mine, near the town of Jal, New Mexico. ICP plans to ship its SOP by truck to an ICP storage and shipping facility, which will be constructed at the railhead near Jal. The ICP products will then be loaded into rail cars or trucks for shipment to their final destinations. A diagram of ICP's product movements is shown on Figure 19-3.

The Jal, New Mexico, location is served by the TNMR short line railroad. The TNMR connects with the Union Pacific/Southern Pacific Railroad (UPSP) at Monahans, Texas. From Monahans, the UPSP can transport ICP products to the Texas International Terminals at the Port of Galveston (Galveston, Texas) for waterborne shipments and to locations throughout ICP's North American market area.

Rail deliveries, whether to the domestic market or to the terminal at Galveston, Texas, will be made in multiple covered hopper car units. Shipments of as few as 5 or 10 train cars to a single inland destination can be expected as routine, while shipments to the terminal at Galveston, Texas, will be made in unit trains of 85 to 92 rail cars.

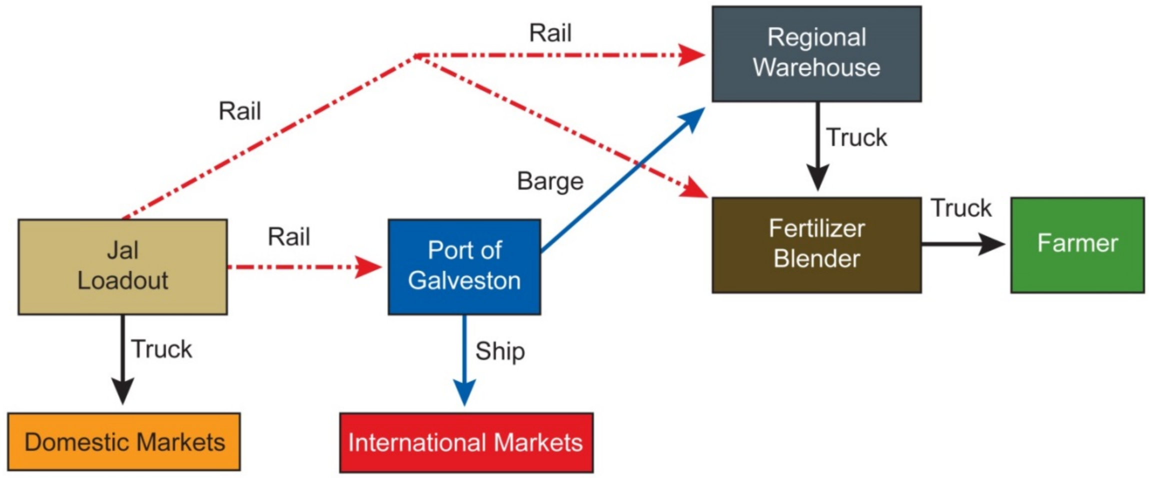


Figure 19-3. ICP's Product Movement

For international shipments, SOP will be shipped by rail from the storage and loadout facility at Jal, New Mexico, to the terminal at the Port of Galveston, where it will be loaded onto vessels for transport overseas. The exception to this will be product going to Mexico; with SOP transported by rail (or by truck), directly from Jal to customers in Mexico.

In North America, SOP may be shipped directly to the domestic buyer's receiving location; or if the buyer's facility is located on a major river, ICP product can be shipped to the terminal at Galveston, loaded on barges, and transported via the inland waterway system to the buyer. Because prompt product availability during the fertilizer season is of critical importance, and many fertilizer blenders and distributors are no longer served by rail, much of ICP's products destined for sale in the domestic market will be shipped by rail to storage locations and then re-shipped by truck to ICP customers as needed. These in-market storage facilities will be leased by ICP on an annual or longer-term basis. The locations of these storage facilities will be determined based on ICP's customer requirements, but ICP intends to have storage facilities in all of the important SOP markets.

19.9.2 Distribution in the Market

There are several ways that the SOP produced by ICP will reach a local blending operation, where it can be combined with other plant nutrients and then delivered to a farm, orchard, or nursery. Fertilizer blending plants generally serve local markets within a 50- to 100-mile radius of the blending plant location. The majority of these blending plants are located in places that rely on trucks to deliver fertilizer materials from regional or nearby warehousing facilities. These regional warehousing operations may be owned and operated by private companies, farm cooperatives, or large agricultural input marketing and distribution organizations.

International and domestic fertilizer trading companies have frequently established warehouses to receive and store product in advance of the application seasons. Finally, there are large fertilizer blenders that often have warehouses and will store product for others as well as for their own use.

Regional and national distribution operations located in key market areas will be important in providing local storage and market outlets for ICP's SOP products.

19.10 Netback Price Forecast

Prices used for the economic analysis are forecast for SOP FOB the loading facility at Jal, New Mexico, and net of sales-related expenses. The netback sales prices are shown in Table 19-2.

Table 19-2. SOP Price Forecast per Ton Plant Gate at Jal

Year	Price per Ton (USD) Plant Gate at Jal		
	Standard Grade SOP	Granular Grade SOP	Soluble Grade SOP
2017	540	586	631
2018	511	557	602
2019	522	568	613
2020	539	585	630
2021	569	614	660
2022	575	621	666
2023	581	627	672
2024	594	639	685
2025 and beyond	607	652	697

ITEM 20: ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

The history section presented herein references and is extracted in part from Gustavson's (2011d) December 30, 2011 NI 43-101 TR and its April 2012 PFS (2012), with additions and updates extracted in part from ICP's and INTERA's (2012, 2013a) work that was presented in the January 2014 SNCL Ochoa Project FS (2014).

Several federal and state mine health and safety laws and regulations and environmental laws and regulations govern the construction activities and operation of the Project.

20.1 Mine Safety and Health Regulations, Permits, Plans, and Approvals

MSHA has federal regulatory oversight at the Project for matters concerning safety and health. Title 30 CFR, Subchapter G, Parts 40 through 49 covers the filing and other administrative requirements, including those governing independent contractors doing work at the Project site. Subchapter H covers education and training requirements in Parts 47 to 49 for the Project. Subchapter I covers the reporting requirements for accidents, injuries, illnesses, employment, and production. Subchapter K, Parts 56 to 58, covers the regulation of metal and non-metal mine safety and health. Subchapter M, Part 62 covers noise exposure limits and hearing conversation programs. Subchapter P, Part 100 outlines the criteria and procedures for assessment of civil penalties for violations of the Federal Mine Safety and Health Act of 1977 (United States Congress 1977). Subchapter Q, Part 104 covers patterns of violations assessments. (All the above Title 30 subchapters and parts are available in MSHA 2013b.)

Table 20-1 lists most of the required MSHA plans and submittals required for the Project. Several of these are required of the operator or contractors prior to commencement of any construction activities. Title 30 CFR Parts 1–199 provides the complete details of the rules and regulations.

Table 20-1. Listing of Required MSHA Plans

30 CFR Part	Plan or Submittal
41	Legal entity operating mine
47	Hazard Communication (HazCOM) program
48	Training plan
49	Mine Emergency Notification Plan
49	Escape and Evacuation Plan
57	Notification of commencement of operations
57	Mine Ventilation Plan
57	Rock Burst Control Plan (if rock burst occurs)
57	Escape and Evacuation Plan (includes firefighting plan)

The state of New Mexico has a mine safety office and specific limited regulations pertaining to underground mine health and safety. New Mexico statutes governing mining in the state can be found in *New Mexico Statutes Annotated 1978*, Chapter 69 Mines (NMSA 2011). New Mexico regulations can be found in the New Mexico Administrative Code (NMAC 2014), Title 19, Chapter 6. Pertinent regulations pertaining to plan submittals include the requirement for an Emergency Notification Plan in 19.6.2 NMAC. Potash mining is exempt from both the *New Mexico Hardrock Mining Act* and the *New Mexico Surface Mining Act* (State of New Mexico 2005) and is therefore not required to obtain mine closure and close-out permits. However, the Mining and Minerals Division of the New Mexico Energy, Minerals, and Natural Resources Department registers all mines (including potash mines, borrow pits, and sand and gravel mines), mills, concentrators, and smelters prior to start-up of the mining operation, as described in Table 20-2.

Table 20-2. New Mexico Mining Regulations

Permit	Statute	Agency	Timing	Progress
Mine registration	NMAC 19.7.1	Mining and Minerals Division of the New Mexico Energy, Minerals, and Natural Resources Department	3 months, but not approved until after ROD on the EIS	Application will be submitted 6 months prior to operation

20.2 Environmental Studies and Permits

Table 20-3 lists the environment permits, leases, and approvals necessary to construct and operate the Project. Of the several environmental laws governing the Project, the National Environmental Protection Act (NEPA) is the nation's broadest environmental law. With a portion of the Project's mineral permits located on federal lands, the Project must comply with NEPA as a federal decision will be made on whether to approve the leases and ROW's necessary to construct the Project. Under NEPA requirements, an EIS must be prepared under the direction of the BLM. The EIS is a systematic, scientific, and interdisciplinary approach used to predict the potential beneficial and adverse impacts of the Project on the environment. The BLM and ICP entered into a MOU effective August 29, 2011 for the preparation of a Draft EIS (DEIS) and a FEIS. The DEIS was completed in August 2013 and the public comment period has closed. The FEIS ROD is expected to be published in March 2014, and the public protest period for the ROD closes in April 2014. If a favorable ROD is made, the granting of the PRLs is expected to follow shortly. Table 20-4 shows the schedule for the remaining phases of the EIS process.

Several supporting studies were conducted by ICP in support of the EIS analysis. These included:

- Groundwater
- Surface water
- Air
- Soil
- Ecological
- Cultural resources

Groundwater—Table 20-5 lists the water consumption requirements for the Project. As discussed in Item 7.6, the water supply will be from the Capitan aquifer.

Surface Water—Walsh (2013a) evaluated the ephemeral steams located from the Project plant site area to the loadout area. A *Request for Jurisdictional Determination of Four Drainages in Lea County, New Mexico* (Walsh 2013a) was made to the US Army Corps of Engineers (USACE). The USACE reviewed the studies and, in accordance with Section 404 of

Table 20-3. Environmental and Other Regulatory Leases and Approvals

Permit	Statute	Agency	Timing	Progress
Air permit to construct (Prevention of Significant Deterioration [PSD] permit)	NMAC 20.2.72	Air Quality Bureau (AQB) of the New Mexico Environment Department (NMED)	30 day administrative review of application followed by 180 day technical review of application	Permit application was determined to be administratively completed by NMED AQB on December 12, 2013
Air permit to operate (PSD permit)	NMAC 20.2.74	AQB, NMED	6 months	Application will be submitted during construction
Federal Potassium Prospecting Permits	43 CFR Ch. 11	BLM	6 months	28 obtained; exploration on seven new permits remain
Federal PRLs	43 CFR Ch. 11	BLM	6 months	26 applications made; exploration on seven new permits remain
State trust land mineral leases	NMAC 19.2.3	Commissioner of Public Lands of the NMSLO	1 year	All New Mexico leases obtained
State trust land water exploration permit	NMAC 19.2.10	NMSLO	1 month	Obtained
State trust land ROW easement	NMAC 19.2.10	NMSLO	2 months	Application to be submitted in October 2014 following expiration of exploration permit
Notice of Intention to Drill Wells to Appropriate Nonpotable Groundwater	NMAC 19.1.2	New Mexico Office of the State Engineer (NMOSE)	6 months	Completed
NMED groundwater discharge permit	NMAC 20.6.2	Ground Water Quality Bureau of the NMED	1 year	Application will be submitted in 2014
Mine drill holes that encounter water—Application for Permit to Drill a Well with No Consumptive Use of Water; Well Plugging Plan of Operations, and Artesian Well Plan of Operations	NMAC 19.27.4	NMOSE	2 months	Obtained
Section 404 Wetlands and Section 401 Water Certification permits	33 CFR 331.2	USACE	9 months	Obtained Jurisdictional Determination – permits not needed
National Pollutant Discharge Elimination System stormwater permit: construction and operation	40 CFR 122	US Environmental Protection Agency	1 year	Obtained Jurisdictional Determination – permits not needed

Table 20-4. Schedule for Remaining Phases of the EIS Process

Task	Start Date	Time Required
DEIS Comment Analysis and Response	August 2013	Completed
Finalize EIS	October 2013	4 months
Publish FEIS	February 2014	1 day
Publish ROD	March 2014	1 day
End Protest Period for ROD	April 2014	1 day
Issue Grant and PRLs	April 2014	1 day

Table 20-5. Water Consumption Assuming a Feed of 25,000-mg/l TDS at 75% Recovery from RO Treatment

Water Type	Flow Rate (gpm)
Produced Water from Well Field	2,714
Raw Water	1,053
RO Water	1,661*
Potable Water	40
Mine	160
Salt Wash	116
Leach Circuit	893
Boiler Makeup	8
Reagents	8
Waste	435

*Note: Due to rounding, RO flow rates may not add up. mg/l = milligrams per liter.

the Clean Water Act and Section 10 of the Rivers and Harbors Act of 1899 (US Environmental Protection Agency 2013a), determined that there were no *Waters of the US* in these Project areas.

Air—ICP completed a preliminary emissions inventory (Class One Technical Services Inc. [COTS] 2012) to describe the potential maximum emission rates for certain gases and particulate matter. Calculated emission rates were developed based on equipment specifications for processing polyhalite mined at the Project. New Mexico Environmental Department Air Quality Board (NMAQB) accepted the modeling and determined that no baseline studies would be required for the Prevention of Significant Deterioration application. ICP submitted the Air Permit to Construct in November 2013 and the permit application was deemed administratively complete by the NMAQB in December 2012.

Soil—Native soil surface conditions within the Project area consist of relatively flat terrain with minor arroyos and low-quality semi-arid rangeland. Windblown sand dunes and limited bedrock exposures, caliche, and poorly developed soil horizons are the predominant soil features found on the Ochoa Project site.

The Project area contains 26 different soil associations, complexes, or map units consisting predominantly of fine sands and loamy fine sands. The top soil is caliche rubble and windblown sand. The northern portion of the Project is situated in sandy dune country. The soils are predominantly well drained, not very susceptible to water erosion, and highly susceptible to wind erosion. Most soils have a moderate restoration potential, but precipitation and soil depth are limiting factors to restoration.

Ecological—Walsh (2011, 2012a, 2012b, and 2013b) conducted baseline vegetation and wildlife surveys in the vicinity of the Project area in 2011 and 2012 and along the water pipeline ROW in 2013.

Vegetation surveys included recording general observations of plant communities and their dominant species and ground-truthing landfire geospatial vegetation data to create a vegetation map. The Project area contains six vegetation communities, including coppice dune and sand flat scrub, creosote desert scrub, mesquite shrubland, mesquite upland scrub steppe, mixed desert scrub steppe, and shinnery oak shrubland. These communities comprise essentially the same mix of shrub, herb, and grass species, with different combinations of dominant shrub and grass species differentiating community types.

Wildlife data and information was summarized from Walsh (2011, 2012a, 2012b, and 2013). A literature review was conducted as part of the wildlife survey to inform biologists of species that may be encountered on-site. Wildlife surveys were primarily observational and were conducted by vehicle and on foot, depending on accessibility. Habitat types known to provide forage, water, shelter, nesting areas, or thermal protection were identified and surveyed. Wildlife signs such as nests, scat, tracks, and burrows observed during the survey were noted. Wildlife observations were recorded with a Trimble Global Positioning Satellite receiver unit. In 2012, the BLM requested additional surveys for reptiles and ungulates. Additionally, bat acoustical monitoring was conducted for 6 months from May through October 2012.

Surveys were conducted for:

- Lesser Prairie-Chicken leks
- Raptor nests
- Reptile pitfall traps
- Ungulate pellets
- Acoustical bats

Wildlife habitat is poor and does not support a diverse or unique wildlife population. Migratory birds and raptors are present throughout the area. There was no wildlife observed at the loadout facility. No threatened or endangered species were observed.

Cultural Resources—ICP conducted Class III Cultural Resource surveys of all exploration core hole locations and the proposed processing, shaft, and loadout areas.

These studies identified three sites that require mitigation or avoidance at the processing area and one at the shaft area. Additionally, six sites at the loadout facility were identified for further study. Based on the updated Jal loadout design, the six sites at the Jal loadout facility will be avoided and will not require additional mitigation. The site near the shaft area will also be avoided through project design.

ICP received approval from the BLM and State Historic Preservation Office (SHPO) of the treatment plant for the three identified sites in the processing area. Data collection at these sites is underway and will be completed before construction starts.

20.2.1 Waste Management

The use of hazardous substances will be limited to only those necessary for the safe operation of the mine and processing facilities. All hazardous substances will be inventoried, used, stored, controlled, and disposed of in accordance with all applicable regulations. Anticipated hazardous substances that could be present on-site during construction, mining, or reclamation activities include vehicle and equipment fluids, cleaners, and roadway treatment chemicals.

ICP will not produce or discard any *Resource Conservation and Recovery Act* (US Environmental Protection Agency 2013b) listed hazardous materials. Non-hazardous binders, flocculants, and dust suppressors will be added to the process as necessary.

Solid waste will be generated during construction, mining, and reclamation activities from office supplies, paper products, laboratory supplies, and other non-hazardous sources. These solid wastes will be disposed of in an appropriate waste disposal facility. Cleanup of spills may generate wastes such as soil, sorbent materials, and personal protective equipment. These wastes will be containerized and disposed of in an appropriate disposal facility.

20.2.2 Reclamation

Reclamation activities are proposed in accordance with relevant closure regulations and standards. Reclamation activities will return the site to pre-project land uses, which include rangeland and ranching and hunting, unless otherwise specified by the BLM.

Reclamation activities at the Ochoa Project will begin after production ends with the exception of concurrent reclamation (i.e., tailings, areas that will not be disturbed further). Mining activities and associated reclamation on BLM administered lands are governed by 43 CFR 3590 (MSHA 2011).

The dry stack tailings pile will be left in place and integrated into the landscape to the maximum practical extent. The final land form will be mechanically stable, promote successful revegetation, prevent wind and water erosion, and maximize visual compatibility with the surrounding land forms. The dry stack tailings will be reclaimed, to the extent possible, during mining operations.

Mine closure and reclamation costs are estimated to be in the range of $15 million in 2013 USD.

20.2.3 Community Impact

Southeastern New Mexico has a long history of petroleum and potash mining. Both industries are major contributors to the regional economy and support many satellite industries. There is local and regional support for the Project: the state of New Mexico and Lea County are supporters of development of the mining industry. Lea County and surrounding communities stand to benefit significantly from the Project, including the creation of approximately 400 direct permanent jobs and the payment of new tax revenue to the state and county.

Wage rates for miners range from $21 to $30 per hour. Annual labor and benefit payments for mining and processing will inject over $33 million into the regional communities, primarily those in Lea and Eddy Counties, New Mexico.

ITEM 21: CAPITAL AND OPERATING COSTS

21.1 CAPEX

21.1.1 Summary

The construction cost of the Project (CAPEX) is estimated to be $1,018 million expressed in September 15, 2013 USD, un-escalated. The CAPEX for the Project is shown in Table 21-1.

Table 21-1. Total Estimated CAPEX by Major Area (in millions USD)

Work Breakdown Structure	CAPEX Total	CAPEX % of Total
Mine Infrastructure and Development	$107	10%
Process Plant	$527	52%
Jal Storage/Loading	$37	4%
Total Direct Costs	$671	66%
EPCM Services	$99	10%
Construction Indirect	$21	2%
Freight, Spare, First Fill, etc.	$35	3%
Total Indirect Costs	$155	15%
Owner Costs	$80	8%
Total Project Contingency	$112	11%
Project Total	$1,018	100%
EPCM = engineering, procurement, and construction management		

The following costs are excluded from the estimated Project CAPEX:

- $114 million of mining and surface mobile equipment will be leased for the initial 5 years of operations and is included in the OPEX.
- An un-escalated, deferred capital cost of $27.2 million for the second granulation train, which is required to increase granular product tonnage as market demand is established, is planned to come online and begin ramp-up in Month 19 (third quarter of 2018). This cost is included in the Sustaining Capital total and is reflected in the Economic Model.
- Risk evaluations or any allowances for risk, including business risk, schedule risk, event driven risk or commercial risk were excluded.

The CAPEX has an intended accuracy of ±15% and is consistent with the standards for a feasibility level estimate defined by the Association for the Advancement of Cost Engineering International for a Class 3 estimate.

21.1.2 Direct CAPEX

21.1.2.1 Mine Infrastructure and Development—AAI is responsible for the mine CAPEX estimate. Mine CAPEX estimate was developed from the list of equipment and infrastructure necessary to produce the ore at the rates designed in the 50-Year Mine Plan.

Cost estimates are based on a number of sources including formal quotations, budgetary quotations, engineering estimates, and allowances. Engineering estimates with respect to the mine are based on a design sufficient to provide the degree of accuracy required. Allowances are based on factored costs of similar equipment and systems recently quoted or constructed.

The estimated mine capital investment is necessary to achieve and sustain the design capacity of an average 3.7-Mtpy mine operation. The initial direct capital investment for the mine is $107 million. The bulk of the mine CAPEX is made up of the engineer, procure, and construct (EPC) contracts for the shaft and the slope development. To minimize initial project capital requirements, approximately 57% of the value of mining equipment required during 2016 and 2017 is leased on 5-year terms, with equipment buyback at the end of the lease term at 25% of the equipment's purchase cost. Lease payments have been calculated and included as an OPEX. A summary of direct CAPEX for the mine life is included in Table 21-2.

21.1.2.2 Process Plant, Infrastructure and Loadout Facility—SNCL is responsible for the CAPEX estimate of the process plant, infrastructure and loadout facility located at Jal. The scope was organized into preliminary commitment packages. Whenever possible, requests for quotations were sent to at least three vendors for each package. A summary of the estimated direct CAPEX of the process plant and loadout facility by major area is shown in Table 21-3.

The following categories of cost information were used in the CAPEX estimate:

- Firm quotations (the crystallizer is the only package identified as a firm quotation)
- Budget quotations sought for packages over $500,000
- Email quotations sought for packages under $500,000
- Estimated prices based on SNCL's extensive records of historical costs
- Allowances for those items known to exist but material take-offs were not available, based on SNCL's knowledge and experience

Local contractors provided unit installation hours and rates for civil, concrete, steel erection, process piping, and architectural work. Approximately 63% of total labor hours are based on direct input from contractors.

Veolia is responsible for an island design concept for the evaporator and crystallizer portion of the process plant which included the price of the equipment only. Layne provided an EPC-based quotation for the water supply and treatment system, including the RO WTP, from which the design was incorporated..

21.1.3 Indirect CAPEX

21.1.3.1 EPCM Services— ICP plans to execute the Project with EPC contracts for the mine construction and water supply and treatments systems, Veolia for engineering and procurement of the crystallization circuit, and an EPCM contract for the balance of the Project.

SNCL's estimate of $99.0 million for EPCM services are based on the list of project deliverables, staffing requirements based on the project schedule, travel costs, construction field expenses, and on SNCL's knowledge and experience.

The engineering and design fees for mining, and the water supply and treatment system were provided by the respective consultants and are included in the indirect cost estimate. Veolia carries its design fee in its estimate.

Table 21-2. Mine Direct CAPEX Summary

Area	Total (in millions USD)
General Site – Mine	$13.1
Mine – Shaft and Slope	$90.8
Ancillary Buildings – Mine	$1.5
Off-Site Facilities – Mine	$1.3
Total Mine	$106.7

Table 21-3. Process Plant and Loadout Facility Direct CAPEX

Area	Total (in millions USD)
General Site – Process Plant	$78.7
Tailings Facility	$38.3
Process Plant	$392.2
Product Loadout	$10.3
Ancillary Facilities – Process Plant	$7.4
Total Process Plant	$526.9
General Site – Jal Loadout	8.7
Jal Storage/Loadout Facilities	28.6
Ancillary Facilities – Jal Loadout	0.2
Total Jal Loadout Facility	37.5

21.1.3.2 Construction Indirect—The following scope, totaling an estimated $21.0 million, will be provided by the EPCM contractor and is included in the line item cost “Construction Indirects” included in Table 21-1:

- EPCM on-site offices
- Personnel and awards for safety programs
- Emergency response team
- Site maintenance staff and equipment
- Furniture, office equipment, and supplies for the facilities directly controlled by the EPCM and owner’s teams
- Temporary generators, fuel storage
- Temporary on-site water storage facilities
- Surveying and testing services
- One 150-t crane for heavy lift
- Allowance for temporary signs, jersey barriers, and general traffic control
- Sanitary and sewer services
- Hard hat and small tools allowance for EPCM personnel, owner’s team, and visitors
- Mobile equipment required for EPCM staff including fuel maintenance, insurance costs
- Allowances for courier services
- Temporary communications, including telephone and fax services, and site radios
- Temporary construction power
- Temporary construction roads

- Temporary fencing for construction areas

The following temporary construction services are included in the owner's cost:

- Site security staff
- Materials management personnel
- Materials management warehouse, and warehouse mobile equipment
- Janitorial staff and supplies

21.1.3.3 Other Indirect Costs—Other indirect costs, estimated at $35.0 million and shown under "Freight, Spare, First Fill, etc." in Table 21-1 include:

- An allowance for start-up assistance during the initial start-up of the plant
- All commissioning costs are included in owner's cost, including any requirement for vendor's representatives, and further assistance from engineering, other consultants, or outside agents
- Cost of vendor representatives included in the CAPEX estimate is intended to be sufficient to bring the Project to mechanical completion, and does not include any vendor's representatives who may be required by the owner for training or commissioning
- Cost of spare parts including first year initial spares, commissioning spares, and crystallizer package critical spares
- At the request of ICP, second year spares are included in the operating cost estimate (OPEX)
- Cost for first fills including on-site diesel fuel tank fills, reagents, hydraulic fluids, oils, lubricants, glycol, and other fills
- Cost for first fills was estimated using approximate quantities, and unit rates are based on SNCL's knowledge and experience
- Crystallizer package first fills information was provided by Veolia
- Freight cost are carried in the freight Work Breakdown Structure of the indirect costs for process equipment, all piping bulks, all electrical bulks and equipment, and all instruments and instrumentation bulks

21.1.4 Owner's Cost

Owner's cost represents those costs incurred during construction that are outside the scope of the EPCM contract or other Project construction contracts. Table 21-4 summarizes the estimated owner's cost that was supplied by ICP. Notable exceptions to this estimate include project financing costs and contingency.

21.1.5 Contingency

Sufficient contingency has been added to achieve a 50% probability that the final costs for constructing the Project will not overrun the budget. A 14% contingency was calculated on all direct and indirect costs and the construction cost contingency amounts to $97.7 million. An additional mining contingency of $14.6 million has been added to the construction cost contingency for a total project contingency of $112.0 million as shown in Table 21-1.

Figure 21-1 shows the probability of the expected final costs of constructing the Project. The mining contingency of $14.6 million and the owner's costs of $80.2 million have been excluded from the probability calculation. The X-axis indicates the possible final costs of the Project while the Y-axis shows the percentage of probability that this cost might occur.

Table 21-4. Owner's Cost Summary

Description	Amount (in millions USD)
Taxes and Duties	$40.3
Owner's Team	$17.1
Owner's Consultants and Related Expenses	$7.5
Project Insurance	$7.2
Training and Other Staffing Costs	$0.9
Land Acquisitions and Payment	$0.7
Other	$6.5
Total	$80.2
Note: Due to rounding, amounts do not add up.	

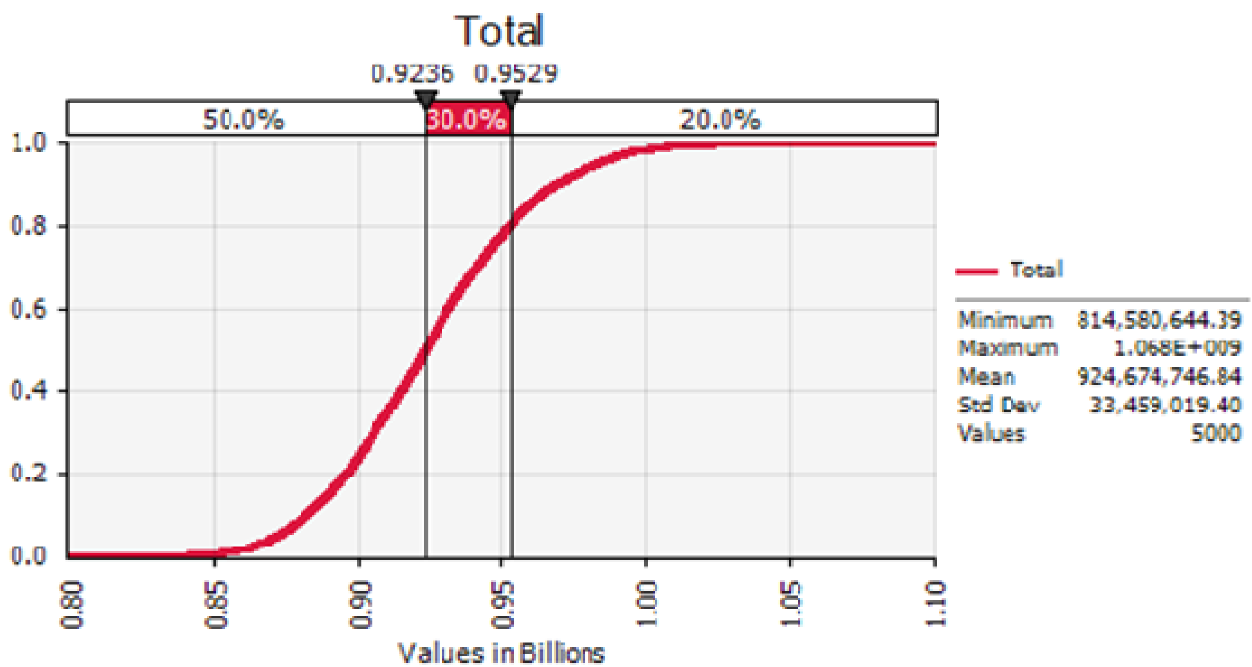


Figure 21-1. Final Project Cost Probability Curve

21.1.6 Cash Flow

The estimated total CAPEX of $1,018 million is expected to be spent during the period starting from April 2014 and ending in June 2017. Table 21-5 summarizes the estimated expenditure of funds on a yearly basis based on the project schedule. Construction is anticipated to take 30 months. The CAPEX does not include any financing fees.

Table 21-5. Estimated CAPEX Cash Flow (in millions USD)

	2014	2015	2016	2017	Total
Equipment	30.5	131.0	68.7	5.5	235.8
Bulk Materials	6.7	68.5	43.0	2.1	120.3
Freight	0.3	4.7	1.8	0.0	6.9
Labor	19.0	139.6	166.2	19.9	344.6
Commissioning	0.0	0.1	2.3	6.8	9.2
Temporary Facilities	8.6	12.0	0.0	0.0	20.6
EPCM	15.0	30.6	28.6	6.7	80.9
Contingency	5.3	11.2	87.1	16.1	119.7
Owner’s Cost	12.3	28.2	32.7	7.1	80.2
Total	97.7	425.9	430.4	64.2	1,018.2
Cumulative	97.7	523.6	954.0	1,018.2	

21.1.7 Sustaining CAPEX

21.1.7.1 Total Project General Sustaining CAPEX for the Life of the Project—Total sustaining CAPEX for the life of the Project has been estimated to be approximately $1.407 billion as shown in Table 21-6. The breakdown between the surface facilities and the mine sustaining CAPEX is itemized in the items 21.1.7.2 and 21.1.7.3.

21.1.7.2 Surface Facilities Sustaining CAPEX—ICP has estimated surface sustaining CAPEX consists of $423.7 million for maintenance of the plant, Jal loadout and the water treatment facilities. The second granulation plant is estimated at $27.2 million. Three additional magnesium sulfate ponds with a total cost of $9.3 million will be added in the first 3 years of operation. The TSF will be expanded in 20 phases, every second year, until reaching final capacity for a total of $115.2 million (all costs include contingency).

21.1.7.3 Mine Sustaining CAPEX—AAI has estimated underground equipment and facilities sustaining CAPEX at $357.9 million, and the mine surface facilities at $103.4 million. Sustaining CAPEX for major maintenance and rebuilds are estimated at $270.1 million. Mine freight, engineering and design and contingency are estimated at $100.55 million.

21.2 OPEX

21.2.1 Summary

All costs are in 2013 USD unless otherwise noted. OPEX estimates do not include other taxes and royalties. Transportation costs are for the transport of the product to the loadout facility in Jal, New Mexico. Costs to transport product from the loadout facility to port or final market have been included as a reduction of revenue in determining net-back prices.

Equipment OPEX estimates for the mine and processing plant include maintenance parts, lube, tires, wear parts, supplies, and diesel fuel where applicable. Electricity costs and labor were tracked separately from the equipment OPEX estimates. Maintenance and operating staff were included in the staff and personnel detail. The OPEX were determined based on the annual production rate of 714,400 t of SOP. Cost per ton of finished product is based on total mineral production.

Table 21-6. Project Sustaining Capital

	Cost (in millions USD)
Surface Sustaining Capital	
Direct Capital Cost (Non HPD)	256.1
Direct Capital Cost HPD	89.0
Direct Capital Cost JAL Loadout	23.8
Direct Capital Cost for RO Water system	20.6
Plant Maintenance Labor	34.2
Second Granulation train	27.2
Magnesium Sulfate Pond Construction	9.3
Gypsum Stack Storage	115.2
Total Surface Sustaining Capital (including contingency)	575.2
Mine Sustaining Capital	
Underground Equipment and Facilities	
Production Equipment – Section	
Outby Mobile Equipment	316.0
Other Underground Equipment/Facilities	34.5
Underground Ore Handling Equipment	4.33
Underground Electrical, Communications and Monitoring	3.0
Ventilation	
Total Underground Equipment and Facilities	357.9
Surface Facilities – Mine	
Shaft and Slope	91.1
Surface Electrical and Communications Systems – Mine	9.3
Other Surface Facilities – Mine	2.60
Surface Mobile Equipment – Mine Site	0.41
Total Surface Facilities – Mine	103.4
Capitalized Major Maintenance/Rebuilds	270.1
Total Underground and Surface Capital (Mine)	731.4
Freight	32.35
Engineering and Design, Other Indirects	0.9
Cost Contingency	67.3
Total Mine Sustaining Capital	831.9
Total estimated sustaining CAPEX for the life of the Project = 1,406.3 million.	

Table 21-7 details the steady-state OPEX for the Project. Steady state has been defined as the operating years from 2022 through 2065. These years generally exclude major one-time OPEX and events that occur early in the Project and as the Project winds down. Years 2017 through 2021 include effects such as lower SOP production during ramp-up, equipment leasing expenses, initial receding face expenditures, and inventory variations. Year 2066 reflects only a partial year of production once ore reserves are exhausted. Major component rebuild costs are not included within the OPEX estimate because these items are capitalized.

The following nominal quantities in Table 21-8 were used in determining the OPEX estimate of the Project. The quantities used are based on an average full production year.

Table 21-7. Steady-State Average Annual OPEX (USD)

OPEX	2022–2065 Cost (millions)	Average Annual (millions)	Cost/ton of Ore	Cost/ton of Product
Mining	$2,475.8	$56.27	$15.13	$78.76
Processing	$3,389.0	$77.02	$20.72	$107.82
General and Administrative	$267.4	$6.08	$1.64	$8.51
Total OPEX	$6,132.3	$139.37	$37.49	$195.09

Table 21-8. Average Annual Operating Quantities

Description	Quantity
Polyhalite Ore (tpy)	3,700,000
Mine Electricity (kWh/year)	210,000,000
Processing Electricity (kWh/year)	494,220,000
Natural Gas (MMBTU/year)	7,801,600
SOP Production (tpy)	714,400

MMBTU = million British Thermal Units

21.2.2 Mine OPEX

Mining OPEX was estimated by breaking down the costs into the following areas:

- Mine labor
- New Mexico wage credits
- Equipment leasing
- Operating supplies
- Mine maintenance
- Power and fuel
- Receding face
- Inventory

The mine production schedule developed from the FS mine planning process provides the basis for estimating OPEX and the timing of expenditures. The mine OPEX estimate is expressed in terms of dollars per ROM ore ton. Ongoing costs are part of normal mining operations and considered direct costs. These cost items are dependent on the ore production rate and include labor, materials, power, and fuel. Indirect or fixed costs are independent of the rate of production and include items such as administration, leases, and property taxes. The OPEX estimate for the underground mine does not include any indirect costs (general administrative, property taxes, insurance etc.), as these are included in the surface costs.

The Mine OPEX estimate is summarized in Table 21-9. The costs are expressed in terms of life-of-mine (LOM) dollars, average annual dollars, and dollars per ton of ROM polyhalite ore, which are then converted to dollars per ton of product based on processing plant annual capacity and recovery.

Table 21-9. Estimated Mine OPEX (LOM USD)

Cost Category	LOM Total Cost (millions)	Average Annual Cost (millions)	Cost per ROM Ore Ton	Cost per SOP Ton
Mine Labor	$1,070.2	$21.4	$5.87	$30.57
New Mexico Wage Credits	($6.0)	($0.12)	($0.03)	($0.17)
Equipment Leasing	$117.5	$2.35	$0.64	$3.36
Operating Supplies	$580.1	$11.6	$3.18	$16.57
Maintenance	$433.1	$8.7	$2.38	$12.57
Power and Fuel	$608.3	$12.2	$3.34	$17.38
Receding Face	$124.2	$2.5	$0.68	$3.55
Inventory	—	—		—
Mine Total	$2,927.5	$58.5	$16.06	$83.63

Note: Numbers in parenthesis represent negative value (credit rather than cost).

Table 21-10 shows the further breakdown of the mine OPEX estimate by major sub-category. Mine labor is the largest component of mine OPEX estimate comprising over 36% of the total. Next is mine power (electricity) and fuel at 21%, then mine operating supplies at approximately 20%, followed by mine repair and maintenance cost at nearly 15%. Receding face is next at slightly over 4%, and equipment leases are at 4%.

Table 21-10. Estimated OPEX Weighted Average (LOM USD)

Cost Category	Cost per Ton	Percentage of Cost
Mine Labor (including benefits)	$5.87	36.5%
Operating Supplies	$3.18	19.8%
Repair and Maintenance	$2.38	14.8%
Power and Fuel	$3.34	20.7%
Receding Face	$0.68	4.2%
Equipment Lease Payments	$0.64	4.0%
Total	$16.09	100.0%

Figure 21-2 illustrates the mine OPEX estimate by production year. Once the mine reaches steady-state production, the OPEX estimate tends to level off and be fairly uniform for the remainder of the mine life.

21.2.2.1 Mine Labor—Labor is estimated from the mine headcount details of hourly underground, hourly mine-related surface, salaried underground, and salary mine-related surface personnel required to match the production schedule. Hourly classifications are shown in Table 21-11. Hourly wage rates are based on skill level. Annual salary rates and classifications are listed in Table 21-12. Table 21-13 shows the average yearly mine manpower costs and Table 21-14 shows the total mine labor cost for the LOM.

21.2.2.2 Mine Operating Materials and Suppliers—Operating materials and supplies include consumables such as ground support, drill steels and bits, cutting bits, ventilation tubing and curtains, and water hoses.

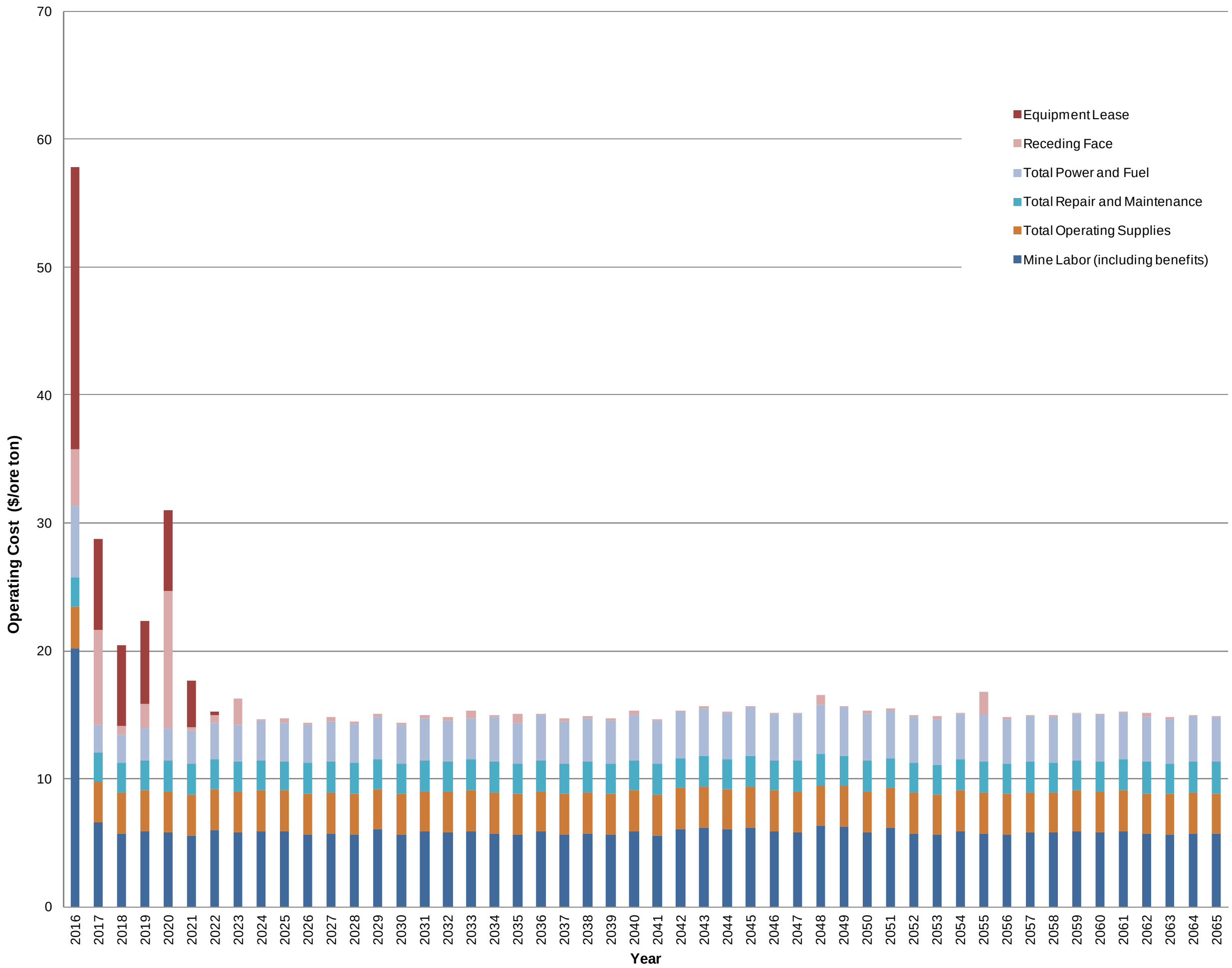


Figure 21-2. Mine OPEX by Production Year

Table 21-11. Hourly Wage and Classification Assumptions

Job Assignment	Hourly Straight Time Wage Range ($/hour)
Production	$26.00 - $29.00
Maintenance	$26.00 - $30.00
General Labor	$21.00 - $26.00
Clerks and Technicians	$23.00 - $27.00

Table 21-12. Salary Wage and Classification Assumptions

Area of Responsibility	Annual Salary ($/year)	Classification
Management	$112,000 – $175,000	E
Operations	$43,000 – $118,000	E/NE
Maintenance	$43,000 – $118,000	E/NE
Engineering and Technical Services	$53,000 – $106,000	E/NE
Administration	$41,000 – $45,000	NE
Safety and Training	$52,000 – $75,000	E
Note: E = exempt, NE = non-exempt.		

Table 21-13. Average Yearly Mine Manpower Costs (USD)

Mine Manpower Summary	Number Per Year	Base Annual Costs	Annual Overtime Costs	Annual Burden Costs	Total Annual Costs
Underground Production					
Hourly Personnel	143	$8,126,648	$1,208,204	$2,893,804	$12,228,657
Salaried Personnel	20	$1,649,389	$2,169	$406,123	$2,057,681
Total Underground Production	163	$9,776,037	$1,210,374	$3,299,927	$14,286,338
Underground Gob Loadout					
Hourly Personnel	12	$632,736	$48,672	$211,236	$892,644
Salaried Personnel					
Total Underground Gob Loadout	12	$632,736	$48,672	$211,236	$892,644
Underground Maintenance					
Hourly Personnel	24	$1,330,368	$209,789	$477,449	$2,017,605
Salaried Personnel	8	$705,389	$2,169	$140,123	$847,681
Total Underground Maintenance	32	$2,035,757	$211,958	$617,572	$2,865,287
Underground General Outby					
Hourly Personnel	16	$843,648	$102,373	$293,267	$1,239,288
Salaried Personnel	4	$320,000		$112,000	$432,000
Total Underground General Outby	20	$1,163,648	$102,373	$405,267	$1,671,288
Suface Mine Operations					
Hourly Personnel	8	$381,264	$60,122	$136,830	$578,216
Salaried Personnel					
Total Surface Mine Operations	8	$381,264	$60,122	$136,830	$578,216
Mine Management and Administration					
Hourly Personnel					
Salaried Personnel	5	$522,284	$2,184	$179,895	$704,363
Total Mine Management and Administration	5	$522,284	$2,184	$179,895	$704,363
Mine Engineering,Technical and Safety					
Hourly Personnel					
Salaried Personnel	10	$544,517	$9,039	$221,059	$774,615
Total Engineering, Technical and Safety	10	$544,517	$9,039	$221,059	$774,615
Project Totals	250	$15,056,243	$1,644,723	$5,071,786	$21,772,752

Table 21-14. Labor Cost by Classification (2013 in thousands USD)

Category	Year							
	2015	2016	2017	2018	2019–2040	2041–2058	2059–2064	2065
Hourly								
Underground								
Production		$3,626	$12,229	$12,229	$12,229	$12,229	$12,229	$12,229
Gob Loadout		$149	$769	$893	$893	$893	$893	$893
Maintenance		$487	$1,669	$1,698	$1,858	$2,018	$2,018	$2,018
General Outby		$489	$1,208	$1,239	$1,239	$1,239	$1,239	$1,239
Total Hourly Underground		$4,751	$15,875	$16,058	$16,218	$16,378	$16,378	$16,378
Surface								
General		$337	$578	$578	$578	$578	$578	$578
Total Hourly Surface		$337	$578	$578	$578	$578	$578	$578
Total Hourly		$5,088	$16,453	$16,637	$16,796	$16,956	$16,956	$16,956
Salary								
Management	$108	$584	$584	$584	$584	$584	$584	$584
Operations		$1,009	$2,472	$2,490	$2,490	$2,490	$2,490	$2,476
Maintenance		$307	$848	$848	$848	$848	$848	$848
Engineering and Technical	$47	$411	$490	$457	$457	$457	$457	$457
Administration		$70	$120	$120	$120	$120	$92	$92
Safety	$14	$164	$318	$318	$318	$318	$318	$318
Salary Total	$169	$2,546	$4,831	$4,816	$4,816	$4,816	$4,788	$4,774
Total Labor	$169	$7,634	$21,284	$21,453	$21,613	$21,773	$21,744	$21,730

Ground support costs are estimated for mains development and production panels. For the FS, for cost development, it is assumed that all intersections would be roof-bolted, as the anhydrite above the polyhalite is selectively removed during the mining cycle, and the halite roof is generally a stable roof and bolted on an as-needed basis. In the production panels, the anhydrite is undercut and supported by a full roof-bolting pattern. Table 21-15 shows the ground support cost estimate.

Table 21-15. Ground Support Estimate

Mining Area	Item Detail	Cost per Item ($)	No. Units per Foot	Cost per Foot ($/ft)	Cost per Ton ($/ton)
Mains Development	#6 x 48-inch, grade 75, headed rebar bolt	$4.65	0.29	$1.37	$0.13
52 bolts every 177 ft of advance (including crosscuts)	6x6, grade 2, domed plates with 1-inch hole	$0.98	0.29	$0.29	$0.03
0.294 bolts per ft	Resin for #6 bar fully encapsulated	$1.60	0.29	$0.47	$0.05
10.22 tons per ft of development	Minimum total roof support cost/ton of development				$0.21
Panels (Advance and Retreat)	#6 x 48-inch, grade 75, headed rebar bolt	4.65	1.20	$5.58	$0.38
6 bolts every 5 ft of advance, 33 ft wide	6x6, grade 2, domed plates with 1-inch hole	0.98	1.20	$1.18	$0.08
1.20 bolts per ft	Resin for #6 bar fully encapsulated	1.60	1.20	$1.92	$0.13
14.67 tons per foot of development and retreat	Minimum total roof support cost/ton of panel mining				$0.59
Minimum total mine ground support cost per ton					$0.71
Estimated cost with 14% mains development and 86% panel mining including additional 20% for wastage and supplemental support					$0.77

Due to the physical characteristics of the polyhalite, continuous miner cutting bits replacements were costed at double that of the nearby potash mines, or 110 bits per machine per shift. Other costs are factored from room-and-pillar operations with similar equipment and methods

21.2.2.3 Underground Repair and Maintenance—These costs are directly reflected in the anticipated mining conditions. These costs include replacement equipment tailing cables and cutter drums, hydraulic oils and lubes, and other materials necessary to maintain the underground equipment. Cutter drums are budgeted to be replaced one per year. Major equipment rebuilds are capitalized in mine sustaining CAPEX estimate. Other costs are factored from room-and-pillar operations with similar equipment and methods.

21.2.2.4 Power and Fuel—High horsepower production and ore haulage equipment in the underground mine is operated using electric power. Electric power is estimated using the engineering design of the distribution system and estimated demand. The mine's LOM electric power estimate is provided in Table 21-16. Diesel-powered equipment will be used for personnel and supply transport, and for service equipment. Fuel cost is estimated from the inventory of diesel equipment and factored from similar mines.

21.2.2.5 Receding Face—Receding face cost is an OPEX category based on US Internal Revenue Service regulations that permit a mine to expense for income tax purposes certain capital equipment. Underground belt conveyors, high-voltage power cables, and water lines used to maintain existing design production as the working faces retreat from the mine opening, are included in this cost category. Table 21-17 provides for the LOM receding face equipment cost.

21.2.2.6 Equipment Leasing—Most of the mine's underground production equipment, ore handling equipment, and outby equipment, and the slope belt conveyor and drives will be leased for the initial 5 years of operation. Lease payments include freight, contingency, and applicable taxes. Lease buybacks at the end of the lease terms are included in the CAPEX estimate at a cost of 25% of the equipment's original purchase price. Table 21-18 shows the equipment lease payment schedule.

21.2.3 Surface Facilities OPEX

Processing and surface area costs were estimated by breaking down the costs into the following areas:

- Labor
- Electrical
- Natural gas and fuel
- Equipment leasing
- Reagents
- Maintenance
- Other miscellaneous costs

Each of the above categories is described in the sections that follow.

21.2.3.1 Manpower—Personnel requirements and wages were based on knowledge in operating and staffing potash mines and processing plants in the Carlsbad region. Wages were also benchmarked against data available through the Bureau of Labor and Statistics (BLS) and other sources. A summary of the annual manpower costs is shown in Table 21-19.

Table 21-16. Annual Underground Power Cost Projections Including Slope Conveyor

Year	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025
Mine Plan Year	1	2	3	4	5	6	7	8	9	10
UG mine demand (kW)	5,398	15,108	17,554	20,240	21,110	22,817	23,738	24,536	27,314	26,101
UG mine energy (kW-hr)	42,557,832	119,111,472	138,395,736	159,572,160	166,431,240	179,889,228	187,150,392	193,441,824	215,343,576	205,780,284
UG mine demand cost ($)	$448,682	$1,255,777	$1,459,088	$1,682,349	$1,754,663	$1,896,549	$1,973,103	$2,039,432	$2,270,340	$2,169,515
UG mine energy cost ($)	$1,472,629	$4,121,614	$4,788,908	$5,521,675	$5,759,020	$6,224,707	$6,475,965	$6,693,667	$7,451,534	$7,120,615
UG mine total power cost ($)	$1,933,585	$5,389,666	$6,260,271	$7,216,299	$7,525,958	$8,133,531	$8,461,342	$8,745,375	$9,734,148	$9,302,405
Ore production (tons)	377,697	3,247,915	3,749,846	3,655,576	3,733,779	3,871,664	3,589,576	3,723,839	3,679,910	3,678,954
Cost per ton ($/ton)	$5.12	$1.66	$1.67	$1.97	$2.02	$2.10	$2.36	$2.35	$2.65	$2.53

Year	2026-2035	2036-2045	2046-2055	2056-2065
Mine Plan Years	11-20	21-30	31-40	41-50
UG mine demand (kW)	280,768	314,068	324,830	324,830
UG mine energy (kW-hr)	2,213,574,912	2,476,112,112	2,565,374,760	2,565,374,760
UG mine demand cost ($)	23,337,436	26,105,332	27,046,417	27,046,417
UG mine energy cost ($)	76,596,333	85,680,907	88,769,663	88,769,663
UG mine total power cost ($)	$99,933,769	$111,786,240	$115,816,080	$115,816,080
Ore production (tons)	37,460,796	36,898,804	36,844,810	37,823,772
Cost per ton ($/ton)	$2.67	$3.03	$3.14	$3.06

RATES:

Southwestern public third revised rate No. 34, backbone transmission of 115kV and above.

Service Company Rates	Summer	Winter	Normalized	Annual
$/kW/month	$8.02	$6.38	$6.93	$83.12
$/kWh	$0.034603	$0.034603	$0.034603	$0.034603
Service availability charge	$1,022.90	$1,022.90	$1,022.90	$12,274.80

Notes: 8,760 hours of operation per year; 0.9 annual load factor; 7,884 net hours of operation; years 31 to 50 estimated from peak year 2045. UG = underground.

Table 21-17. Receding Face Equipment (USD millions)

Year	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026–2035	2036–2045	2046–2055	2056–2065
Mine Plan Year	1	2	3	4	5	6	7	8	9	10	11–20	21–30	31–40	41–50
Equipment Category														
Fresh water pipes and fittings		$0.848	$0.043	$0.046	$0.290	$0.060	$0.046	$0.184	$0.011	$0.066	$0.661	$0.274	$0.535	$0.146
60-inch conveyor components		$11.210	$0.306	$0.339	$1.697	$0.476	$0.339	$2.731		$0.530	$7.126	$1.761	$9.563	$0.360
42-inch conveyor components	$1.520	$7.655	$1.911	$6.015	$35.633	$0.179	$1.831	$3.903	$0.179	$0.179	$1.791	$1.435	$1.791	$1.791
Electrical, control, monitoring and communications cables	$0.120	$2.020	$0.089	$0.135	$0.305	$0.033	$0.032	$0.382	$0.027	$0.129	$1.250	$0.758	$1.201	$1.183
Ventilation		$2.477	$0.262	$0.378	$2.089	$0.145	$0.073	$0.502	$0.076	$0.123	$1.415	$0.979	$1.349	$1.232
Total	$1.640	$24.209	$2.611	$6.914	$40.014	$0.893	$2.322	$7.703	$0.294	$1.027	$12.243	$5.207	$14.438	$4.712
Receding Face Operating Cost per Ton ($/t)	$4.343	$7.454	$0.696	$1.891	$10.717	$0.231	$0.647	$2.068	$0.080	$0.279	$0.327	$0.141	$0.392	$0.125

Table 21-18. Equipment Lease Payment Summary (USD)

Equipment Category	Year 2016	2017	2018	2019	2020	2021	2022
Production equipment—section	$3,332,144	$10,968,439	$11,723,713	$11,723,713	$11,596,241	$7,764,158	$739,001
Outby mobile equipment	$362,468	$914,704	$952,657	$952,657	$952,657	$529,778	$37,954
Other underground equipment / facilities	$17,394	$34,789	$34,789	$34,789	$34,789	$14,495	
Underground ore handling equipment	$1,146,263	$2,641,969	$2,641,969	$2,641,969	$2,641,969	$1,362,903	
Surface/portal facilities shaft and slope	$2,018,938	$4,037,876	$4,037,876	$4,037,876	$4,037,876	$1,682,448	
Surface mobile equipment - mine site	$13,589	$27,179	$27,179	$27,179	$27,179	$4,530	
Frieght allowance on leased equipment	$463,609	$1,390,828	$1,390,828	$1,390,828	$1,390,828	$927,219	
Contingency on leased equipment	$496,277	$1,488,830	$1,488,830	$1,488,830	$1,488,830	$992,553	
Gross receipts tax on leased equipment	$486,790	$1,460,370	$1,460,370	$1,460,370	$1,460,370	$973,580	
Grand Total	$8,337,473	$22,964,985	$23,758,212	$23,758,212	$23,630,741	$14,251,665	$776,955

Note: Lease payments developed by ICP

Table 21-19. Average Yearly Process Manpower Costs (USD)

Manpower Summary	Number Per Year	Base Annual Costs	Annual Overtime Costs	Annual Burden Costs	Total Annual Costs
Plant Operations					
Hourly Personnel	64	$3,427,840	$650,433	$1,199,744	$5,278,017
Salaried Personnel	6	$602,382	-	$210,834	$813,216
Total Plant Operations	70	$4,030,222	$650,433	$1,410,578	$6,091,233
Plant Maintenance					
Hourly Personnel	20	$1,116,669	$211,888	$390,834	$1,719,391
Salaried Personnel	2	$202,000	-	$70,700	$272,700
Total Plant Maintenance	22	$1,318,669	$211,888	$461,534	$1,992,091
Loadout and SOP Transport					
Hourly Personnel	33	$1,740,960	$397,097	$609,336	$2,729,393
Salaried Personnel	4	$320,000	-	$112,000	$432,000
Total Loadout and SOP Transport	37	$2,060,960	$397,097	$721,336	$3,161,393
Quality Control Lab					
Hourly Personnel	5	$247,187	$54,810	$85,516	$388,513
Salaried Personnel	1	$90,000	-	$31,500	$121,500
Total Quality Control Lab	6	$337,187	$54,810	$118,016	$510,013
Project Totals	135	$7,747,038	$1,296,228	$2,711,463	$11,754,729

The process plant and trucking operations to the loadout facility will operate 24 hours per day, 7 days per week, with either three 8-hour shifts (trucking) or two 12-hour shifts (plant). The loadout facility for outbound shipments will operate on a single 8-hour shift per day, 5 days per week; however, it will receive product 24 hours per day from the process plant.

All hourly workers have a 5% overtime allowance at 150% of base wages, plus 7.65% Federal Insurance Contributions Act addition based on their base salary. Burden is 35% of base wages for all employees of the plant. Where possible, salaries for most positions, and specifically nearly all hourly positions, were derived from BLS data for the calendar year 2012. Unless local prevailing wages were known to be higher, positions were costed at the 75th percentile for the BLS category in which they fell. When BLS data did not have a position listed, salary data was gathered from other public sources or ICP experience.

21.2.3.2 Electrical—Electrical loads were developed for the surface facilities (process plant, loadout facility, the well field, and the RO water system). In a Letter of Intent dated June 18, 2013, Xcel Energy provided a basis for calculation as well as a current rate tariff. Current engineering design indicates that the surface facilities will demand approximately 70 MW of electrical load at peak operation. The Xcel Energy letter also established a commitment to provide 115-kV service to the Property at the referenced 115-kV tariff rate. Using these rate schedules and the associated load list, annual electrical operating costs for the surface areas were developed. These costs are detailed in Table 21-20.

Table 21-20. Projected Annual Surface Facilities Electrical Usage and Costs (USD)

Surface Area	Service Voltage (kV)	Maximum Load (MW)	Annual Usage (MWh)	Annual Cost
Process Plant and Water Treatment System	115	63.4	451,080	$21,101,127
Loadout Facility	69	1.5	4,036	$284,575
Well Field	69	5	39,104	$1,855,526
MWh = megawatt-hour				

21.2.3.3 Natural Gas and Fuel—Natural gas quantities were derived from SNCL and vendor calculations. Natural gas pricing was compiled using various sources, with the final derivation being developed by a regional energy broker in Midland, Texas. Estimated gas usage and annual cost is detailed in Table 21-21.

Table 21-21. Projected Annual Natural Gas Usage and Cost (USD)

Gas Usage Area	Consumption (MMBTU/year)	Unit Cost ($/MMBTU)	Annual Cost
Calcination	4,130,702	$3.69	$15,242,289
Steam Generation	3,325,373	$3.69	$12,270,627
SOP Drying	345,524	$3.69	$1,274,985

21.2.3.4 Equipment Leasing—Mobile surface equipment will be leased for the first 5 years of operation. This equipment, in total, is valued at $8.3 million. Equipment has been calculated to be leased at an implicit rate of 8% with an additional 2% included in owner's costs for upfront fees. At the termination of the leases in Year 5, a charge equivalent to 25% of the initial value has been included in sustaining capital to purchase the equipment. All equipment purchases beyond the initial purchase have been accounted for in sustaining capital.

21.2.3.5 Reagents—Reagents used throughout the process fall into one of five categories: de-dusting (or anti-dusting agents), boiler water treatment chemicals, granulation binders (lignin), RO water treatment, and laboratory chemicals. De-dusting addition rates were based on SNCL's previous experience in conjunction with recommended addition rates by AkzoNobel, a de-dusting agent supplier. Boiler water treatment and granulation binder rates were derived solely from SNCL's experience. While boiler water treatment chemical is a direct function of SOP production and is modeled accordingly, de-dusting agent and granulation binder usages are directly related to the individual product splits produced by the plant. For example, standard SOP is modeled using only 1.0 lb/t of de-dusting agent while granular product requires 4.0 lb/t. Furthermore, lignin is only used during the production of granular SOP. Therefore, each of these chemicals was modeled in accordance with the projected product splits taken into account. RO water treatment costs average $1,143,040 per year. These chemical costs include sodium hypochlorite, sodium hydroxide, hydrochloric and sulfuric acids, anti-scaling agents, and water softening agents for potable water production for the surface facilities. Table 21-22 outlines the average annual costs reagents.

Table 21-22. Projected Annual Reagent Use and Costs (USD)

Reagent Use Area	Average Annual Cost	Cost/Ton of Ore	Cost/Ton of SOP
De-dusting	$1,374,100	$0.38	$1.96
Boiler Water Treatment	$122,239	$0.03	$0.17
SOP Granulation	$883,931	$0.24	$1.26
RO Water Treatment	$1,143,040	$0.31	$1.63
Laboratory	$20,000	$0.01	$0.03
Total	$3,520,450	$0.97	$5.03

21.2.3.6 Maintenance—Maintenance and repair (M&R) costs were divided into one of four categories: non-crystallizer sections of the plant, crystallizer area, loadout facility, and water treatment system. M&R costs have been escalated from an initial 0.50% of applicable direct capital expenditures to a set maximum for each category based on expected major wear parts.

ICP assumed that minimal M&R will be needed in years 2017 through 2019 because all equipment and parts will be new. Beginning in year 2020, all costs escalate at the rate of 0.075% until reaching the maximum. Years 2017 and 2066 were further prorated based on the time the plant and other surface facilities are scheduled to operate. All M&R costs reflect only materials and, if needed, minor outside (contract) labor to repair equipment not including the tailings and product transport areas. ICP assumed that the labor is sufficient for completing the majority of all repairs to the surface facilities. Table 21-23 outlines these costs, including maintenance area labor from the above referenced table.

Table 21-23. Annual Maintenance and Repair OPEX (USD)

M&R Area	Minimum Percentage of Direct CAPEX	Maximum Percentage of Direct CAPEX	Average Annual Cost	Cost/Ton of Ore	Cost/Ton of Product
Process Plant (Non-Veolia)	0.50%	1.10%	$3,472,499	$0.95	$4.96
Crystallizer Area (Veolia)	0.50%	1.15%	$1,263,759	$0.35	$1.81
Loadout	0.50%	0.90%	$403,914	$0.11	$0.58
RO Water System	0.50%	0.80%	$310,068	$0.09	$0.44
Manpower	N/A	N/A	$1,358,264	$0.37	$1.94
Total M&R Costs			$6,808,504	$1.86	$9.72

21.2.3.7 Other Miscellaneous Costs—Other miscellaneous OPEX are summarized in Table 21-24.

Table 21-24. Detail of Other OPEX (USD)

Description	Average Annual OPEX	Cost/Ton of Ore	Cost/Ton of Final Product
Gypsum Tailings and Waste Brine Management	$2,297,581	$0.63	$3.28
Product Transportation	$2,654,332	$0.73	$3.79
Reclamation	$300,000	$0.08	$0.43
Environmental Monitoring	$125,000	$0.03	$0.18
Communications	$60,000	$0.02	$0.09
Total	$5,436,913	$1.49	$7.77

21.2.3.8 Gypsum Tailings and Waste Brine Management—Solid tailings costs were developed using an approach that blended original equipment manufacturer (OEM) data for haul trucks with industrial time study standards. Design haulage distances and various truck capacities, speed limits, and operating hours were compiled. It was determined that a minimum of three 46-t haul trucks will be required to haul the anhydrite (gypsum) tailings from the plant to the TSF.

M&R costs for tailings equipment was estimated using a blend of OEM fluid capacities, service intervals, recommended tire lives, and factors of initial capital costs for other miscellaneous repairs. All of these tailings operating costs, inclusive of manpower are summarized in Table 21-25.

Table 21-25. Annual Tailings and Pond Costs (USD)

Description	Average Annual OPEX	Cost/Ton of Tailings	Cost/Ton of Ore	Cost/Ton of Final Product
Tailings Haul Trucks				
Fuel	$847,804	$0.35	$0.23	$1.21
Maintenance	$320,560	$0.13	$0.09	$0.46
Tailings Bulldozer				
Fuel	$66,089	$0.03	$0.02	$0.09
Maintenance	$63,892	$0.03	$0.02	$0.09
Manpower	$999,236	$0.41	$0.27	$1.43
Total Tailings Costs	$2,297,581	$0.95	$0.63	$3.28

21.2.3.9 Product Transportation— Finished product will be transported by semi-tractor and trailer to the loadout facility. ICP will operate its own trucking fleet to transport the material. Five trucks per shift will be needed to haul the product from the plant to the loadout facility. OPEX in Table 21-26 include all materials, supplies, mechanical parts, diesel, and manpower needed to operate the fleet.

Table 21-26. Annual SOP Transport OPEX (USD)

Description	Average Annual OPEX	Cost/Ton of Ore	Cost/Ton of Final Product
Manpower	$1,640,340	$0.45	$2.34
Fuel and Maintenance Costs	$953,602	$0.26	$1.36
Annual Taxes and Fees	$60,390	$0.02	$0.09
Total SOP Product Transportation Costs	$2,654,332	$0.73	$3.79

21.2.3.10 Other—Reclamation costs were determined using the Standardized Reclamation Cost Estimator developed by the Nevada Department of Environmental Protection, BLM, and Nevada Mining Association. This cost came to a total of $15 million, which equates to an annual cost of $300,000 over the 50-year life of the Ochoa Project.

Environmental monitoring consists of testing and analysis expenditures required for compliance with all environmental permits. This includes, but is not limited to, the monitoring of air quality, climate, rainfall, subsidence, and groundwater.

Communications costs include ICP communication of safety, human resources, and other issues to employees.

21.2.4 General and Administrative

General and administrative labor costs include non-production related management, accounting, environmental, purchasing, and security. Items such as courier services, association memberships, office supplies, travel expenses, IT materials and services, and local and state community support are included. Project insurance is also covered within this category. A burden of 35% has been applied to wages and salaries. For the FS, a quotation was obtained for project policies with a minimum of $5 million retention. General and administrative costs are listed in Table 21-27. Table 21-28 shows the labor portion of the general and administrative costs.

Table 21-27. Average Annual General and Administrative Costs (USD)

Description	Average Annual OPEX	Cost/Ton of Ore	Cost/Ton of Final Product
Manpower	$1,664759	$0.46	$2.38
Office Supplies/Expenses	$1,423,973	$0.39	$2.03
Insurance	$2,885,893	$0.79	$4.12
Total	$5,974,625	$1.64	$8.53

Table 21-28. Average Yearly Administration Manpower Costs (USD)

Manpower Summary	Number Per Year	Base Annual Costs	Annual Overtime Costs	Annual Burden Costs	Total Annual Costs
Administration					
Hourly Personnel	12	$437,632	$32,632	$153,171	$623,435
Salaried Personnel	11	$800,470	-	$280,165	$1,080,635
Total Administration	23	$1,238,102	$32,632	$433,336	$1,704,070

21.2.5 Accuracy Assessment

The OPEX estimate presented in this chapter were based on a Class 3 estimate methodology. ICP estimates the expected order of accuracy is in the range of ±15%, as required by this estimate class and is appropriate for an FS. The primary purpose of a Class 3 estimate is to provide a level of engineering effort that is sufficient to evaluate configurations and select a preferred configuration as the basis for the execution of the Project.

The definitions of the accuracy in this document were based on the following:

- Public domain unit costs were obtained for power, fuel (natural gas and diesel), reagents, and material.
- A detailed electrical power tariff was obtained from Xcel Energy covering all power supply surcharges, including winter and summer rates, downtime rate, Lea County tax, New Mexico tax, and the appropriate power factor, for an accurate estimated supply cost of power for the LOM of the plant.
- The quantities of power, fuel, and reagents were based on the level of engineering details produced.
- Costs for water supply and treatment, tailings management, and mining were obtained from contractor proposals.
- Labor cost was estimated for each work discipline based on a plant staffing plan.
- Maintenance and operating supplies were established as a percentage of the relevant capital cost.

ITEM 22: ECONOMIC ANALYSIS

The economic analysis was performed using estimates of CAPEX and OPEX elsewhere described in this TR. Since the economic analysis is based on a cash flow estimate, it should be expected that actual results will vary from these predictions.

Details of the assumptions on which the analysis is based and results of the sensitivity analysis are provided in the following sections.

22.1 Inputs to the Financial Analysis Model

22.1.1 Summary of General Input Data

The financial analysis was carried out using the following assumptions for the business case:

- All amounts, including cash flows during construction and operations, are expressed in September 15, 2013 USD, with no allowance for escalation or inflation.
- Project IRR is estimated using the discounted cash flow methodology.
- Total project life is approximately 53 years (i.e. 50 years of operation after a 3-year construction period).
- Project IRR was calculated after consideration of estimated federal and New Mexico state income taxes, other taxes, and public and private royalties.
- The analysis was performed on the basis that the Project will be 100% equity financed.
- Working capital during construction is based on an average third-party payables period of 30 days.
- Working capital during operation is based on an average receivable collection period of 35 days and on an average third-party payables payment period of 30 days.

22.1.2 Revenues

The market outlook for SOP prices is based on estimations from ICP. Prices used in the economical analysis are forecasts for SOP FOB the loading facility at Jal, New Mexico less sales related expenses, without inflation, and are expressed in USD per ton, which is shown in Table 22-1.

Revenue was calculated on the basis of nominal production of the three products produced. Upon full production of 714,400 tpy, revenues have been calculated based on a product mix of 229,400-t standard SOP, 385,000-t granular SOP, and 100,000-t soluble grade SOP at Jal less other sales-related expenses.

22.1.3 Expenses

22.1.3.1 OPEX—Equipment OPEX for the mine and processing plant include maintenance parts, lube, tires, wear parts, supplies, and diesel fuel where applicable. Electricity costs and labor were tracked separately from the equipment OPEX. Maintenance and

Table 22-1. SOP Price Forecast USD per Ton Plant Gate at Jal

Year	USD per Ton Plant Gate at Jal		
	Standard Grade SOP	Granular Grade SOP	Soluble Grade SOP
2017	540	586	631
2018	511	557	602
2019	522	568	613
2020	539	585	630
2021	569	614	660
2022	575	621	666
2023	581	627	672
2024	594	639	685
2025 and beyond	607	652	697

operating staff were included in the staff and personnel detail. OPEX was determined based on the annual production rate of 714,400 t of SOP. The cost per ton of finished product is based on total mineral production. Major component rebuild costs are not included within the OPEX because these items are capitalized, as discussed in the section on sustaining CAPEX.

Table 22-2 details the steady-state costs for the Ochoa Project. Steady state has been defined as the operating years from 2022 through 2065. These years generally exclude major one-time OPEX that are included in years 2017 through 2021 and 2066 such as equipment leasing, initial receding face expenditures, and inventory adjustments as well as the costs associated with project start-up and closure.

Table 22-2. Steady-State Average Annual OPEX (USD)

Operating Cost	2022 to 2065 Cost (millions)	Average Annual Cost (millions)	Cost/Ton of Ore	Cost/Ton of Product
Mining	$2,475.8	$56.27	$15.12	$78.76
Processing	$3,389.0	$77.02	$20.70	$107.82
General and Administrative	$267.4	$6.08	$1.63	$8.51
Total OPEX	$6,132.3	$139.37	$37.46	$195.09

22.1.3.2 Sustaining CAPEX—Total sustaining CAPEX for the life of the Project has been estimated to be approximately $1.407 billion USD. Anticipated sustaining CAPEX of $831.9 million for the mine were provided by AAI. ICP estimated sustaining CAPEX for the process facilities and infrastructure at $450.7 million. Three additional magnesium sulfate ponds with a total cost of $9.3 million will be added in the first 3 years of operation. The TSF will be expanded in 20 phases, every second year, until reaching final capacity for a total cost of $115.2 million. The sustaining CAPEX are illustrated in the graph in Figure 22-1. The average sustaining CAPEX per ton per year is approximately $40.

22.1.3.3 CAPEX—The construction cost of the Project is estimated to be approximately $1,018 million USD, of which $671 million is direct costs, $155 million is indirect construction costs (EPCM, field indirect, etc.), $112 million in contingency costs, and $80 million is owner's costs. The CAPEX is spent during the period starting from Q2 2014 and ending Q2 2017. Operations are scheduled to commence in the Q3 of 2017.

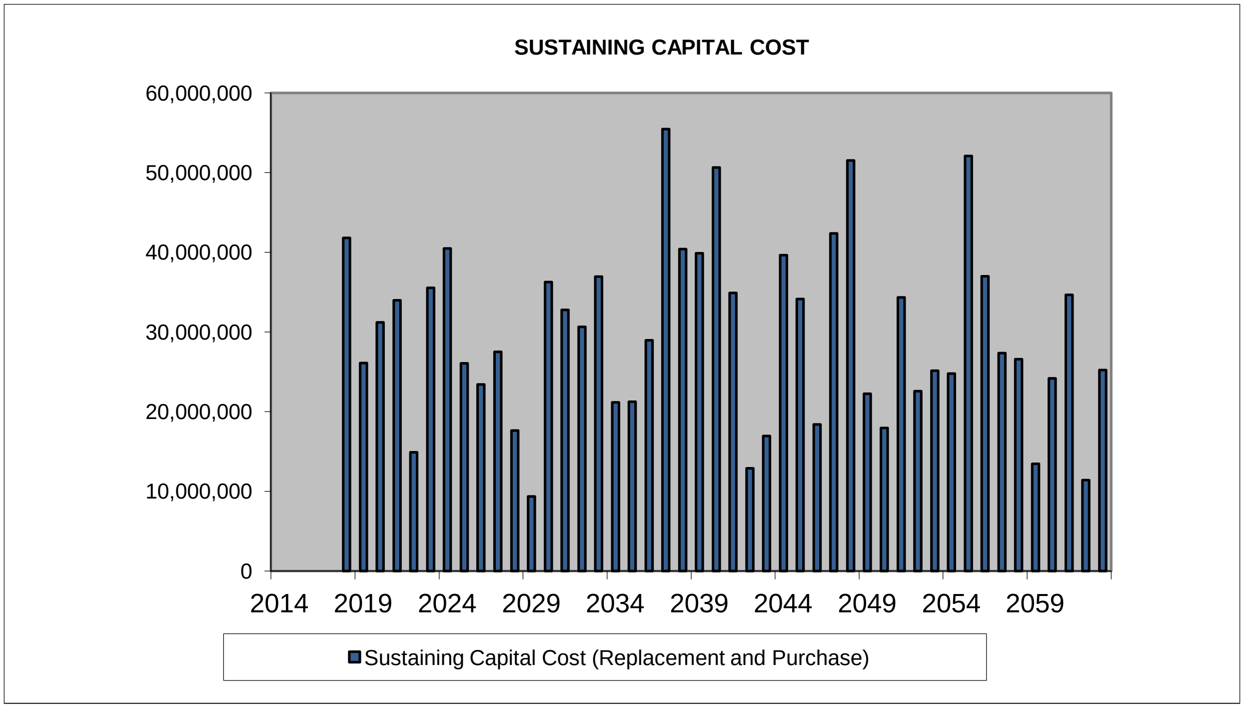


Figure 22-1. Projected Sustaining CAPEX Schedule During Operation

CAPEX does not include any financing fees or interest during construction.

22.1.4 Financing

Analyses were carried out for a 100% equity financed project. The financial model does not include any financing up-front fee or equity underwriting fee.

22.1.5 Taxes and Royalties

Taxes were included in the cash flow. In addition to New Mexico state and US federal corporate income tax, the Project is also subject to severance tax, resource excise tax, property tax, and gross receipts tax as well as State and BLM royalties and production royalties.

22.2 Cash Flow and Financial Valuation Analysis

The project was evaluated under Discounted Cash Flow analysis. All 2013 real values for revenues and costs have not been inflated. Basic cash flow calculations were developed by feeding estimates and assumptions into a financial model constructed in Microsoft Excel™. The project IRR was calculated according to Discounted Cash Flow methodology, and sensitivity analyses were completed. The financial model covers approximately 3 years of construction, from Q2 2014 through Q2 2017, and 50 years of operations. SOP production in 2017 is estimated at 48% of annual capacity, with full capacity expected in 2018.

The economic results yielded a full equity base case IRR of 16.0%. The NPV is $1,018.9 million at a discount rate of 8% and $612.0 million when discounted at 10%. The estimated investment payback period is 5.4 years of operations. These results are calculated after income taxes, other taxes, and royalties.

22.2.1 Overall Cash Flow

Figure 22-2 shows the assumed equity injection during the construction period and the estimated free cash flow generated during the life of the Project. Table 22-3 shows the Project's cash flow for the 50-Year Mine Plan. Figure 22-3 shows the different estimated cash flow allocations over the Project's life.

22.2.2 Sensitivity Analysis and Risks

A sensitivity analysis was performed on the economic analysis taking into account variations in CAPEX, revenues, OPEX and sustaining CAPEX. The sensitivity analysis, summarized in Figure 22-4, shows that the Project's IRR is mainly sensitive to variations in revenues and CAPEX.

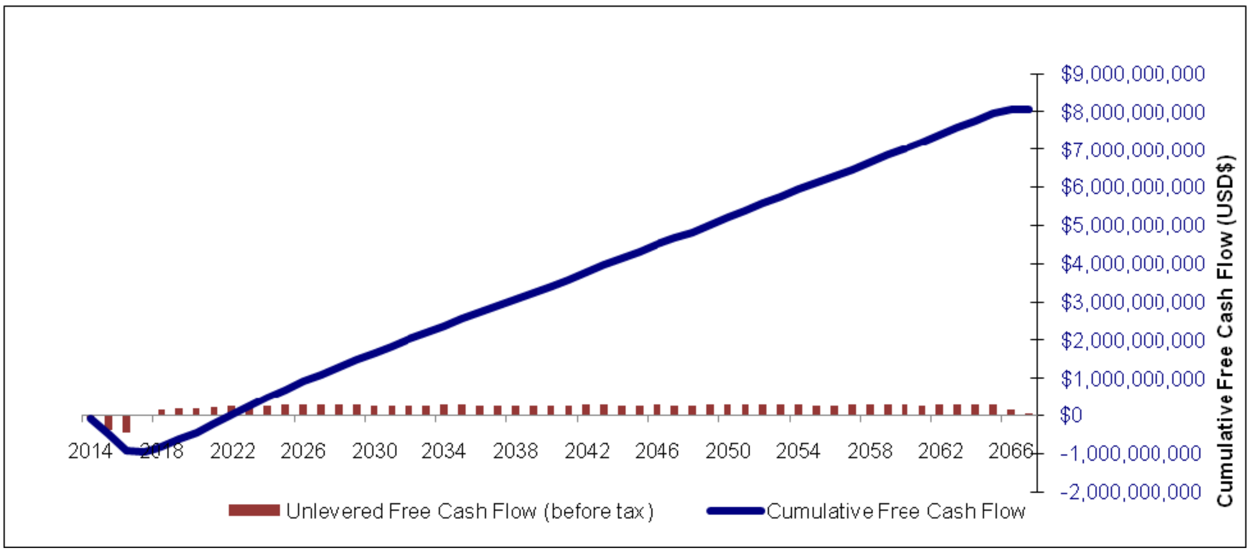


Figure 22-2. Free Cash Flow Over Life of Project

Table 22-3. Ochoa Project Cash Flow for 50-Year Mine Plan

		Totals	4/1/2014 12/31/2014	1/1/2015 12/31/2015	1/1/2016 12/31/2016	1/1/2017 12/31/2017	1/1/2018 12/31/2018	1/1/2019 12/31/2019	1/1/2020 12/31/2020	1/1/2021 12/31/2021	Steady State Annual Average 2022 - 2065	1/1/2066 12/31/2066	1/1/2067 12/31/2067
PRODUCTION													
Polyhalite Mined - Short Tons	Tons	182,336,936	0	0	377,697	3,247,915	3,749,846	3,655,576	3,733,779	3,871,664	3,720,465	0	0
SOP Production - Short Tons													
Standard	Tons	11,353,181	0	0	0	180,217	341,981	229,400	229,400	229,400	229,400	49,183	0
Granular	Tons	18,752,419	0	0	0	131,383	272,419	385,000	385,000	385,000	385,000	253,617	0
Soluble	Tons	4,900,000	0	0	0	33,333	100,000	100,000	100,000	100,000	100,000	66,667	0
Finished Product - Short Tons		35,005,600	0	0	0	344,933	714,400	714,400	714,400	714,400	714,400	369,468	0
NET BACK PRICE ($)													
SOP Price/Short Ton, Net Back													
Standard			$ -	$ -	$ -	$ 540	$ 511	$ 522	$ 539	$ 569	$ 605	$ 605	$ -
Granular			$ -	$ -	$ -	$ 586	$ 557	$ 568	$ 585	$ 614	$ 650	$ 651	$ -
Soluble			$ -	$ -	$ -	$ 631	$ 602	$ 613	$ 630	$ 660	$ 695	$ 696	$ -
STATEMENT OF OPERATIONS ($ millions)													
Revenues, net	USD $	$ 22,256.7	$ -	$ -	$ -	$ 195.3	$ 386.7	$ 399.7	$ 411.9	$ 432.9	$ 458.8	$ 241.3	$ -
Operating Expenses:													
Mining		$ 2,927.5	$ -	$ -	$ 21.8	$ 91.9	$ 75.1	$ 80.2	$ 114.2	$ 68.4	$ 56.3	$ -	$ -
Processing		3,780.3	-	-	-	51.2	74.0	74.5	75.0	76.6	77.0	39.9	-
General and Administrative		298.7	-	-	-	4.5	5.9	5.9	5.9	6.0	6.1	3.1	-
By-Product Revenue/Expense, Net		-	-	-	-	-	-	-	-	-	-	-	-
Sub-total: Operating Expenses		$ 7,006.4	$ -	$ -	$ 21.8	$ 147.6	$ 155.0	$ 160.6	$ 195.1	$ 151.0	$ 139.4	$ 43.1	$ -
Non-Operating Expenses:													
Non-income Taxes		$ 440.9	$ -	$ -	$ -	$ 4.5	$ 7.7	$ 7.9	$ 8.2	$ 8.6	$ 9.1	$ 3.8	$ -
State and BLM Royalties		495.2	-	-	-	4.3	8.6	8.9	9.2	9.6	10.2	5.4	-
Production Royalties		220.7	-	-	0.4	1.9	1.9	1.8	1.9	1.9	4.8	-	-
Interest and Fees		-	-	-	-	-	-	-	-	-	-	-	-
Sub-total: Non-Operating Expenses		$ 1,156.8	$ -	$ -	$ 0.4	$ 10.7	$ 18.2	$ 18.7	$ 19.2	$ 20.2	$ 24.1	$ 9.2	$ -
Earnings/(Loss) before Income Tax		$ 14,093.4	$ -	$ -	$ (22.2)	$ 37.0	$ 213.6	$ 220.4	$ 197.5	$ 261.8	$ 295.4	$ 189.0	$ -
Income Taxes		$ 3,593.5	$ -	$ -	$ -	$ -	$ 0.4	$ 0.7	$ 0.5	$ 1.6	$ 80.5	$ 49.2	$ -
Net Income/(Loss), excluding Depreciation and Amortization		$ 10,499.9	$ -	$ -	$ (22.2)	$ 37.0	$ 213.2	$ 219.7	$ 197.0	$ 260.2	$ 214.9	$ 139.8	$ -
STATEMENT OF CASH FLOWS ($ millions)													
Net Income/(Loss), Excluding Depreciation and Amortization	USD $	$ 10,499.9	$ -	$ -	$ (22.2)	$ 37.0	$ 213.2	$ 219.7	$ 197.0	$ 260.2	$ 214.9	$ 139.8	$ -
Adjustments:													
Working Capital Increase/(Decrease)		$ (0.0)	$ 10.7	$ 24.3	$ 1.4	$ 1.5	$ (19.8)	$ (2.0)	$ 2.1	$ (5.5)	$ 0.2	$ (33.7)	$ 11.0
Cash Flow Provided by Operations		$ 10,499.9	$ 10.7	$ 24.3	$ (20.7)	$ 38.5	$ 193.4	$ 217.7	$ 199.2	$ 254.7	$ 215.1	$ 106.2	$ 11.0
Investing Activities													
CAPEX		$ (1,018.2)	$ (97.7)	$ (425.9)	$ (430.4)	$ (64.2)	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Interest and Fees During Construction		-	-	-	-	-	-	-	-	-	-	-	-
Sustaining CAPEX		(1,407.1)	-	-	-	-	(41.8)	(26.1)	(31.2)	(34.0)	(28.7)	(9.3)	-
Cash Flow Esed in Investing Activities		$ (2,425.4)	$ (97.7)	$ (425.9)	$ (430.4)	$ (64.2)	$ (41.8)	$ (26.1)	$ (31.2)	$ (34.0)	$ (28.7)	$ (9.3)	$ -
Net Increase/Decrease in Cash Flow		$ 8,074.6	$ (87.0)	$ (401.6)	$ (451.1)	$ (25.8)	$ 151.6	$ 191.5	$ 167.9	$ 220.7	$ 186.4	$ 96.9	$ 11.0
Cumulative Increase/Decrease in Cash Flow	USD $	$ 8,074.6	$ (87.0)	$ (488.6)	$ (939.7)	$ (965.5)	$ (813.9)	$ (622.4)	$ (454.4)	$ (233.7)	$ 7,966.7	$ 8,063.6	$ 8,074.6
Net Increase/Decrease in Cash Flow (Pre-Tax)	USD $	$ 11,934.9	$ (87.0)	$ (401.6)	$ (451.1)	$ (25.8)	$ 152.0	$ 192.3	$ 168.4	$ 222.3	$ 266.9	$ 146.0	$ 11.0
Cumulative increase/decrease in Cash Flow (Pre-Tax)	USD $	$ 11,668.1	$ (87.0)	$ (488.6)	$ (939.7)	$ (965.5)	$ (813.5)	$ (621.2)	$ (452.8)	$ (230.5)	$ 11,511.1	$ 11,657.1	$ 11,668.1
INTERNAL RATE OF RETURN (IRR)													
IRR Before Tax	%	17.8%											
IRR After Tax	%	16.0%											

NET PRESENT VALUE ($ millions)		10%	8%	0%
NPV (before taxes)	USD $	$ 942.7	$ 1,502.3	$ 11,668.1
NPV (after taxes)	USD $	$ 612.0	$ 1,018.9	$ 8,074.6

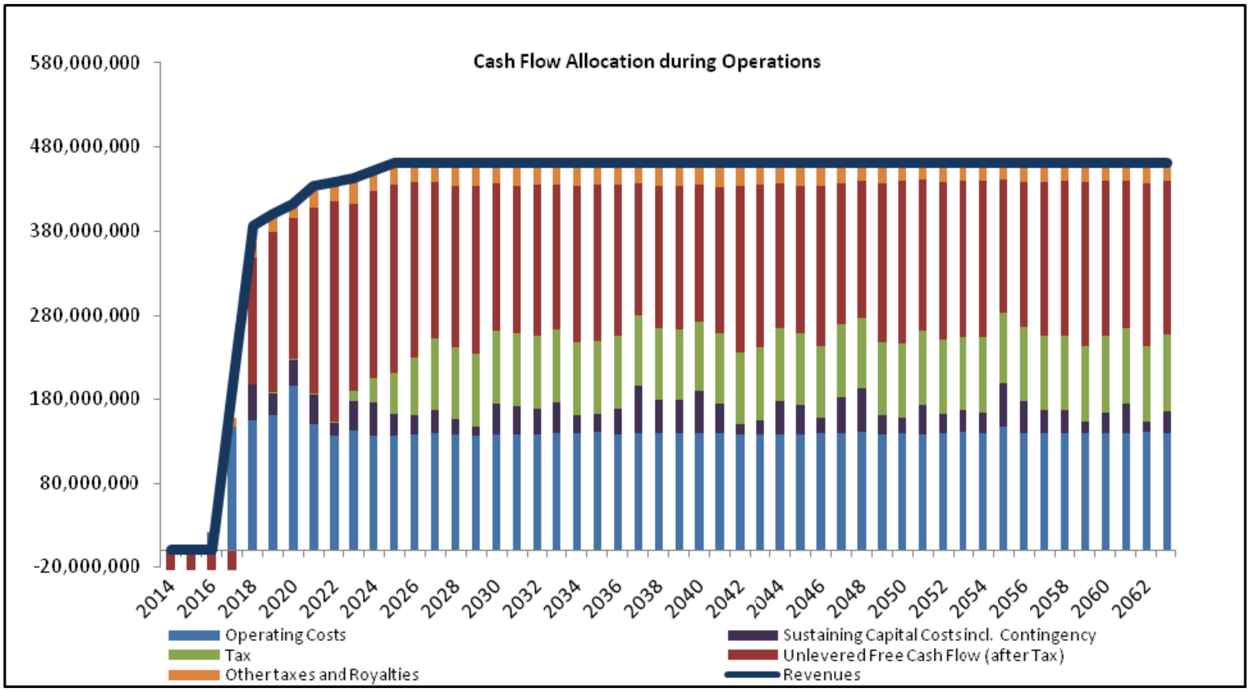
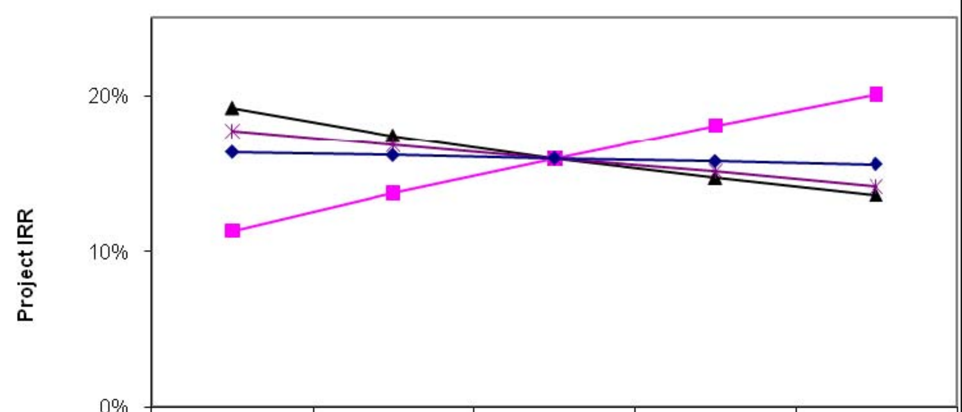


Figure 22-3. Cash Flow Allocations



	80%	90%	100%	110%	120%
Capital Cost including Contingency	19.3%	17.5%	16.0%	14.7%	13.6%
Revenues	11.3%	13.8%	16.0%	18.1%	20.1%
Operating Costs	17.7%	16.8%	16.0%	15.1%	14.2%
Sustaining Capital Costs including Contingency	16.4%	16.2%	16.0%	15.8%	15.6%

Figure 22-4. Sensitivity of Unleveraged Project IRR to Key Parameters

ITEM 23: ADJACENT PROPERTIES

The information presented herein references and is extracted in part from Gustavson's (2011d) December 30, 2011 NI 43-101 TR, its April 2012 PFS, and the SNCL FS (2014), with additions and updates.

Adjacent mineral and surface land owners consist of the BLM, NMSLO, and private entities. BLM and NMSLO have leased gas and oil rights to numerous petroleum companies. Therefore, fluid mineral is being leased to entities separate than solid minerals leases. The BLM has leased surface grazing rights to local ranchers.

The southeastern portion of New Mexico is home to the Carlsbad potash mining district, which hosts the largest USA production of potash, primarily in the form of MOP, with limited SOP or SOPM production. There are no solid mineral mines within 20 miles of the Property and adjacent properties have no known existing impact on the Project. There are no commercial polyhalite mines operating in the USA. Some polyhalite lenses, generally less than 2 ft thick, occur occasionally in the Carlsbad area potash mines and are mined along with the potash mineral. No specific processing facilities exist at those operations to process polyhalite.

Geophysical logs from gas and oil wells located on adjacent properties were used to assist in determining the extent of polyhalite deposits of mineable thickness.

ITEM 24: OTHER RELEVANT DATA AND INFORMATION

The following information is paraphrased from ICP's website and serves to explain the differences in potassium fertilizers and their production.

24.1 NPK Nutrients

Potash contains potassium ("K"), one of three primary nutrients required for plant growth. The other two are nitrogen ("N") and phosphorus ("P"). These elements are often combined and distributed as NPK formula fertilizer. Any natural or manufactured material that contains at least 5% of one or more of the three primary nutrients generally can be considered a fertilizer.

Commercial fertilizers containing nitrogen, phosphorus, or potassium can improve plant yield. There are several sources which can provide these nutrients to plants; the two most important are organic manure and mineral fertilizers. When manure and crop residues are used, mineral fertilizers supply the outstanding nutrient balance needed for good crop yields. In most parts of the world, the balance supplied by mineral fertilizers is substantial.

24.2 Potash

The potassium used in fertilizers is found in a salt form called potash. Potash deposits are derived from evaporated sea water and occur in sediment beds only a few places in the world. Canada, Germany, and Russia contain large deposits of potash. Typically, the potash deposits are mined (dry or solution) and processed for shipping. Southeastern New Mexico contains the largest economically extractable deposits of potash in the USA.

Muriate of Potash (MOP)—MOP is the most common form of potash. It is particularly effective when used in the commercial cultivation of the carbohydrate crops including wheat, oats, and barley. MOP is composed of potassium and chloride in the forms of charged atoms, and therefore in the form of a salt which is soluble in water. MOP has a total global market size of approximately 60 million short tons.

Sulfate of Potash (SOP)—SOP is the second major form of potash, with a chemical formula of K_2SO_4. It is particularly effective in the cultivation of fruits, vegetables, potatoes, tobacco, and tree nuts. SOP has a total global market size of approximately 5.5 million short tons.

SOP is considered superior to MOP because it does not contain chloride, which has a toxic impact on many food plants, especially fruits and vegetables. When MOP is used, soils fall victim to increasing levels of chloride salt which can hurt plant yields. Chloride-free fertilizer enhances plant health, so the demand for SOP has increased.

SOP provides the potassium needed to nourish and strengthen plants, ward off disease, improve transportability, and add flavor. SOP improves crop yield and sells at a premium to MOP. In addition, SOP has a lower salinity index than MOP. The higher salinity of MOP can cause plants to have difficulty absorbing water and nutrients from the soil, thereby diminishing the quality and yield of the crop. SOP has a salinity index of 46, the lowest of the potassium fertilizers, while MOP has a salinity index of 116. For these reasons, producers of high value crops tend to prefer SOP over MOP.

SOP is produced from primary and secondary sources. The primary sources of SOP are minerals and naturally occurring brines predominantly located in China, Germany, Chile, and the USA. The remaining portion of the world's supply comes from the processing of potassium chloride (MOP) with sulfuric acid or with a sulfate salt. This source of supply can be produced anywhere raw materials can be shipped and processed, and is known as the Mannheim Process.

The primary producers of SOP include Sociedad Quimica y Minera (SQM), SDIC Luobupo Potash, GSL, and K+S KALI. SQM, SDIC Luobupo Potash, and GSL produce SOP by utilizing salt lakes, whereas K+S KALI produces SOP from ore mined at its own sites. Due to this fact, K+S KALI could be considered a secondary producer. The secondary source of SOP supply is highly fragmented with Tessenderlo Chemie of Belgium having the highest annual capacity of 750 thousand short tons. Other notable secondary producers of SOP include Qing Shang Chemical and Migao Corp. of China.

24.3 SOP Production

SOP is not a naturally occurring mineral and is produced by chemical methods. Only a few of these processes exist. Four process methods are used to produce SOP and are shown in Table 24-1.

Table 24-1. SOP Production Process Methods

Process Method	World Capacity	Process Inputs	Products
Mannheim	60%	MOP Sulfuric acid Energy	SOP Hydrochloric acid
MOP and kieserite	25%	MOP Kieserite Energy	SOP Magnesium chloride
Salt Lakes	15%	Lake brines Energy	SOP Magnesium chloride Sodium chloride
Ochoa Process	—	Polyhalite Water Energy	SOP

Mannheim Process—The most common method of producing potassium sulfate is the Mannheim Process, which is the reaction of potassium chloride with sulfuric acid at high temperatures. The raw materials are poured into the center of a muffle furnace heated to above 1,112°F. Potassium sulfate is produced along with hydrochloric acid in a two-step reaction via potassium bisulphate. This method for creating SOP accounts for 50% to 60% of global supply. The Mannheim Process is also the most expensive of the processing techniques due to the high input costs associated with purchasing MOP and sulfuric acid, and high energy requirements.

Potassium Chloride and Sulfate Salts—Potassium chloride can be reacted with various sulfate salts to form a double salt that can be decomposed to yield potassium sulfate. The most common raw material employed for this purpose is sodium sulfate. Sodium sulfate, either in the form of mirabilite (also known as Glauber's Salt) or sulfate brine, is treated with brine saturated with MOP to produce glaserite. The glaserite is separated and treated with fresh MOP brine, decomposing into potassium sulfate and sodium chloride. These methods of production are the second greatest source of global supply at 25% to 30%.

Naturally Occurring Brines—Some operations produce SOP from the salt mixtures harvested from natural brines. Three companies produce potassium sulfate in such a way on a large scale: GSL (Great Salt Lake, Utah), SQM (Salar de Atacama, northern Chile), and SDIC Luobupo Potash (Lop Nur, northwest China). This method requires brines with high sulfate levels such as those found within these salt lakes. The sulfate is typically present in the harvest salts in the form of the double salt kainite, which is converted to schoenite by leaching with sulfate brine. The leach process may be hampered by high sodium chloride content in the harvest salts, and the halite is first removed by flotation. After thickening, the schoenite is decomposed by simply adding hot water, whereupon the magnesium sulfate enters solution leaving SOP crystals. This process is currently the lowest cost method to make SOP. As lakes with sufficient brine mineral levels are rare, this method only accounts for 15% to 20% of global supply.

Ochoa Process—ICP's Ochoa Process will convert polyhalite into SOP using unit operations common to the industrial minerals industry. Processing polyhalite to produce SOP involves the following steps: crushing and washing, calcination, leaching, crystallization, and granulation.

24.4 SOP Grades

SOP is available in three main agricultural grades including standard, granular, and soluble.

Standard SOP is used for direct application on hardy crops and the manufacturing of compound fertilizers. It contains 50% potassium oxide, 45% sulfur trioxide, and a maximum of 1% chloride. It appears as fine crystals with a typical particle size range of 0.008 to 0.066 inch.

Granular SOP is the most important and most widely used grade in the USA and many other parts of the world. This grade is used in bulk blends, for mechanized spreading, and for manual application on crops that have even soil nutrient distribution. It contains 50% potassium oxide, 45% sulfur trioxide, and a maximum of 1% chloride. It appears as small granules with a typical particle size range of 0.033 to 0.132 inch. Granular SOP is produced by mechanically compressing the product, and then breaking and screening it to achieve a desired particle size. Granular grade can also be produced by granulating the material and using a binding agent.

Soluble SOP is used in open field fertigation, foliar feeding, and greenhouse and hydroponic systems. It contains 52% potassium oxide, 45% sulfur trioxide, and a maximum of 0.5% chloride. It appears as a fine powder, which dissolves rapidly in water, with a typical particle size range of 0.004 to 0.012 inch. This form of SOP holds less than 5% of the SOP market.

ITEM 25: INTERPRETATION AND CONCLUSIONS

25.1 General

The Ochoa Property contains significant polyhalite mineralization in sufficient quantities and of sufficient grade to be attractive for mining and processing under current market conditions, notwithstanding the risk inherent to proving and developing any mining property. Mineral Resources and Mineral Reserves are stated for the Ochoa polyhalite bed based on an FS (SNCL 2014) for underground room-and-pillar mining and ore processing completed January 2014.

Adequate mine design, permitting requirements, hydrogeologic testing, processing testing, and marketing analysis were conducted to support the mining methods and processing plant design and infrastructure requirements at the FS level. The FS projects an economically viable mining and processing facility with the capacity and polyhalite reserves to produce 714,400 t of SOP per year for a minimum of 50 years.

Other interpretations were:

- There is local support for the Project.
- Lea County and surrounding communities stand to benefit significantly from the Project, including the creation of approximately 400 direct permanent jobs and the payment of new tax revenue to the state and county.
- Three fertilizer-grade SOP products can be produced by incorporating processes and technologies proven viable by testing during the FS.

25.2 Financial Conclusions

25.2.1 General

Tables 25-1 through 25-4 provide summary support for the financial conclusions in this Section 25.2. The financial model as set out in Item 22 covers approximately 3 years of construction and commissioning beginning in Q2 2014 and continuing through Q2 2017, followed by 50 years of operation. SOP production in 2017 is estimated at 48% of annual capacity, with full capacity expected in 2018. In the financial model, no inflation or escalation factors were applied to cash inflows and outflows. Table 25-1 shows the summary of financial results.

After-tax IRR is sensitive to CAPEX, OPEX, and revenue assumptions. Table 25-4 shows the effect of changing those assumptions to ±20%. Table 25-2 shows the sensitivity analysis.

25.2.2 CAPEX

The CAPEX of the Project is estimated to be $1,018 million, with an accuracy of ±15%. Preparation of the CAPEX estimate is consistent with standards defined by the Association for the Advancement of Cost Engineering International for a Class 3 Estimate. Table 25-3 summarizes the total estimated CAPEX by major area.

Table 25-1. Financial Results (USD)

Full Equity Basis (i.e. No Debt)	Before Tax	After Tax
Capital Cost	$1,018 million	$1,018 million
Operating Cost per Ton SOP at Steady State	$195	$195
IRR	17.8%	16.0%
NPV, 8% Discount Factor	$1,502.3 million	$1,018.9 million
NPV, 10% Discount Factor	$942.7 million	$612.0 million
Payback Period (from start of production)	–	5.4 years

Table 25-2. Sensitivity Analysis

Input Variable to Financial Model	–20%	–10%	Base Case	+10%	+20%
CAPEX	19.3%	17.5%	16.0%	14.7%	13.6%
Revenue	11.3%	13.8%	16.0%	18.1%	20.1%
OPEX	17.8%	16.8%	16.0%	15.1%	14.2%

Table 25-3. CAPEX

	Cost (in millions USD)
Mine Infrastructure and Development	$107
Process Plant	$527
Storage and Loading	$37
Total Direct Costs	$671
EPCM Services	$99
Construction Indirect	$22
Freight, Spares, and First Fills	$34
Total Indirect Costs	$155
Owner Costs	$80
Contingency	$112
Project Total	$1,018

Table 25-4. OPEX

	Cost (USD)
Steady-State Production	714,400 tpy of SOP
Mining Cost per Ton	$78
Processing Cost per Ton	$108
General and Administrative Cost per Ton	$9
Total OPEX per Ton	$195
Percentage of OPEX – Labor	24.8%
Percentage of OPEX – Electricity	24.5%
Percentage of OPEX – Natural Gas	20.7%
Sustaining Capital per Ton per Year	$40

25.2.3 OPEX

OPEX is based on scheduled production, equipment requirements, operating hours, equipment operating costs, and manpower requirements. Steady state has been defined as the operating years from 2022 through 2065. Steady-state years generally exclude major one-time costs that are included in years 2017 through 2021, such as start-up activities, equipment rentals, initial receding face expenditures, and inventory adjustments. Table 25-4 summarizes the estimated OPEX.

The plant and mine are expected to employ approximately 400 people at full production. Other conclusions were:

- The Project estimated CAPEX of $1.018 billion USD translates to $1,425 per annual ton of SOP capacity.
- The steady-state OPEX of $195/t of SOP is well below most competing technologies.

25.3 Mining

The FS concludes that sufficient polyhalite reserves exist for a mine life of at least 50 years. The FS determined that a steady-state annual production rate averaging 3.7 Mt of ROM polyhalite ore with a life-of-mine average ore grade of 78.05% polyhalite should be achievable given the mining assumptions used in the FS. Annual polyhalite ore grade averages between 73.24% and 83.38%. Other ore grade parameters are within acceptable limits for SOP production.

There is sufficient data from the 2009–2013 exploration programs to support the geologic interpretations of the mineral deposit on the ICP Property that were used in the FS. Determination of Mineral Resources and Mineral Reserves were based on a geologic model developed using Carlson Software's Geologic Module (Carlson 2013) and the mine projections, scheduling, and production tons and ore grade were based on Carlson's Underground Mining Module (Carlson 2013). Carlson is a commonly used mine planning software for bedded (tabular) deposits. No other Exploration Targets, Mineral Resources, or Mineral Reserves were identified.

Anticipated annual ore tonnage ranges from 3.59- to 3.89-Mt ROM polyhalite. Adequate mine geotechnical testing and modeling analysis was conducted to support establishing the Project's mining methods, ground control, equipment productivities, mine ventilation, mine electrical and communications/monitoring, ore haulage, and other underground infrastructure requirements at the FS adequacy level.

There are sufficient M&I Mineral Resources outside the 50-Year Mine Plan to support extending the mine life beyond the 50-year FS scope.

The geotechnical characteristics of polyhalite ore make the mining of the ore challenging. Testing by continuous miner manufacture and an independent rock mechanics testing laboratory confirmed the suitability of using heavy-duty, high-powered drum-type continuous miners for polyhalite production.

To mitigate the risk of methane incursion into the mine workings, the mine is designed to MSHA metal/non-metal gassy mine regulatory requirements and certain best practices used in coal mines for ventilation and mining near gas and oil wells.

25.4 Processing

The laboratory and pilot plant testing and design work performed during the FS concludes that the processing of polyhalite ore into salable SOP products is technically feasible, based on the process outlined in the FS. There is sufficient flexibility built into the production capacity of each of the three SOP product streams (soluble, standard, and granular) to meet the projected sales requirements.

The plant is designed to operate 7,912 hours annually. The plant model projects a K_2O process recovery of 82.2% based on the pilot test work carried out by independent consultants and equipment providers. As a result of the pilot test work, the FS projects an SOP product with potassium content, or K_2O equivalent, between 50.3% and 53.7%.

25.5 Risks

A two-dimensional risk assessment process was conducted by key FS team members to identify areas of potentially significant Project risks. A risk register resulted from the FS team members' assessment of the risks. Mitigation measures were incorporated into the FS that should have reasonable probability of reducing these risks to acceptable levels were incorporated into the FS.

Risk management activities were undertaken on the following key areas of the Project:

- Geology and mining
- Infrastructure and services
- Processing: plant, tailings, and loadout
- Environment
- Construction
- Financial
- Community relations
- Government and regulatory requirements
- Human resources, security, health and safety

Based on the results of the risk identification workshops and subsequent post-workshop revisions, the Project has a risk profile consistent with the current state of development for a new commercial process. Attendees for the risk assessment workshops were selected based upon their experience and role in the project team.

The workshop was structured to provide opportunity for the risk events to be identified and analyzed using the standard format for the workshop as follows:

- Introductory session
- Risk event identification and review
- Risk ranking (i.e., the selection of probability, consequence, and manageability)
- Analysis of current controls
- Identification of new and/or amended controls

All threats that had been identified as having a potential financial impact to the Project, and which were evaluated during the FS, were considered part of the financial analysis and divided into the categories of CAPEX and OPEX. The threats in each category were subjected to a Monte Carlo analysis using @RISK™ software.

Risk management is a continuous process that is performed over the full life-cycle of a project; therefore, risk management is only complete when the project is complete. Consequently, the data and information in this risk register is a snapshot of the Project risk profile as understood at the completion of the FS. Because of the continuous nature of the risk management process, open risk issues exist at this time, not all risks have been fully evaluated and not all risks are accompanied by mitigation plans or actions. This is to be expected at this stage in the Project.

The sums of all residual CAPEX threats are shown in Figure 25-1. The first chart, a histogram, shows the distributions of all Monte Carlo outcomes. The horizontal axis is in million USD, while the vertical axis is the proportion of outcomes associated with each vertical bar of the histogram. The total area under the curve is unity (i.e., all potential outcomes). The second chart represents the risk allowance for any required level of confidence that the risk allowance will not exceed. The horizontal axis represents the risk allowance in million USD and the vertical axis represents the probability of spending less than that amount.

The equivalent OPEX results to the CAPEX results discussed in the previous section are shown in Figure 25-2.

25.6 Opportunities

25.6.1 General

If local labor availability increases due to slowdowns in other industries in the area (gas and oil, construction), then reduced construction and operating costs may be possible.

25.6.2 Mining

Increased extraction ratios in the mine’s production panels may be plausible given the results of the mine geotechnical modeling conducted for the FS. Increased extraction ratios would reduce overall mine expansion costs over time, and delay certain future OPEX and CAPEX for several years.

Value engineering was conducted on various aspects of the mine design throughout development of the FS to reduce CAPEX and OPEX. These efforts should be continued during the detailed design phase. Potential areas for further investigation are:

- Blind-drilled shafts (at least two contractors are capable of blind drilling shafts)
- Smaller, multiple shafts that can be blind- or raised-drilled
- Slope configuration: side-by-side versus arched-back over/under

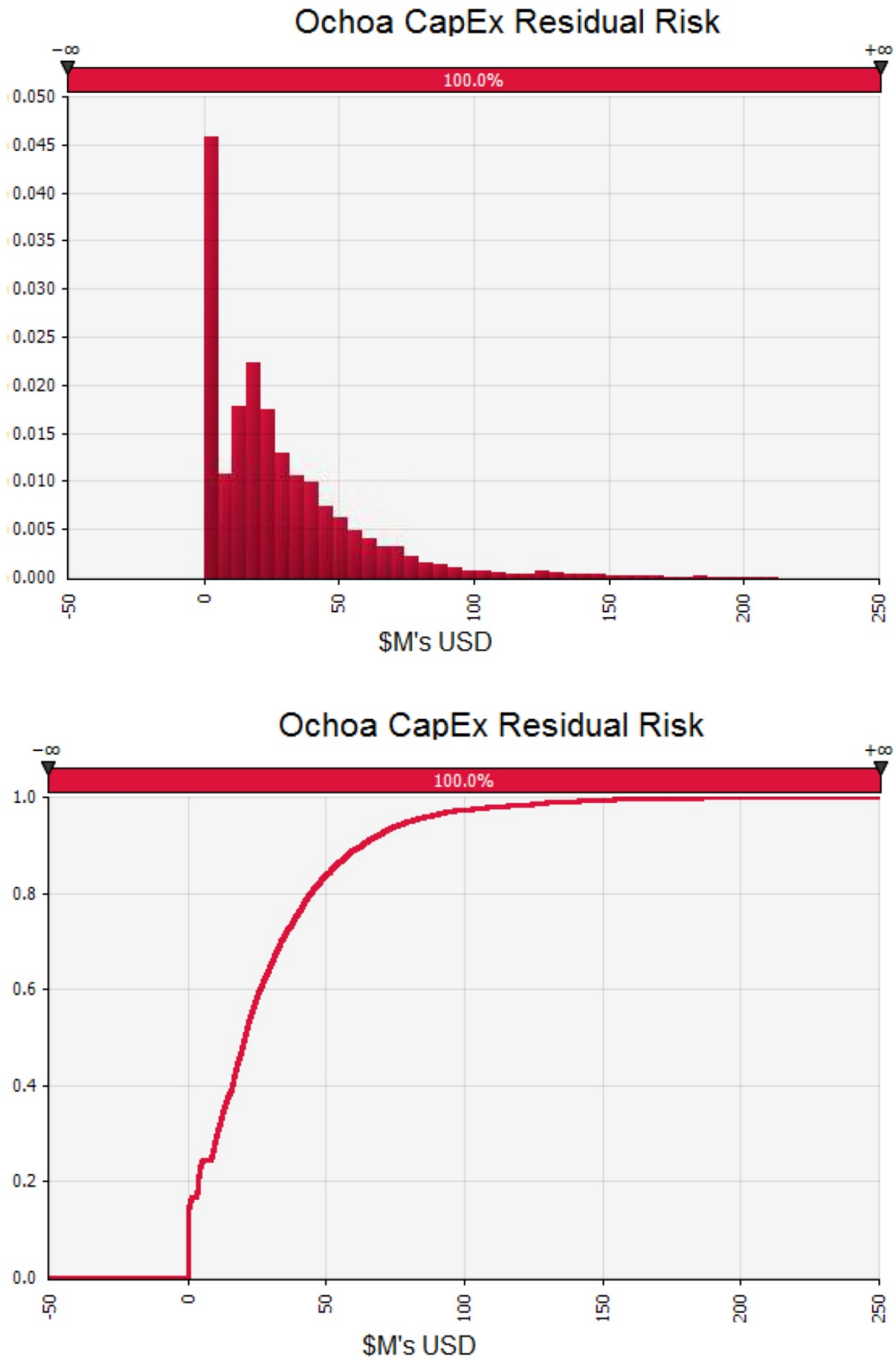


Figure 25-1. CAPEX Risk Allowance (from SNCL 2014, Figure 24.1, p. 24-6)

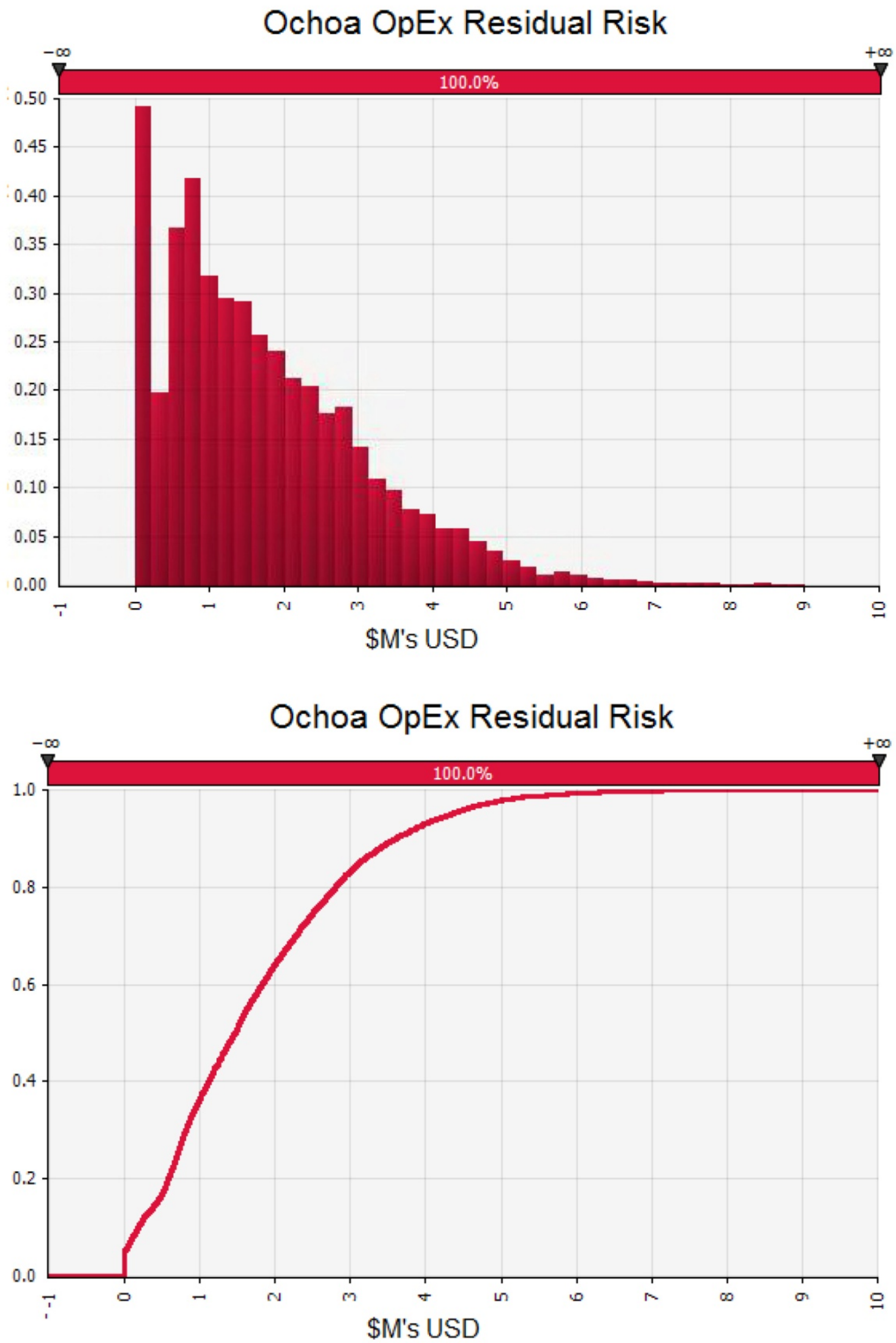


Figure 25-2. OPEX Risk Allowance (from SNCL 2014, Figure 24.2, p. 24-8)

- Additional tunnel boring machine contractor quotes to confirm that roadheader slope development is the best option for slope construction
- ICP's development of the rock portion of the slope and installation of the slope portion of the slope conveyor structure, and contractor assistance with the installation of the intermediate slope drives

25.6.3 Processing and Infrastructure

Alternate by-products may possibly be produced (gypsum, epsonite, kieserite), potentially increasing the profitability of the Project.

ITEM 26: RECOMMENDATIONS

26.1 General

The FS recommends that the Company move to implementation by:

- Commencing EPCM activities
- Completing environmental permitting
- Arranging Project financing

26.2 Mining

Specific recommendations for mining are as follows:

1. During 2014, ICP should have the CSM EMI complete a continuous miner cutter head modeling study. This study will sum all the forces acting on the cutter drum as it rotates. The summation of these forces can then be compared to the continuous miner manufacturers' specification to permit the selection of the most appropriate machine for the mining conditions. The model can also address concerns about bit shank survivability. The anticipated cost for this study is less than $25,000.
2. Proceed with detailed mine design, mine substation design, detailed slope belt conveyor design, and the selection of the shaft and slope contractor, and the final design of the shaft and slope. The estimated cost is $2.2 million USD for 2014, excluding Priorities 3 and 4 of the geotechnical testing program.
3. Arrange for a meeting in early 2014 with the appropriate MSHA district manager to discuss the Project in detail, and confirm that the proposed mine plans are acceptable to MSHA.
4. Conduct additional geotechnical modeling in 2014 to determine whether the production panel extraction ratio percentage could be increased to greater than 60%, and thereby reduce projected mine OPEX. The estimated cost is $50,000.
5. Design a monitoring program for surface subsidence and an underground geotechnical monitoring program of data collection and analysis. The design cost is estimated to be $8,500 for the subsidence monitoring plan and $14,000 for the underground monitoring plan.
6. Conduct FLAC3D modeling of gas and oil well casing and various well protective barrier pillar sizes, using updated extraction ratios. The estimated cost is $30,000.
7. Include Priorities 3 and 4 of the mine geotechnical testing program as part of any exploration drilling conducted in 2014 or later. The estimated cost of mine geotechnical testing is $130,000, exclusive of drilling costs.

26.3 Processing

Specific recommendations for processing are discussed in the following subsections.

26.3.1 Process Design Finalization

During the bridge engineering phase, several process activities will be carried out to advance the design of the Project to a level where implementation can successfully be initiated.

These process finalization activities will be focused on resolving issues identified late in the FS phase, such as finalizing the design and revising the process flow diagrams to take forward into detailed design. The following is a summary of these activities:

1. During pilot testing, sub-10 micron particles appeared in the first-stage leach brine. These may be eliminated during subsequent processes downstream. To ensure a clear brine, extra clarification equipment was added to the circuit. After testing is performed in the detailed and bridge engineering phase, this filtration equipment may be removed or modified, depending on the results. The anticipated cost for this testing work is $25,000 with an additional $20,000 of design work anticipated.
2. Additional heat is required in the leaching circuit. Adding either heat exchangers or steam sparge tubes are two possible solutions. A larger boiler system may also be required. These changes are not reflected in the process flow diagrams or in the CAPEX. Additional design work is required to determine an adequate solution. Additional natural gas for a larger boiler system has been included in OPEX. The anticipated cost for this design work is $20,000.

26.3.2 Process Optimization

Optimization activities will be focused on revising the design to lower the CAPEX of the Project while improving the technical capability of the design put forward. The following is a summary of these activities:

1. Quotations for all major equipment will be reevaluated during bridge and detailed design. This includes the fluid bed calciner units, which are one of the larger equipment costs for the Project. Final vendor selection will be based on the most economic equipment meeting the technical requirements. The anticipated cost for this engineering and procurement work during the bridge engineering phase is $220,000.
2. Further trade-off studies and value engineering activities should be conducted to finalize designs and layouts. The anticipated cost for this work during the bridge engineering phase is $100,000.
3. Results from the pilot work, which was completed at the end of the FS provided data indicating room for optimization of the second-stage leach circuit. Review of these test results and redesign of the second-stage leach circuit is also recommended to be studied. The anticipated cost for this design work is $50,000.

26.3.3 TSF

Recommendations for future work required to complete the detailed design of the TSF are listed below:

1. The design of the TSF is based on assumed material properties for gypsum tailings. Laboratory tests should be completed to adequately characterize the tailings and the design and recommendations should be refined accordingly. The anticipated cost for this laboratory testing is $20,000.
2. The design of the TSF is based on assumed material properties for the foundation soils. Laboratory tests should be completed to adequately characterize the foundation soils, especially strength properties, and the design and recommendations should be refined accordingly. The estimated cost for this laboratory testing is $40,000.

3. The number of boreholes located in the proposed TSF area, especially in the south of the TSF is limited. Additional boreholes are required to adequately define the stratigraphy and hydrogeology in the vicinity of the TSF. Additionally, the engineering properties of the foundation materials should be further investigated with sampling and testing programs (in situ and laboratory). For drilling investigation planning, field supervision and reporting estimated cost is $30,000. This does not include expenses (travel, accommodations, sustenance) or disbursements (drilling subcontractor, materials, courier/shipping, etc.).

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CERTIFICATE of AUTHOR

I, Gary L. Skaggs, do hereby certify that:

1. I am currently employed as Vice President, Principal; Agapito Associates, Inc. located at 715 Horizon Drive, Suite 340, Grand Junction, Colorado 81506, USA.

2. I graduated in 1969 with a Bachelor of Science degree in Mining Engineering from Virginia Polytechnic Institute (Virginia Tech), Blacksburg, Virginia, USA, and in 1986 with a Masters of Business Administration degree, Executive Program, from the University of Denver, Denver, Colorado, USA. I have practiced my profession since 1969.

3. I am a licensed professional engineer in the states of Alabama (24632), Colorado (24551), Idaho (15833), Illinois (062-039549), Kentucky (21299), Missouri (E2000152287), Montana (14359PE), Nevada (15060), New Mexico (14806), Ohio (E-46141), Utah (4879147-2202), Virginia (0402-034807), West Virginia (008155), Wyoming (8934), and the Provinces of Alberta (M83174) and Saskatchewan (23765), Canada.

4. I am a Registered Member (2974570) of the Society of Mining, Metallurgy and Exploration, Inc., Englewood, Colorado, USA.

5. I have worked as a mining engineer for 44 consecutive years since graduation from Virginia Tech, with 26 years' experience working for mining companies in engineering, operations, and executive management, and 18 years consulting practice.

6. As a consulting engineer, I have completed mineral resource and mineral reserve estimates, mine planning and design, scoping, prefeasibility, and feasibility studies, infrastructure design for metals, coal, limestone, and the industrial minerals of trona, potash (USA and Canada), and phosphate rock projects. Extraction methods which I have experience with include room-and-pillar, longwall, stoping and backfill, augering, and surface mining,

7. I have more than 5 years' senior technical and general managerial responsibility entailing the exercise of independent judgment in mining operations and in consulting.

8. I have read the definition of "qualified person" set out in NI 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

9. I am co-author of the report titled "NI 43-101 Technical Report, Ochoa Project Feasibility Study, Lea County, New Mexico," effective date January 9, 2014 (the "Technical Report"), being responsible for Items 1.1, 1.2, 1.3.2, 1.4, 1.7.2, 1.8.1, 1.9.1, 1.9.2, 1.10, 2, 3.1–3.7 , 4, 5, 6, 7.6, 15, 16, 19, 20, 21.1.2, 21.1.7.3, 23, 24, 25.1, 25.3, 25.6.2, 26.2, and 27. I have jointly reviewed and edited the entire document.

10. I undertook a site visit to the property September 19 and 20, 2012.

11. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the parts of this Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

12. I do not hold, nor do I expect to receive, any securities or any other interest in any corporate entity, private or public, with interests in the properties that are the subject of this report or in the properties themselves, nor do I have any business relationship with any such entity apart from a professional consulting relationship with the issuer, nor to the best of my knowledge do I have any interest in any securities of any corporate entity with property within a 2 mile distance of any of the subject properties.

13. I am independent of Intercontinental Potash Corp, according to the criteria stated in Section 1.5 of NI 43-101.

14. I have read NI 43-101 and Form NI 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated this 7th day of March 2014.

/s/ Gary L. Skaggs (Signature)
Signature of Qualified Person

SEAL

Gary L. Skaggs, P.E. (New Mexico)
Print name of Qualified Person

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CERTIFICATE of AUTHOR

I, Leo J. Gilbride, do hereby certify that:

1. I am currently employed as Vice President, Agapito Associates, Inc. located at 715 Horizon Drive, Suite 340, Grand Junction, Colorado 81506, USA.

2. I graduated with a degree in Civil Engineering *summa cum laude* from California Polytechnic State University, San Luis Obispo, California, USA, in 1992, and a Master of Science in Mining Engineering at the Mackay School of Mines, University of Nevada, Reno, USA, in 1995.

3. I am licensed as a professional engineer in the State of Colorado (Number 33329).

4. I am a member of the Society of Mining, Metallurgy and Exploration, Inc. (Member Number 4028449) and the American Society of Civil Engineers (Member Number 271529).

5. I have practiced as a consulting mining engineer for 17 years since graduation from the Mackay School of Mines, University of Nevada, Reno, in 1995.

6. As a consulting engineer, I have completed mineral resource and mineral reserve estimations, and scoping, prefeasibility, and feasibility studies in industrial minerals, metals and coal, including trona, potash, nahcolite, phosphate, uranium, vanadium, molybdenum, cobalt and nickel. Extraction methods with which I have experience include room-and-pillar, longwall, drift-and-fill, open stoping, block caving, open pit, and solution mining.

7. I have consulted on projects for more than one dozen underground mines located in the western USA in the last 5 years.

8. I have read the definition of “qualified person” set out in NI 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.

9. I am co-author of the report titled “NI 43-101 Technical Report, Ochoa Project Feasibility Study, Lea County, New Mexico,” effective date January 9, 2014 (the “Technical Report”), being responsible for Items 1.7.1 and 14.

10. I undertook a site visit to the property September 26 through 28, 2012.

11. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the parts of this Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

12. I do not hold, nor do I expect to receive, any securities or any other interest in any corporate entity, private or public, with interests in the properties that are the subject of this report or in the properties themselves, nor do I have any business relationship with any such entity apart from a professional consulting relationship with the issuer, nor to the best of my knowledge do I have any interest in any securities of any corporate entity with property within a 2 mile distance of any of the subject properties.

13. I am independent of Intercontinental Potash Corp, according to the criteria stated in Section 1.5 of NI 43-101.

14. I have read NI 43-101 and Form NI 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated this 7th day of March 2014.

/s/ Leo J. Gilbride (Signature)
Signature of Qualified Person

SEAL

Leo J. Gilbride, P.E. (Colorado)
Print name of Qualified Person

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CERTIFICATE of AUTHOR

I, Vanessa Santos, do hereby certify that:

1. I am currently employed as Chief Geologist, Agapito Associates, Inc. located at 715 Horizon Drive, Suite 340, Grand Junction, Colorado 81506, USA.

2. I graduated with a Bachelors of Science degree in Geology in 1981 and a Masters of Science degree in Geology in 1983 from the University of Kentucky, Lexington, Kentucky, USA.

3. I am licensed as a professional geologist in the state of Georgia (1664) and South Carolina (2403).

4. I am a registered member of the Society of Mining, Metallurgy and Exploration, Inc. (Member Number 4058318).

5. I have practiced as a geologist for 30 years since graduation from the University of Kentucky and have 15 years of experience as a geologist and 15 years as a consulting geologist with industrial minerals, coal, and aggregate mining and exploration companies.

6. As a geologist, I have worked in all facets of mining and exploration: evaluation, geologic reconnaissance, field mapping, drilling/coring, ore zone definition, geologic modeling and reserve estimation, QA/QC in minerals and commodities including potash, phosphate, trona, lithium, mica, feldspar, high purity quartz, and phlogopite, industrial sand, talc, limestone, dolomite, crushed stone, kaolin, ball and specialty clays and alluvial and marine diamonds.

7. I have worked on multiple industrial minerals projects, including phosphate and potash, in North America, South America, Europe and Africa in the last 5 years.

8. I have read the definition of "qualified person" set out in NI 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

9. I am co-author of the report titled "NI 43-101 Technical Report, Ochoa Project Feasibility Study, Lea County, New Mexico," effective date January 9, 2014 (the "Technical Report"), being responsible for Items 1.3.1, 1.5, and 7 through 12.

10. I undertook a site visit to the property September 25 through 28, 2012.

11. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the parts of this Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

12. I do not hold, nor do I expect to receive, any securities or any other interest in any corporate entity, private or public, with interests in the properties that are the subject of this report or in the properties themselves, nor do I have any business relationship with any such entity apart from a professional consulting relationship with the issuer, nor to the best of my knowledge do I have any interest in any securities of any corporate entity with property within a 2 mile distance of any of the subject properties.

13. I am independent of Intercontinental Potash Corp, according to the criteria stated in Section 1.5 of NI 43-101.

14. I have read NI 43-101 and Form NI 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated this 7th day of March 2014.

/s/ Vanessa Santos (Signature)
Signature of Qualified Person

SEAL

Vanessa Santos, P.G. (Georgia)
Print name of Qualified Person

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CERTIFICATE of AUTHOR

I, Susan B. Patton, do hereby certify that:

1. I am currently employed as Senior Associate, Agapito Associates, Inc. located at 715 Horizon Drive, Suite 340, Grand Junction, Colorado 81506, USA.

2. I graduated with a Bachelor's degree in Mining Engineering from New Mexico Tech, Socorro, New Mexico, USA in 1983; and a Masters of Mineral Engineering degree from the University of Alabama, Tuscaloosa, Alabama, USA in 1989; and an Interdisciplinary Doctor of Philosophy degree in Mining and Environmental Engineering from the University of Alabama, Tuscaloosa, Alabama, USA in 1993. I have practiced my profession since 1983.

3. I am a licensed professional engineer in the states of Alabama (19875), Colorado (30176), and Montana (14542).

4. I am a Member (2482200) of the Society of Mining, Metallurgy and Exploration, Inc., Englewood, Colorado, USA.

5. I have worked as a mining engineer for 29 years since graduation from New Mexico Tech. Positions include academic, and research faculty, academic administration, mine site engineering, and consulting.

6. As a consulting engineer, I have completed mineral resource and mineral reserve evaluations, mine planning, prefeasibility and feasibility studies, and ventilation design for metal, non-metal and coal. Extraction methods with which I have mine site experience include room-and-pillar and open cast.

7. I have more than 5 years' senior technical and general managerial responsibility entailing the exercise of independent judgment in mining operations and in consulting.

8. I have read the definition of "qualified person" set out in NI 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

9. I am co-author of the report titled "NI 43-101 Technical Report, Ochoa Project Feasibility Study, Lea County, New Mexico," effective date January 9, 2014 (the "Technical Report"), being responsible for Items 16.2.4, 16.7, 21.1.7.3, and 21.2.2.

10. I did not undertake a site visit to the property.

11. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the parts of this Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

12. I do not hold, nor do I expect to receive, any securities or any other interest in any corporate entity, private or public, with interests in the properties that are the subject of this report or in the properties themselves, nor do I have any business relationship with any such entity apart from a professional consulting relationship with the issuer, nor to the best of my knowledge do I have any interest in any securities of any corporate entity with property within a 2 mile distance of any of the subject properties.

13. I am independent of Intercontinental Potash Corp, according to the criteria stated in Section 1.5 of NI 43-101.

14. I have read NI 43-101 and Form NI 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated this 7th day of March 2014.

/s/ Susan B. Patton (Signature)
Signature of Qualified Person

SEAL

Susan B. Patton, P.E. (Colorado)
Print name of Qualified Person

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CERTIFICATE of AUTHOR

I, Thomas L. Vandergrift, do hereby certify that:

1. I am currently employed as Principal, Agapito Associates, Inc. located at 1536 Cole Boulevard, Suite 220, Lakewood, Colorado 80401, USA.

2. I graduated in 1985 with a Bachelor of Science degree in Mining Engineering from the Colorado School of Mines, Golden, Colorado, USA, and in 1992 with a Master of Science degree in Mining Engineering from the same institution. I have practiced my profession since 1986.

3. I am a licensed professional engineer in the State of Colorado (36213).

4. I am a Registered Member (4105017) of the Society of Mining, Metallurgy and Exploration, Inc., Englewood, Colorado, USA.

5. I have worked as a mining engineer for 27 consecutive years since graduation from the Colorado School of Mines, with 9 years' experience working in government mining research, and 18 years consulting practice.

6. As a consulting engineer, I have completed or contributed to geotechnical, mine planning and design, environmental impact, scoping, prefeasibility and feasibility studies for coal, metals, and industrial minerals projects. Extraction methods with which I have experience include room-and-pillar, longwall, stoping and backfill, highwall, and surface mining.

7. I have more than 5 years' senior technical and general managerial responsibility entailing the exercise of independent judgment in mining consulting.

8. I have read the definition of "qualified person" set out in NI 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

9. I am co-author of the report titled "NI 43-101 Technical Report, Ochoa Project Feasibility Study, Lea County, New Mexico," effective date January 9, 2014 (the "Technical Report"), being responsible for Item 16.2.1.

10. I did not undertake a site visit to the property.

11. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the parts of this Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

12. I do not hold, nor do I expect to receive, any securities or any other interest in any corporate entity, private or public, with interests in the properties that are the subject of this report or in the properties themselves, nor do I have any business relationship with any such entity apart from a professional consulting relationship with the issuer, nor to the best of my knowledge do I have any interest in any securities of any corporate entity with property within a 2 mile distance of any of the subject properties.

13. I am independent of Intercontinental Potash Corp, according to the criteria stated in Section 1.5 of NI 43-101.

14. I have read NI 43-101 and Form NI 43-101F1, and the portions of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

Dated this 7th day of March 2014.

/s/ Thomas L. Vandergrift (Signature)
Signature of Qualified Person

SEAL

Thomas L. Vandergrift, P.E. (Colorado)
Print name of Qualified Person

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CERTIFICATE of AUTHOR

I, Lawrence Berthelet, do hereby certify that:

1. I am currently employed as Vice President, Potash, SNC-Lavalin Inc. located at 216 – 1st Avenue South, Saskatoon, Saskatchewan, S7K 1K3, Canada.

2. I graduated in 1986 with a Bachelor of Science degree in Chemical Engineering from the University of Saskatchewan, Saskatoon, Saskatchewan, Canada, and in 2012 with a Masters of Business Administration degree, Process Program, from the Edwards School of Business, Saskatoon, Saskatchewan, Canada. I have practiced my profession since 1987.

3. I am a licensed professional engineer in the province of Saskatchewan (06268), Canada, and a member in good standing since 1990.

4. I have worked as a professional engineer for 27 consecutive years since graduation from the University of Saskatchewan, working for mining companies in engineering, operations, and executive management.

5. As a consulting engineer, I have completed scoping, prefeasibility and feasibility level studies for potash projects in Canada. I have managed the execution of potash project in Canada, and have completed due diligence reviews on potash projects in Canada and Russia.

6. I have more than 15 years' senior technical and general managerial responsibility entailing the exercise of independent judgment in mining operations and in consulting.

7. I have read the definition of "qualified person" set out in NI 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

8. I am co-author of the report titled "NI 43-101 Technical Report, Ochoa Project Feasibility Study, Lea County, New Mexico," effective date January 9, 2014 (the "Technical Report"), being responsible for Items 1.8.3, 1.9.1, 3.4, 18, 21.1 (except 21.1.2.1 and 21.1.7.3), 21.2 (except 21.2.2), 22, 25.2, 25.4, 25.5, 25.6.1, 25.6.3, 26.1, 26.3, and 27.

9. I undertook a site visit to the property on November 28, 2012 and February 12, 2013.

10. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical Report that I am responsible for

contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

11. I do not hold, nor do I expect to receive, any securities or any other interest in any corporate entity, private or public, with interests in the properties that are the subject of this report or in the properties themselves, nor do I have any business relationship with any such entity apart from a professional consulting relationship with the issuer, nor to the best of my knowledge do I have any interest in any securities of any corporate entity with property within a 2 mile distance of any of the subject properties.

12. I am independent of Intercontinental Potash Corp, according to the criteria stated in Section 1.5 of NI 43-101.

13. I have read NI 43-101 and Form NI 43-101F1, and the parts of the Technical Report that I am responsible for have been prepared in compliance with that instrument and form.

Dated this 7th day of March 2014.

/s/ Lawrence Berthelet (Signature)
Signature of Qualified Person

SEAL

Lawrence Berthelet, P.Eng. (Saskatchewan)
Print name of Qualified Person

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CERTIFICATE of AUTHOR

I, Jack M. Nagy, do hereby certify that:

1. I am currently employed as Process Manager, SNC-Lavalin, located at 216-1st Avenue South, Saskatoon, Saskatchewan, Canada S7K 1K3.

2. I graduated in 1975 with a Bachelor of Science degree in Chemical Engineering from the University of Saskatchewan, Saskatoon, Saskatchewan, Canada. I have practiced my profession since 1975.

3. I am a licensed professional engineer in the Province of Saskatchewan (05065), Canada, and have been a member in good standing since 1981.

4. I have worked as a process/chemical engineer for 38 consecutive years since graduation from the University of Saskatchewan, with 37 years' experience working for mining companies in engineering, operations, and executive management, and 1 year consulting practice.

5. As a consulting engineer, I have completed scoping, prefeasibility, and feasibility studies for potash (USA, Russia and Canada), and phosphate rock projects.

6. I have more than 5 years' senior technical and general managerial responsibility entailing the exercise of independent judgment in processing operations and in consulting.

7. I have read the definition of "qualified person" set out in NI 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

8. I am co-author of the report titled "NI 43-101 Technical Report, Ochoa Mine Project, Lea County, New Mexico," effective date January 9, 2014 (the "Technical Report"), being responsible for Items 1.6, 1.8.2, 1.9.3, 3.8, 13, and 17.

9. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

10. I did not undertake a site visit to the property.

11. I do not hold, nor do I expect to receive, any securities or any other interest in any corporate entity, private or public, with interests in the properties that are the subject of this report or in the properties themselves, nor do I have any business relationship with

any such entity apart from a professional consulting relationship with the issuer, nor to the best of my knowledge do I have any interest in any securities of any corporate entity with property within a 2 mile distance of any of the subject properties.

12. I am independent of Intercontinental Potash Corp, according to the criteria stated in Section 1.5 of NI 43-101.

13. I have read NI 43-101 and Form NI 43-101F1, and the parts of the Technical Report that I am responsible for have been prepared in compliance with that instrument and form.

Dated this 7th day of March 2014.

/s/ Jack M. Nagy (Signature)
Signature of Qualified Person

SEAL

Jack M. Nagy, P.Eng. (Saskatchewan)
Print name of Qualified Person