TECHNICAL REPORT
RESOURCES AND RESERVES OF THE BLAWN MOUNTAIN PROJECT
BEAVER COUNTY, UTAH

Submitted to:
POTASH RIDGE CORPORATION

Report Date:
December 2, 2013

Report Effective Date:
November 6, 2013

Norwest Corporation
136 E. South Temple, 12th Floor
Salt Lake City, UT
84111 USA
Tel: (801) 539-0044
Fax: (801) 539-0055
Email slc@norwestcorp.com

www.norwestcorp.com

Authors:
STEVEN KERR, CPG, PG
LAWRENCE D. HENCHEL, P.GEO, PG.
JASON N. TODD, QP
ROBERT I. NASH, PE
L. RAVINDRA NATH, QP

NORWEST
CORPORATION

CERTIFICATE OF QUALIFICATIONS

I, Steven B. Kerr, CPG, PG of Salt Lake City, Utah, do hereby certify that:

1. I am currently employed as a Geologic Project Manager by Norwest Corporation, 136 East South Temple, Suite 1200, Salt Lake City, Utah, USA 84111.
2. I attended the Utah State University where I earned a Bachelor of Science degree in Geology in 1981 and a Master of Science degree in Geology in 1987.
3. I am a Certified Professional Geologist with the American Institute of Professional Geologists (CPG-10352). I am licensed as a Professional Geologist in the states of Alaska (# 512), Utah (#5557442-2250) and Wyoming (PG-2756).
4. I have worked as a geologist for a total of twenty-eight years since my graduation from university, working with companies involved in the mining and exploration of metal and industrial mineral deposits in the western United States. As a consultant I have worked on worldwide projects involving Ag/Pb/Zn vein, bauxite, coal, Cu/Au skarn, disseminated, Archaen, and placer gold deposits; iron ore, limestone, mineral sands, oil shale, trona, volcanic-hosted vein deposits, and uranium.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with professional associations (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.
6. I am responsible for the preparation of all sections (excepting Sections 13 through 22) of the technical report titled "Technical Report Resources and Reserves of the Blawn Mountain Project, Beaver County, Utah" dated November 20, 2013 (Technical Report) relating to the Blawn Mountain Alunite Property, with an **Effective Date of November 6, 2013**
7. I personally visited and inspected the Blawn Mountain Property on several occasions since 2012. The first visit to property occurred on February 9, 2012 and the most recent visit to the property was on April 11, 2013.
8. I previously contributed to the preparation of two technical reports on the Blawn Mountain Project which were titled: (1) "Technical Report Preliminary Economic Assessment, Blawn Mountain Project, Beaver County, Utah" dated November 5, 2012; and (2) "Blawn Mountain Project, Beaver County, Utah" dated April 16, 2012.
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 for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10. I am independent of the issuer applying all of the tests in Section 1.5 of NI 43-101.
11. I have read NI 43-101 and the Technical Report, and the parts of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.

Dated at Salt Lake City, Utah this 22nd Day of November, 2013.

"ORIGINAL SIGNED AND SEALED BY AUTHOR"

Steven B. Kerr, CPG, PG
Project Manager Geologic Services, Norwest Corporation

CERTIFICATE OF QUALIFICATIONS

I, Lawrence D. Henchel, P.Geo., do hereby certify that:

1. I am currently employed as Vice President of Geologic Services by Norwest Corporation, Suite 1200, 136 East South Temple Street, Salt Lake City, Utah 84111 USA.
2. I graduated with a Bachelor of Science Degree in Geology from Saint Lawrence University, Canton, NY, USA in 1978.
3. I am a licensed Professional Geoscientist in the province of Alberta, Canada, #159013. I am a licensed Professional Geologist in the State of Utah, #6087593-2250 and I am a Registered Member of The Society for Mining, Metallurgy and Exploration, Inc., #4150015RM.
4. I have worked as a geologist for a total of thirty years since my graduation from university, both for coal mining and exploration companies and as a consultant specializing in coal and industrial minerals. I have worked with industrial minerals such as potash, trona, nahcolite, phosphate and gypsum over the past 20 years of my career in the United States, Mongolia, Africa and the Middle East. My experience with potash includes exploration, geological modeling and resource estimation for bedded deposits, SOP from alunite alteration and from mineral brines.
5. I have read the definition of “qualified person” set out in National Instrument 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.
6. I am responsible for Section 14 of the technical report titled, “Technical Report Resources and Reserves of the Blawn Mountain Project, Beaver County, Utah” dated November 20, 2013 (Technical Report) relating to the Blawn Mountain Alunite Property, with an **Effective Date of November 6, 2013**.
7. I personally inspected the Blawn Mountain Alunite Property on October 30, 2012 and on January 20 and 21, 2013.
8. I previously contributed to the preparation of the technical report on the Blawn Mountain Project which was titled “Technical Report Preliminary Economic Assessment, Blawn Mountain Project, Beaver County, Utah” dated November 5, 2012.
9. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10. I am independent of the issuer applying all of the tests in Section 1.5 of NI 43-101.
11. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated at Salt Lake City, Utah this 22nd Day of November, 2013.

“ORIGINAL SIGNED AND SEALED BY AUTHOR”

Lawrence D. Henchel, PG, P.Geo.
Vice President Geologic Services, Norwest Corporation

CERTIFICATE OF QUALIFICATIONS

I, Jason N. Todd, of Salt Lake City, Utah, do hereby certify that:

1. I am an Engineering Project Manager with Norwest Corporation, 136 East South Temple, 12th Floor, Salt Lake City, Utah, 84111 USA.
2. I attended Montana Tech of the University of Montana where I earned a Bachelor of Science degree in Mining Engineering in 1998.
3. I am a Qualified Professional Member of the Mining and Metallurgical Society of America, Member #01414QP. I am a “qualified person” for the purposes of National Instrument 43-101.
4. I have worked as a mining engineer for a total of fifteen years since my graduation from university for mining companies and as a consultant specializing in coal and industrial minerals.
5. I have read the definition of “qualified person” set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with professional associations (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.
6. I am responsible for the preparation of Sections 15, 16, 18 through 20, and 22 of the report, titled, “Technical Report Resources and Reserves of the Blawn Mountain Project, Beaver County, Utah” dated November 20, 2013 (Technical Report) relating to the Blawn Mountain Alunite Property, with an **Effective Date of November 6, 2013**.
7. I personally inspected the Blawn Mountain Property on March 15, 2012 and February 12, 2013.
8. I previously contributed to the preparation of two technical reports on the Blawn Mountain Project which were titled: (1) “Technical Report Preliminary Economic Assessment, Blawn Mountain Project, Beaver County, Utah” dated November 5, 2012; and (2) “Blawn Mountain Project, Beaver County, Utah” dated April 16, 2012.
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 for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical not misleading.
10. I am independent of the issuer applying all of the tests in Section 1.5 of NI 43-101.
11. I have read NI 43-101 and Form 43-101 (F1), and the Technical Report has been prepared in compliance with that instrument and form.

Dated at Salt Lake City, Utah this 22nd Day of November, 2013.

“ORIGINAL SIGNED AND SEALED BY AUTHOR”

Jason N. Todd, QP
Project Manager, Norwest Corporation

CERTIFICATE OF QUALIFICATIONS

I, Robert I. Nash, PE, of Salt Lake City, Utah, do hereby certify that:

1. I am currently employed and serve as a Principal of Intermountain Consumer Professional Engineers, Inc., 1145 East South Union Avenue, Midvale, Utah, USA 84047.
2. I attended the Brigham Young University where I earned a Bachelor of Science degree in Mechanical Engineering in 1985.
3. I have worked as a licensed professional engineer (1988) for a total of twenty-six years since my graduation from a university, for companies and projects involved with the processing of metals, specialty and precious metals, and mineral-based projects.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with professional associations (as defined in NI 43-101) and past relevant experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I am responsible and have responsible charge for the preparation of Sections 13, 17, and 21 of the report titled "Technical Report Resources and Reserves of the Blawn Mountain Project, Beaver County, Utah" dated November 20, 2013 (the "Technical Report") relating to the Blawn Mountain Alunite property, with an **Effective Date of November 6, 2013**.
6. I personally inspected the Blawn Mountain Property on March 15, 2012.
7. I previously contributed to the preparation of two technical reports on the Blawn Mountain Project which were titled: (1) "Technical Report Preliminary Economic Assessment, Blawn Mountain Project, Beaver County, Utah" dated November 5, 2012; and (2) "Blawn Mountain Project, Beaver County, Utah" dated April 16, 2012.
8. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
9. I am independent of the issuer applying all of the tests in Section 1.5 of NI 43-101.
10. I have read NI 43-101 and the Technical Report, and parts of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.

Dated at Salt Lake City, Utah this 22nd Day of November, 2013.

"ORIGINAL SIGNED AND SEALED BY AUTHOR"

Robert I. Nash, PE

CERTIFICATE OF QUALIFICATIONS

I, L. Ravindra Nath, QP, of Salt Lake City, Utah, do hereby certify that:

1. I am currently employed and serve as Chief Process Engineer of Intermountain Consumer Professional Engineers, Inc., 1145 East South Union Avenue, Midvale, Utah, USA 84047.
2. I attended the University of Mysore, Bangalore, India where I earned a Bachelor of Science degree in Physics, Chemistry and Mathematics in 1956 and attended the S. J. Polytechnic Institute, Bangalore, India where I earned a Diploma in Mining Engineering in 1959.
3. I am a Qualified Professional of the Mining and Metallurgical Society of America, Member #01436QP, and a Legion of Honor Member of the Society for Mining, Metallurgy and Exploration, Member #2337700.
4. I have worked as Process Engineer for a total of over forty years since my graduation from a university, for companies and projects, domestic and abroad, involving smelting, refining, and mineral processing of nonferrous and precious metals, specialty metals, uranium, coal, and industrial minerals.
5. I have read the definition of “qualified person” set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with professional associations (as defined in NI 43-101) and past relevant experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.
6. I am responsible and have responsible charge for the preparation of Sections 13, 17, and 21of the report titled “Technical Report Resources and Reserves of the Blawn Mountain Project, Beaver County, Utah” dated November 20, 2013 (the “Technical Report”) relating to the Blawn Mountain Alunite property, with an **Effective Date of November 6, 2013**.
7. I personally inspected the Blawn Mountain Property on March 15, 2012.
8. I previously contributed to the preparation of the technical report on the Blawn Mountain Project which was titled “Technical Report Preliminary Economic Assessment, Blawn Mountain Project, Beaver County, Utah” dated November 5, 2012.
9. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10. I am independent of the issuer applying all of the tests in Section 1.5 of NI 43-101.
11. I have read NI 43-101 and the Technical Report, and parts of the Technical report for which I am responsible have been prepared in compliance with NI 43-101.

Dated at Salt Lake City, Utah this 22nd Day of November, 2013.

“ORIGINAL SIGNED AND SEALED BY AUTHOR”

L. Ravindra Nath, QP

TABLE OF CONTENTS

LIST OF TABLES

List of Figures

1 SUMMARY

The following technical report was prepared by Norwest Corporation (Norwest) for Potash Ridge Corporation (PRC), a publically traded mineral exploration and development company with corporate offices in Toronto, Ontario, Canada. This report summarizes a Prefeasibility Study (PFS) performed on the Blawn Mountain Project by Norwest and Intermountain Consumer Professional Engineers Inc. (ICPE) that focuses on the mining and processing of alunite ore for the production of sulfate of potash (SOP).

1.1 LOCATION AND TENURE

The Blawn Mountain Project consists of 15,403.72 acres (ac) of Utah State surface and mineral tracts administered by the State of Utah School and Institutional Trust Lands Administration (SITLA). PRC has rights to the property through an Exploration/Option Agreement (ML 51983.0 OBA) and two state mineral leases (ML 52513, and ML 52364) administered by SITLA. The agreement consists of a main tract of land that covers 14,923.72ac and six individual 80ac tracts located 3.5 to 4.5 miles northeast of the main tract of land.

The property is located approximately 30 air miles southwest of the town of Milford, Utah and 30 air miles from the Nevada state border, as shown in Figure 4.1. The area is accessed from Interstate 15 (I-15), the main north-south travel corridor through Utah, by traveling west on the surfaced State Route 21 (SR-21) to the town of Milford, from Milford 24 miles farther west on SR-21, turning south onto a graveled secondary road and traveling approximately 17 miles. The property is located about 20 air miles west of the Union Pacific (UP) Railroad route, running north-south and connecting Salt Lake City with Las Vegas and farther points on the UP rail system.

SITLA lease ML 51983.0 OBA is comprised of 17 full sections and two half sections of SITLA land and potash mineral rights. PRC's entitlement is through the Exploration/Option Agreement with SITLA which was executed in the spring of 2011 and issued to Utah Alunite, LLC, a 100% owned entity of PRC. In April 2012, Utah Alunite, LLC was merged into Utah Alunite Corporation (UAC) and the mineral lease was assigned by Utah Alunite, LLC to UAC.

UAC added five full sections adjacent to the north of ML 51983.0 OBA through the acquisition of SITLA lease ML 52513, effective June 1, 2013. The six individual 80ac parcels were acquired from SITLA under lease ML 52364, effective January 7, 2013.

Two small mineral leases occur within the Blawn Mountain Project, controlled by other parties. One lease is a 40ac tract located along the western edge of the project area and the second lease is a 155ac tract within the boundaries of the project area. Both of these mineral leases are for metallic minerals only and do not include potash mineral rights. Both leases are administered by SITLA.

PRC's Exploration/Option Agreement with SITLA is for a 3-year term and required a front-end payment of $200,000[1], annual payments of $69,300 ($6/ac) and a $1,020,000 bonus for lease issuance which is due on or before March 31, 2014. Primary lease term would be 10 years, renewable in 5-year extensions. Annual rental amounts would apply to the lease agreement as well as 4% gross royalty for metalliferous minerals and 5% for potash minerals. The initial lease terms included a provision to provide a "Positive PFS" to SITLA documenting the project's economic viability, possible markets, mining methods, and potential environmental issues and providing evidence showing the possible continuity and grade of the ore. In May 2013, SITLA provided a letter to PRC stating that the Preliminary Economic Assessment (PEA) completed in November 2012, satisfied the Positive PFS requirement and that PRC could proceed with exercising the option to convert the exploration agreement to a lease.

Mineral leases ML52513, and ML 52364, are standard metalliferrous mineral leases, each with a 10-year primary term and option to extend beyond the primary term. SITLA mineral leases carry an annual rental rate of $1/ac, a gross production royalty of 8% on fissionable minerals and 4% on non-fissionable minerals. Advance royalty payments equating to three times the annual rental rate were assessed at the time of issuance.

1.2 History

Mining operations have been conducted on alunite occurrences in southwest Utah since the early 1900s. Early extraction targeted both potash and aluminum. The Blawn Mountain Project area was the subject of extensive study and exploration activity conducted by a Denver-based exploration and development company, Earth Sciences, Inc. (ESI). However, much of this work targeted the property's aluminum potential. Exploration and geological studies were augmented by mining and processing evaluations as well. They delineated four distinct areas of alunite mineralization at Blawn Mountain, then known as the NG Alunite property. Their historic areas A through D correspond to the PRC nomenclature of Areas 1 through 4. The project was taken to advanced stages of development, but was eventually abandoned due to challenging economic conditions and depressed pricing for alumina and potash in the 1980s.

Previous resource estimates are difficult to relate to current assessments primarily due to focus of past programs on alumina production with potash as a secondary product. Cut-off grades were based on alumina (Al_2O_3) content and therefore skew the potassium (K_2O) estimates since

[1] Currency in US Dollars

potassium was not optimized. Historic estimates ranged from 142.6 million tons (Mt) short to 151.8Mt of in-place alunite resource proven and probable (relates to measured and indicated resource), with corresponding grade estimates of K_2O ranging from 3.85% to 4.15% and of Al_2O_3 ranging from 13.03% to 14.13%. (Walker, 1972; Chapman 1974; Couzens, 1975) None of these studies are deemed to be National Instrument 43-101 (NI 43-101) compliant although reasonable methodologies were applied at the time. Furthermore, a qualified person has not done sufficient work to classify historical estimates as current mineral resources. PRC is not treating the historical (pre-2011) estimates as current mineral resources.

In 2011, PRC initiated a 34-hole drilling program on Area 1 primarily to validate the previous exploration efforts. Norwest (2012) concluded that the PRC validation drilling program had adequately tested the Area 1 deposit, both spatially and in number of twinned drilling locations. A three dimensional geological block model (3DGBM) was constructed by Norwest for Area 1 using historic and PRC validation drilling data. Norwest subsequently used the 3DGBM for reporting of resources for Area 1 in accordance with CIM Standards on Mineral Resources and Reserves, with an effective date April 16, 2012 (Norwest, April 2012). At a 1% K_2O cut-off grade, the combined measured plus indicated resource was estimated to be 162Mt, carrying an average grade of 3.23% K_2O and 13.90% Al_2O_3. The calculated potassium sulfate grade (K_2SO_4) at a 1% K_2O cut-off grade was estimated to be 5.98%. As of April 16, 2012 approximately 66% of the identified resource was classified as measured resource and 34% as indicated resource.

Between July and September, 2012, PRC completed a 50-hole infill drilling program in Area 2 to define the extent of alunite mineralization. Previous exploration in Area 2 was limited to 18 drill holes completed by ESI. Following the 2012 drilling program, Norwest believed there was sufficient geologic and analytical data to support a resource estimate for Area 2 in addition to Area 1. The Area 2 historic resources were reported by Norwest as part of a Preliminary Economic Assessment (PEA) that included Area 1 and Area 2 (Norwest, November 2012).

The measured plus indicated historic resources reported by Norwest in the 2012 PEA were 156.3Mt for Area 1 and 464.4Mt for Area 2. The difference in in-situ tons for Area 1 outlined in this report, (shown in Table 6.4) when compared to previous estimates (Norwest, April 2012) are attributed to the removal of two feet (ft) of surficial material to account for potential weathering on near-surface mineralization.

An open pit conventional truck/shovel method was identified as the preferred mining method for Area 1 and Area 2 in the PEA (2012). Using a target K_2SO_4 requirement of 750,000 tons per year (tpy), a total of 121.6Mt of run of mine (ROM) alunite in Area 1 and 387.9Mt of ROM alunite in Area 2 were identified by Norwest as potentially extractable over a 30 year life of mine (LOM).

There has been no known production of alunite or any other mineral resource from the Blawn Mountain Project.

1.3 GEOLOGICAL SETTING AND MINERALIZATION

The Blawn Mountain Project is located in the southern Wah Wah Mountains, of the eastern Basin and Range province, in an area characterized by a thick Paleozoic sedimentary section that was:

- Thrust faulted during the Sevier Orogeny
- Buried under a thick layer of regionally distributed Oligocene volcanic rocks and locally-derived volcanic rocks
- Extended to the west by the Basin And Range event
- Altered by H_2S rich hydrothermal alteration related to a postulated shallow laccolithic intrusive which domed, and altered the overlying calc-alkaline volcanic rock (Hofstra, 1984)
- Affected by continual erosion of the ranges contributing to colluvial and alluvial deposition in the valleys.

The geologic characterization of the deposit is essentially that of an altered volcanic tuff. The host tuff deposit ranges in thickness from several hundred to one thousand feet at its thickest point. The property is moderately faulted with normal faults related to Basin and Range extensional block faulting. The deposit is controlled by its original alteration geometry, block faulting, and by erosion.

The Blawn Mountain deposit occurs along four ridges, three of which occur within PRC's exploration tracts. Alteration tends to be in linear bodies reflecting the role of normal faults in controlling the mineralization. Alteration is zoned away from the point of hydrothermal fluid upwelling. The mineralized ridges are erosional remnants of a once larger altered area.

1.4 EXPLORATION

The Blawn Mountain Project was first evaluated by ESI as part of a nationwide alunite exploration program in 1969 which included literature searches, aerial reconnaissance for the bleached alunite zones, and field studies. In 1970 ESI started the first systematic exploration of the Blawn Mountain deposit. ESI completed a total of 320 drill holes throughout the property.

Area 1 had been the most extensively delineated area by advancement of 230 drill holes. Approximately 33 drill holes terminated in the alunite deposit so mineralization may continue vertically downward in places. Areas 2, 3, and 4 were not fully delineated horizontally or vertically; 12 drill holes were advanced in Area 3 (one of which stopped in the mineral deposit), 17 drill holes were advanced in Area 2 (four of which stopped in the mineral deposit) and three drill holes were advanced in Area 4 (one of which stopped in the mineral deposit). Previous drill samples no longer exist so additional study of these samples is not possible.

After acquiring the property in 2011, PRC initiated a validation drilling program on Area 1 primarily to validate the previous exploration efforts. Under the guidance of North American Exploration Company (NAE), a combination of 19 core holes and 16 reverse circulation holes were completed on Area 1 between October 2011 and February 2012. All 35 drill holes were twinned to locations of previous drill holes completed by ESI. PRC's validation drilling program was followed by an infill drilling program during the summer of 2012 in Areas 1 and 2 combined with exploration/reconnaissance drilling in other areas. The drilling program included 17 additional holes on Area 1, 50 holes on Area 2, 2 holes on Area 4, and 21 holes on the ridgeline extending southwest of Area 1 referred to as the Southwest Extension. A third reverse circulation drilling program was conducted in January and February of 2013. The program included two holes on Area 1 and 17 holes on Area 2 for a combined total of 8,310ft. The primary purpose of the drilling was to further increase geologic assurance for resource assessment.

A total of 90 drill holes were completed including 74 reverse circulation holes, 8 HQ (2.5 inch (in.) diameter) core holes, and 8 PQ (3.4in. diameter) core holes. PQ core holes were completed to collect material for metallurgical testing. A total of 32,392ft were completed in the reverse circulation and core drilling program. In addition to the resource drilling, PRC completed 10 groundwater monitor wells in valley fill material to begin baseline characterization of near-surface hydrology. The 10 monitor wells represent a total drilling footage of 2,400ft. Norwest provided an onsite quality assurance/quality control (QA/QC) manager who oversaw all procedures being employed in data collection and sampling. The QA/QC manager was responsible to ensure that geology logs, geophysics, sampling, and surveying were meeting established protocols and procedures and that a proper chain of custody was followed for the disposition of all samples. Figures 9.2 and 9.3 show the drilling completed by PRC in Area 1 and Area 2, respectively.

1.5 MINERAL PROCESSING AND METALLURGICAL TESTING

Metallugical testing of the Blawn Mountain deposit has been performed during three major programs. The earliest was performed under the direction of Earth Sciences Inc, during the 1970's. The next two programs were performed under the direction of Potash Ridge Corporation during the period 2011 to 2013. The majority of testing has been performed by Hazen Research Inc. in Golden, Colorado.

In 1970, ESI contracted Hazen Research Inc. (HRI) in Golden, CO to develop and perform an extensive metallurgical testing program on alunite composite samples from the NG Alunite project.

Beginning in 2011, PRC engaged HRI to perform a series of bench-scale experiments on the alunite samples taken from the test pit developed by ESI in the 1970's, to recover SOP and possibly Al_2O_3 from alunite. The objective of the program was to confirm the results of the testwork performed in the 1970's and provided the basis for the Preliminary Economic Assessment issued by PRC in November 2012.

The 2013 program was a continuation of the work and, with HRI as the main laboratory, PRC also enlisted the services of several laboratories, turnkey proprietary technology suppliers, and equipment vendors to assist in process optimization through additional testing of ROM ore samples, pilot plant test residues, and desktop process simulation and modeling studies. The laboratories included:

- Hazen Research, Inc. (HRI) Golden, CO
- ALS Metallurgy Kamloops, BC, Canada
- JK Tech, Brisbane, Queensland, Australia
- Phillips Enterprises LLC, Golden, CO.

Metallurgical tests performed to date include size reduction of ROM ore, concentration of alunite by flotation, drying, calcination and roasting of whole ore and concentrate including an assessment of the composition of roaster off-gases as feed to the sulfuric acid plant, calcine leaching for extracting soluble SOP, solid/liquid separation of leached slurry to recover the brine (filtrate), evaporation and crystallization of brine to recover the product, and management of alumina-rich process residues.

Results of metallurgical experiments from the 2013 test program at HRI (T. J. Salisbury, June 24, 2013) including testwork subsequent to the June report, from the 2011 and 2012 HRI test program (R. J. Mellon, May 21, 2012) as well as those from 1972 for Earth Sciences Inc., ESI, also by HRI (F. J. Bowen, et al. April 12, 1973) are summarized in the following sections.

Metallurgical tests were performed at HRI on four distinct bulk and PQ core composite samples, identified as Master Composites MC-A, MC-B, MC-C, and Bulk 2 Composite, prepared from 200 kg of samples received in May 2013.

Table 1.1 presents the results of analyses of the master composite head samples.

TABLE 1.1 ICP-OES CHEMICAL HEAD ASSAYS OF MAJOR CONSTITUENT ELEMENTS*

Sample ID	Analysis, %					
	K	S	Al	Na	Fe	Ti
Bulk 2	4.47	NA	10.2	0.190	0.820	0.284
MC-A	2.74	5.47	6.82	0.388	1.88	0.306
MC-B	3.31	5.47	7.50	0.221	1.13	0.245
MC-C	2.48	4.12	7.25	0.343	1.57	0.201

***** Adopted from T. J. Salisbury, June 24, 2013.

Determination of mineral composition and fragmentation characteristics using Particle Mineral Analysis (PMA) via Quantitative Evaluation of Materials by Scanning Microscopy (QEMSCAN™):

- Alunite and quartz are the dominant minerals in the samples and for >75 µm the assays ranged, respectively, between 26 and 41% and 53 and 62%. As the alunite concentration increases the quartz content decreases.

During 2013, HRI completed a comprehensive comminution testing program utilizing the PQ core composite samples, which included the following tests:

Table 1.2 provides a summary of test results of abrasion index and work indices.

TABLE 1.2 SUMMARY OF AI, BWI, RWI, AND CWI RESULTS*

PRC ID	Ai, g	BWi, kWh/t	RWi, kWh/t	CWi, kWh/t
MC-A	0.4057	12.0	10.6	10.3
MC-B	0.4132	14.7	13.3	10.0
MC-C	0.4838	14.5	10.9	10.2

*T. J. Salisbury, June 24, 2013.

Table 1.3 is a summary of the Semi-autogenous grinding Mill Comminution test results.

TABLE 1.3 SUMMARY OF SMC TEST RESULTS*

Parameter	MC-A	MC-B	MC-C
Specific gravity	2.60	2.49	2.32
A - Maximum Breakage	79.8	70.9	75.4
b – Relation between energy and impact breakage	1.07	1.25	1.64
A x b = Overall AG-SAG hardness	85.4	88.6	123.7
DW_i = drop-weight index, kWh/m^3	3.04	2.8	1.88
DW_i %	17	15	8
M_{ia} = Coarse particle component, kWh/t	10.7	10.5	8.2
M_{ih} = High-pressure grinding roll (HPGR) component, kWh/t	6.8	6.5	4.7
M_{ic} = Crusher component, kWh/t	3.5	3.4	2.4
T_a = Low-energy abrasion component of breakage	0.85	0.92	1.38

*T. J. Salisbury, June 24, 2013; AG = Autogenous Grinding

Rougher Flotation tests were performed at primary grind size of 80% passing 80 µm at pH 10 with potassium hydroxide (KOH) for pH control, oleic acid and diesel fuel oil as collectors, and methyl isobutyl carbinol (MIBC) frother:

- Approximately 99% of the alunite in the feed (ROM ore) was recovered, averaging 77% of the mass of the feed from the MC-A and MC-B samples
- Approximately 90% of the alunite in the feed was recovered, which averaged 51% of the mass of the feed from the MC-C sample.

Cleaner Flotation tests performed at pH 10 with KOH for pH control, oleic acid and diesel fuel oil as collectors, methyl isobutyl carbinol (MIBC) frother and several depressants such as sodium silicate (Na_2SiO_4), starch-based PE_2S as well as Cyquest 40E and Cyquest 32243 for gangue minerals.

Drying and Roasting Experiments:

- During 2013, a parallel path of process development consisted of experiments evaluating drying and calcination of both whole ore and alunite flotation concentrate.
- Moisture content of the filter cake is an important factor because drying and roasting are energy-intensive and the filter cake is the feed to the thermal processing units. Additionally, the amount of filtrate recovered has a significant role in water conservation in the operations. Therefore, PRC commissioned Pocock Industrial to perform sedimentation and filtration tests on the water-leach slurry
- In the 3rd quarter of 2013, PRC contracted FLSmidth Pyrometallurgy group in Bethlehem, PA to evaluate the most energy-efficient method of drying, calcining and roasting either alunite flotation concentrate or whole ore Filter Cake at an estimate 10 wt.% moisture and P80 of 1000 µm (1.2mm maximum)
- FLSmidth has performed desktop simulation studies on Flash Dryer/Calciner/Roaster circuit. Preliminary evaluations estimate that four thermal processing units, each rated at 330tph with a residence time of 2 seconds are required
- FLSmidth also reviewed the effect of the size of particles in the feed and the temperature regime of treatment on selection of the Rotary Kiln, or the Fluidized Bed Reactor, or the Gas Suspension Calciner as the thermal processing unit
- FLSmidth estimates from simulation studies the concentration of SO_2 and SO_3 at 9.44 vol. % in the off gases as feed to the acid plant.

Leaching Calcine with Water and Effects of pH:

- During 2013, HRI completed a total of 38 calcine leaching experiments to determine the staged roasting concept, establish operating conditions for the leaching stage, and produce brine or leach liquors for the crystallization experiments

- During 2013, a series of water leach tests were completed on calcines generated using reductants such as sulfur, carbon monoxide/carbon dioxide, hydrogen sulfide, and natural gas
- The results of these tests show that sulfur and natural gas give the best possible overall calcine-leach results
- It is estimated that better than 90% of the potassium can be extracted by leaching the calcine with water
- A series of calcine leach tests were completed using Master Composite B to evaluate the effect of particle size of the feed on calcine-leach performance. The tests used natural gas as the reductant. The feed size varied between 100 and 1500 µm in five batch kiln tests, followed by leaching the calcine with water.
- Preliminary test results indicated that potassium can be leached from calcine feed of 1.5mm with only a minor reduction in extraction. 1000 µm was chosen as the design particle size.

During 2012, HRI leached of calcines with water at 90°C. To evaluate the effect of leaching temperature on extraction, a sample of calcine generated at 800°C and 30 minutes residence time was leached at room temperature (25°C):

- Leaching at 25°C produced a potassium extraction of only 61% and 4% for aluminum.
- Comparable leaches conducted at 90°C resulted in 83% extraction for potassium and 9% for aluminum.

Leaching Alumina: During 2013, HRI conducted medium and high temperature sodium hydroxide leach tests, using autoclaves, to determine whether the alumina contained in the SOP leach residue, which was produced by low temperature calcining, remained in a form that is soluble in hot sodium hydroxide solutions similar to those used in the Bayer alumina refining process. The tests were successful and confirmed that, with low temperature calcining, the alumina remained in a soluble form.

Solid Liquid Separation Testing:

- Solid liquid separation tests were performed by Pocock Industrial on samples of roaster feed and leach residue material for each of the master composites.
- The results of tests performed in October 2013, by Pocock Industrial on Composite A-, Composite B- and Composite C Calcine Leach Slurry samples, respectively, are summarized below:
    - **Physical Properties:** The respective values for Composite A-, Composite B- and Composite C Calcine Leach Slurry samples are: ***Liquid specific gravity:*** 1.04, 1.05 and 1.05; ***Solids specific gravity:*** 2.76, 2.78, and 2.90; and ***pH:*** 5.5, 5.6, and 5.2
    - **Flocculant Screening:** Hychem AF 304, a medium-to-high molecular weight, 15% charge density, anionic polyacrylamide is the recommended flocculant. For Composite

A-, Composite B- and Composite C Slurry samples at 20^{o}C and initial pulp density of 30%, 20% and 30% solids, respectively, the maximum effective dosage at flocculant concentration of 0.1g/L was in the range 20 – 30g/Mt, 25 – 35g/Mt and 30 – 40g/Mt

- **Conventional (Static) Thickening:** At recommended flocculant dosages for Composite A-, Composite B- and Composite C Calcine Leach Slurry samples, the respective maximum unit area for Conventional (Static) Thickener sizing is 0.125m^2/Mtpd, 0.160 – 0.205m^2/Mtpd and 0.125m^2/Mtpd.
- The maximum solids content of the feed to the thickeners is 25 – 35wt.%, 30 – 40 wt.%, and 20 – 30 wt.% and the estimated underflow pulp density is 69 – 73wt.%, 61 – 65wt.% and 69 – 73 wt.%, respectively, for the three slurry samples
- **Pulp Rheology:** Pulp viscosity data were collected using the Fann (Model 35A) Viscometer, after destroying the long-chain molecular structure of the flocculant used. The data classify the thickener underflow pulps as pseudoplastic class of "non-Newtonian fluids," or in which the flow behavior is dependent on the shear rate and changes with applied stress
- **Vacuum Filtration:** The design criteria for selection of the Horizontal Belt Vacuum Filter, based on tests performed on Composite A-, Composite B- and Composite C Calcine Leach Slurry samples, are as follows:
  - Filter feed solids = 71.3%; pH = 7.2; vacuum level = 67.7kPa; moisture content of filter cake ranged from 17.8wt.% to 22.5wt.% for cake thickness in the 10mm to 15mm range
  - For Composite A-, Composite B- and Composite D samples the production rate ranged, when flocculant was ***not*** used, respectively, from 487 to 594kg/m^2**.**hr, 163 to 225kg/m^{2}hr, and 166 to 231 kg/m^{2}hr and at flocculant dosage of 70g/Mt ranged, respectively, from 1089 to 1363kg/m^{2}hr, 1185 to 1564kg/m^{2}hr, and 1189 to 1608kg/m^{2}hr
- **Pressure Filtration:** The results of two sets of Automatic Pressure Filter tests performed on each of Composite A, Composite B and Composite D samples are as follows:
  - Feed solids ≈ 70 wt.%; pH = 7.2; Feed pressure = 551.6kPa; Cake thickness = 60mm; moisture content of Filter cake from **11.5 wt.% to 14.2 wt.%;** Total cycle time from 16 to 18 minutes; sizing basis in m^3 of pressure filter volume per metric tons of dry solids ranged from 10.74 to 16.19m^3/Cycle, including 1.25 scale-up factor.

Sulfuric Acid Plant: Based on information developed by FLSmidth by desktop simulations for the whole ore case, Du Pont-MECS estimates that two Sulfuric Acid plants each, with a throughput capacity of 2,000tpd of concentrated acid be constructed, each dedicated to process the off-gases from two (2) lines of Roasters and with provisions for treating off-gases bypassing any of the Roasters being serviced.

Recovery of elemental sulfur from the roaster off-gases was considered as an alternative to manufacture of sulfuric acid at the Project site. PRC contracted Fluor Corporation (Fluor) to evaluate the technical feasibility and economic viability of a Sulfur Recovery Unit (SRU):

- Fluor identified the following two technologies: the Fluor SO_x® Claus process for production of bright yellow sulfur and use of commercially-proven technology. which allows achievement of 99.9 plus percent sulfur recovery
- Manufacture of sulfuric acid has been adopted as the preferred option at the Project site.

Crystallization of SOP:

- During 2012, HRI performed one crystallization experiment with the goal to generate crystals in several stages so that the co-precipitation of impurities can be evaluated
- The crystallization experiment recovered 43% of the potassium in the feed solution by reducing the liquor volume 91% by evaporation and the crystals assayed 25 wt.% K with small amounts of other contaminants
- The Cycle 1 filtrate as feed to Cycle 2 experiments was evaporated from 327mL, or an additional 79%. After washing, 200mg of crystals were collected, which assayed 25 wt.%K, < 0.05 wt.%Al, and < 0.001 wt.%Ti, 0.021 wt.%Na, < 0.025 wt.%Fe, and < 0.0025 wt.%Mg
- In a commercial-scale operation, a limited amount of K_2SO_4 will be recovered until other salts begin to crystallize, contaminating the product SOP
- A bleed from the crystallizer will be necessary to prevent contamination of the K_2SO_4 product. Impurity build-up should be controlled in the recirculated centrate by establishing a purge or “bleed” stream from the Centrifuge dewatering circuit
- Bench-scale investigations will be required to determine the amount of bleed and to identify a method of treatment to recover the contained values. After treatment, the barren solution essentially containing sodium chloride can be routed to an evaporation pond.

1.6 MINERAL RESOURCES AND RESERVES

Four potential mine development targets have been identified within the Blawn Mountain Project area. Only Area 1 and Area 2 have sufficient geologic and analytical data to support resource estimation at this time. Areas 3 and 4 are defined by a limited number of historical holes and surface mapping, along with only two validation holes in Area 4. Both areas are recognized as future exploration targets.

Norwest has estimated resources from 3DGBM’s constructed in MineSight®, a software package developed by Mintec Inc. The estimate was prepared in compliance with NI 43-101 requirements for the definition of mineral resources. The 3DGBM’s are based on the assays and lithologies of the current drilling database and on a series of 30 interpreted geological cross-sections constructed through Area 1 and 29 cross-sections constructed through Area 2.

Resource classification is based on set distances from drill hole sample intervals in 3D space. These distances were based on semi-variogram analysis of K_2O sample data (Table 1.4).

TABLE 1.4 RESOURCE ASSURANCE CRITERIA FROM VARIOGRAPHY

Compound	Measured	Indicated	Inferred
K_2O	<150ft	<350ft	<2,000ft

Other estimation criteria include assumed density of alunite and waste established at 153.8 pounds per cubic foot (lb/ft^3) or specific gravity of 2.46 grams per cubic centimeter (g/cc).

Resource classification is based on the CIM Standards on Mineral Resources and Reserves, a set of definitions and guidelines established by the Canadian Institute of Mining and Metallurgy and Petroleum. Table 1.5 shows the estimated classified resource for Area 1 at increasing incremental K_2O cut-off grades and Table 1.6 shows the estimated classified resource for Area 2 at increasing incremental K_2O cut-off grades.

TABLE 1.5 CLASSIFIED RESOURCE ESTIMATE FOR THE BLAWN MOUNTAIN AREA 1 ALUNITE DEPOSIT

			IN SITU GRADES						CONTAINED RESOURCES				
RESOURCE CLASSIFICATION	CUTOFF GRADE K_2O (%)	IN SITU (TONS)	K_2O (%)	K_2SO_4 (%)	AL_2O_3 (%)	SO_4 (%)	Alunite based on K_2O (%)	Alunite based on Al_2O_3 (%)	K_2O (TONS)	K_2SO_4 (TONS)	Al_2O_3 (TONS)	Alunite based on K_2O (TONS)	Alunite based on Al_2O_3 (TONS)
MEASURED	0.00	72,400,282	3.42	6.32	15.68	2.09	30.04	42.45	2,473,700	4,574,597	11,353,088	21,746,273	30,734,994
	1.00	71,529,372	3.45	6.39	15.71	2.06	30.36	42.52	2,469,909	4,567,586	11,235,691	21,712,945	30,417,176
	2.00	64,979,040	3.64	6.73	16.19	2.09	32.01	43.83	2,366,212	4,375,819	10,520,042	20,801,341	28,479,776
	2.50	56,872,179	3.84	7.10	16.33	2.13	33.75	44.21	2,183,323	4,037,604	9,286,829	19,193,568	25,141,232
	3.00	48,362,178	4.03	7.44	16.62	2.19	35.39	45.01	1,946,916	3,600,418	8,040,067	17,115,319	21,766,008
	3.50	34,526,334	4.33	8.00	17.31	2.31	38.03	46.87	1,493,713	2,762,313	5,976,957	13,131,213	16,180,773
	4.00	19,624,648	4.78	8.84	18.71	2.55	42.01	50.64	937,725	1,734,127	3,671,046	8,243,526	9,938,226
INDICATED	0.00	93,823,555	3.26	6.03	15.19	2.03	28.65	41.12	3,057,710	5,654,601	14,250,203	26,880,292	38,578,041
	1.00	93,313,743	3.27	6.05	15.19	2.02	28.78	41.12	3,054,532	5,648,725	14,172,865	26,852,358	38,368,671
	2.00	83,493,484	3.48	6.43	15.88	2.05	30.57	42.98	2,902,985	5,368,470	13,255,676	25,520,109	35,885,665
	2.50	74,184,688	3.63	6.71	16.06	2.09	31.91	43.47	2,692,385	4,979,008	11,911,168	23,668,726	32,245,822
	3.00	57,939,557	3.87	7.15	16.40	2.14	34.01	44.40	2,241,624	4,145,419	9,503,304	19,706,088	25,727,272
	3.50	36,959,714	4.21	7.78	17.30	2.29	36.97	46.84	1,554,489	2,874,705	6,395,250	13,665,492	17,313,172
	4.00	17,565,100	4.73	8.75	19.11	2.57	41.61	51.73	831,391	1,537,486	3,356,462	7,308,752	9,086,589
MEASURED AND INDICATED	0.00	166,223,837	3.33	6.15	15.40	2.06	29.25	41.70	5,531,410	10,229,198	25,603,291	48,626,565	69,313,034
	1.00	164,843,115	3.35	6.20	15.41	2.04	29.46	41.73	5,524,441	10,216,310	25,408,555	48,565,303	68,785,847
	2.00	148,472,524	3.55	6.56	16.01	2.07	31.20	43.35	5,269,197	9,744,288	23,775,718	46,321,450	64,365,441
	2.50	131,056,867	3.72	6.88	16.17	2.11	32.71	43.79	4,875,708	9,016,612	21,197,996	42,862,294	57,387,054
	3.00	106,301,735	3.94	7.29	16.50	2.17	34.64	44.68	4,188,540	7,745,837	17,543,371	36,821,407	47,493,280
	3.50	71,486,048	4.26	7.89	17.31	2.30	37.49	46.85	3,048,201	5,637,017	12,372,207	26,796,705	33,493,946
	4.00	37,189,748	4.76	8.80	18.90	2.56	41.82	51.16	1,769,116	3,271,614	7,027,508	15,552,278	19,024,815
INFERRED	0.00	2,255,374	3.18	5.87	14.62	2.04	27.92	39.57	71,626	132,458	329,697	629,665	892,554
	1.00	2,255,374	3.18	5.87	14.62	2.04	27.92	39.57	71,626	132,458	329,697	629,665	892,554
	2.00	1,919,126	3.48	6.44	15.87	2.10	30.60	42.95	66,797	123,527	304,485	587,213	824,299
	2.50	1,793,895	3.56	6.59	15.98	2.11	31.33	43.26	63,938	118,240	286,657	562,078	776,036
	3.00	1,429,416	3.77	6.97	16.36	2.11	33.15	44.29	53,899	99,675	233,868	473,825	633,126
	3.50	665,917	4.37	8.08	18.68	2.44	38.41	50.56	29,097	53,809	124,375	255,794	336,706
	4.00	407,414	4.78	8.84	20.45	2.68	42.05	55.35	19,486	36,036	83,298	171,303	225,504

TABLE 1.6 CLASSIFIED RESOURCE ESTIMATE FOR THE BLAWN MOUNTAIN AREA 2 ALUNITE DEPOSIT

RESOURCE CLASSIFICATION	CUTOFF GRADE K_2O (%)	IN SITU (TONS)	IN SITU GRADES K_2O (%)	K_2SO_4 (%)	AL_2O_3 (%)	SO_4 (%)	Alunite based on K_2O	Alunite based on Al_2O_3	CONTAINED RESOURCES K_2O (TONS)	K_2SO_4 (TONS)	Al_2O_3 (TONS)	Alunite based on K_2O (TONS)	Alunite based on Al_2O_3 (TONS)
MEASURED	0.00	110,497,331	2.87	5.31	12.42	1.94	25.23	33.62	3,170,721	5,863,592	13,720,785	27,873,773	37,144,805
	1.00	104,377,825	3.02	5.58	13.05	2.04	26.54	35.33	3,150,958	5,827,044	13,622,767	27,700,035	36,879,452
	2.00	93,679,360	3.18	5.87	13.38	2.11	27.91	36.22	2,974,320	5,500,388	12,532,237	26,147,212	33,927,177
	2.50	79,064,980	3.34	6.17	13.60	2.20	29.33	36.81	2,638,240	4,878,879	10,750,703	23,192,741	29,104,220
	3.00	50,041,863	3.68	6.80	14.26	2.43	32.34	38.60	1,841,190	3,404,900	7,134,769	16,185,883	19,315,191
	3.50	28,969,753	3.98	7.36	14.75	2.62	34.97	39.93	1,152,272	2,130,888	4,272,459	10,129,610	11,566,369
	4.00	9,150,291	4.47	8.26	15.62	2.72	39.27	42.29	408,771	755,937	1,429,550	3,593,501	3,870,067
INDICATED	0.00	307,822,418	2.84	5.25	12.47	1.90	24.96	33.77	8,739,386	16,161,686	38,398,692	76,827,849	103,952,645
	1.00	293,961,004	2.96	5.47	12.97	1.97	25.98	35.12	8,688,605	16,067,777	38,135,561	76,381,434	103,240,300
	2.00	263,614,932	3.10	5.73	13.34	2.05	27.25	36.12	8,171,536	15,111,563	35,174,931	71,835,880	95,225,305
	2.50	212,810,329	3.30	6.10	13.64	2.15	29.00	36.93	7,020,826	12,983,563	29,031,159	61,720,000	78,592,933
	3.00	130,484,506	3.64	6.74	14.25	2.37	32.04	38.57	4,755,377	8,794,086	18,592,476	41,804,470	50,333,410
	3.50	71,126,489	3.96	7.32	14.70	2.57	34.81	39.79	2,816,324	5,208,209	10,454,527	24,758,277	28,302,416
	4.00	20,689,481	4.48	8.28	15.52	2.62	39.37	42.03	926,537	1,713,438	3,211,794	8,145,177	8,694,943
MEASURED AND INDICATED	0.00	418,319,749	2.85	5.27	12.46	1.91	25.03	33.73	11,910,107	22,025,277	52,119,477	104,701,621	141,097,450
	1.00	398,338,829	2.97	5.50	12.99	1.99	26.13	35.18	11,839,563	21,894,821	51,758,329	104,081,470	140,119,752
	2.00	357,294,292	3.12	5.77	13.35	2.07	27.42	36.15	11,145,855	20,611,952	47,707,169	97,983,092	129,152,483
	2.50	291,875,309	3.31	6.12	13.63	2.17	29.09	36.90	9,659,066	17,862,442	39,781,862	84,912,741	107,697,153
	3.00	180,526,369	3.65	6.76	14.25	2.39	32.12	38.58	6,596,568	12,198,986	25,727,245	57,990,353	69,648,601
	3.50	100,096,242	3.96	7.33	14.71	2.59	34.85	39.83	3,968,596	7,339,097	14,726,986	34,887,887	39,868,784
	4.00	29,839,772	4.47	8.28	15.55	2.65	39.34	42.11	1,335,308	2,469,376	4,641,344	11,738,678	12,565,010
INFERRED	0.00	150,481,703	2.64	4.88	12.08	1.67	23.18	32.70	3,967,300	7,336,699	18,175,180	34,876,487	49,203,709
	1.00	134,770,366	2.90	5.37	13.25	1.84	25.51	35.88	3,911,306	7,233,150	17,862,464	34,384,244	48,357,127
	2.00	124,717,186	2.99	5.54	13.46	1.88	26.31	36.44	3,733,035	6,903,475	16,787,557	32,817,068	45,447,146
	2.50	94,690,184	3.23	5.97	13.70	1.97	28.37	37.10	3,055,842	5,651,146	12,974,828	26,863,870	35,125,355
	3.00	55,899,862	3.56	6.59	14.25	2.15	31.31	38.58	1,990,874	3,681,709	7,967,240	17,501,748	21,568,850
	3.50	21,879,368	4.02	7.44	14.72	2.51	35.38	39.85	880,623	1,628,529	3,220,577	7,741,544	8,718,722
	4.00	9,143,043	4.45	8.23	15.46	2.63	39.13	41.86	406,939	752,549	1,413,798	3,577,392	3,827,423

The resources outlined in Table 1.5 and Table 1.6 reflects a material change in Area 1 and Area 2 from previous resource estimate reported in the 2012 PEA. The measured plus indicated resources at 1% K_2O cut-off grade have increased by 8.5Mt for Area 1 and decreased by 66.0Mt for Area 2 when compared to the historic 2012 PEA estimates. The inferred resources at 1% K_2O cut-off grade have increased by 1.9Mt for Area 1 and decreased by 116.0Mt for Area 2 when compared to the historic 2012 PEA estimates. The material change is attributed to the inclusion of additional infill drill hole data, the decision to use only PRC drilling data in the geologic models to maintain a common and verifiable assay reporting standard and improvements in separating alunite mineralization from surrounding country rock using sulfate grade data.

The preferred scenario for resource presentation is a 1% K_2O cut-off grade. At a 1% cut-off grade, the combined measured plus indicated resource for Area 1 is 164.8Mt at an average grade of 3.35% K_2O and 15.41% Al_2O_3. For Area 2, the combined measured plus indicated resource is 398.4Mt at a 1% cut-off grade with an average grade of 2.97% K_2O and 12.99% Al_2O_3. The calculated potassium sulfate grade (K_2SO_4) at a 1% K_2O cut-off grade for Area 1 is 6.20% and for Area 2 is 5.50%.

Approximately 43% of the identified resources for Area 1 are classified as measured, 56% as indicated resource and 1% as inferred resource. Approximately 20% of the identified resources for Area 2 are classified as measured, 55% as indicated resource and 25% as inferred resource.

Utilizing the geologic model and resource areas, various mine plans were developed to meet certain criteria related to project economics, grade, target production rates and processing methods. Ultimately, a base case mine plan was developed by applying various criteria in selecting the method and approach to mining, including:

- Annual production rate (ROM ore) to be constrained by processing capacities of 10.4 million tons.
- Ore cut-off grades of approximately 3.5% K_2O (Area 1) and 3.25% K_2O (Area 2) were utilized during the mining phase (Years 2017 through 2041) of the project, and a declining grade ranging from roughly 3.5% K_2O to 2.5% K_2O during the stockpile reclaiming phase (Years 2041 through 2057) of the project.
- Maximize economic use of the resource.

Taking into consideration the above, mine plans were developed (detailed further in Section 16) that use standard surface mining 'truck-shovel' techniques to mine the deposit. Mine development was scheduled using MineSight® software to generate a LOM schedule of waste and ore volumes. The mine plan formed the basis of workforce demands and schedules leading ultimately to estimates of capital and operating costs. Taking into account commodity pricing and market conditions, a cash flow of revenues and direct and indirect costs for both the mine and the processing plant were developed, which ultimately led to an estimate of project economics and value. The mine plan, at a prefeasibility level of assurance, was found to be of positive economic value and forms the basis of mineral reserves reported in Table 1.7.

TABLE 1.7 MINERAL RESERVES, BY CATEGORY

	Reserve Category		
	Proven ('000 tons)	**Probable ('000 tons)**	**Total**
Alunite Ore (ROM tons)	**136,254**	**289,540**	**425,794**
Ore (average K_2O (%) grade)	3.56	3.49	3.51
Ore (average K_2SO_4 (%) grade)	6.59	6.46	6.49
SOP (tons)	8,457	17,970	26,427
Sulfuric Acid (tons) @ 98% Purity	18,888	40,136	59,024

This estimate of resources and reserves was generated using the best information available concerning issues related to environmental, permitting, legal, title, taxation, socio-economics, marketing and political factors that could have a material influence on Norwest's findings. Norwest is not aware of any additional factors which may affect our reserve estimate.

The accuracy of resource and reserve estimates is, in part, a function of the quality and quantity of available data and of engineering and geological interpretation and judgement. Given the data available at the time this report was prepared, the estimates presented herein are considered reasonable. However, they should be accepted with the understanding that additional data and analysis available subsequent to the date of the estimates may necessitate revision. These revisions may be material. There is no guarantee that all or any part of the estimated resources or reserves will be recoverable.

1.7 MINING METHODS

Mining operations at Blawn Mountain Project will utilize conventional truck/shovel techniques to remove ore and waste material from Areas 1 and 2. During the course of developing the PFS, Norwest examined nine separate mining cases (Cases A – I). As the study progressed, various economic trade-off studies were performed between the mining cases. The trade-off studies examined different processing methods (flotation versus whole-ore calcining, roasting and leaching) as well as the impact of utilizing various ore grade cut-offs. Ultimately, Case I exhibited the best economic results and was chosen as the base mining case for the PFS. Case I uses approximate ore grade cut-offs of 3.5% K_2O in Area 1 and 3.25% K_2O in Area 2 and utilizes whole ore processing methods to produce SOP. A significant portion of the low grade ore encountered during mining will be segregated and stockpiled for later use. The stockpiled ore will be reclaimed and processed in the later years of the project. The mining schedule and sequence involve active mining operations occurring for approximately 24 years and stockpile reclamation for about 16 years, which gives the project a 40 year life. The economic results of this mine plan, as well as a more detailed description of all mining cases are presented in this report.

Before mining operation commence, salvageable growth-media material will be removed and placed in temporary storage areas. Mining operations begin in Area 1 and once the targeted ore has been removed from this area, mining will transition to Area 2. Ore and waste material will be removed using area and bench mining techniques. A relatively small equipment fleet will be needed for mining operations, employing a medium-size hydraulic excavator as well as a mid-size front-end-loader to load end-dump mining trucks. A typical fleet of support equipment (dozers, graders, etc.) will complement the operations. Mine pre-development activities will involve stockpile area and haul road and access road construction. These activities are anticipated to begin in 2016. Two years of mine production ramp-up (2017 and 2018) will occur with full mining production reached in 2019. The mining schedule for the base mining case (Case I) is presented as Table 1.8.

TABLE 1.8 CASE I MINING SCHEDULE YEAR

	2016	2017	2018	2019	2020	2021-2025	2026-2030	2031-2035	2036-2040	2041-2050	2051-2057
Topsoil (Myd^3)	0.2	0.4	0.4	0.4	0.3	1.1	0.9	0.7	0.5	1.2	0.6
Waste (Myd^3)	2.6	0.6	0.9	1.8	5.1	21.7	15.3	21.9	16.5	2.0	0.0
Ore (Mt)	0.0	3.5	7.1	10.6	10.6	53.2	53.2	53.2	53.2	9.2	0.0
Al_2O_3 (%)	N/A	15.2	15.9	16.1	16.3	17.4	14.2	14.3	14.6	15.2	N/A
K_2O (%)	N/A	3.7	3.8	4.0	4.2	4.3	3.7	3.6	3.8	4.11	N/A
NA_2O (%)	N/A	0.3	0.4	0.4	0.3	0.4	0.3	0.2	0.2	0.2	N/A
*LGO (Mt) Stockpiling	0.0	3.6	5.2	9.3	10.8	26.7	40.7	46.0	28.4	1.0	0.0
*LGO (Mt) Reclaiming	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	97.2	74.5
Al_2O_3 (%)	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A	13.9	13.0
K_2O (%)	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A	3.2	2.7
NA_2O (%)	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A	0.3	0.3
SOP (Mt)	0.00	0.23	0.48	0.75	0.79	4.1	3.5	3.4	3.6	6.0	3.6

*Low Grade Ore

The production schedule and mining sequence utilized to develop an equipment fleet that would adequately meet the needs of the mining operation. Two spreads of mining equipment are envisioned to remove ore and waste material from the mine. Table 1.9 presents the type, size and quantity of major mining equipment anticipated to be used. The quantity of equipment presented in these tables is at maximum levels. The primary equipment will remove both ore and waste material from the mine. This equipment was selected as it provides flexibility to the support the various tasks encountered during mining operations. The economics supporting the PFS assumes contract mining with the contractor providing the equipment.

TABLE 1.9 CASE I MAJOR MINING EQUIPMENT

Primary Equipment		
Excavator	$22yd^3$	1
FEL	$16yd^3$	1
End-Dump Truck	148t	19
Support Equipment		
Water Truck	16,000gal	2
Grader	297Hp	2
Dozer	580Hp	5
Drill	50,000lb	1

The major mining equipment will be supported by a fleet of smaller support equipment including pumps, light plants, lube and fuel trucks, mechanics trucks, pick-up trucks, etc.

Production requirements necessitate a 7 day per week schedule operating approximately 356 days per year. The remaining idle days account for holiday and other events such as training and inclement weather. The average workforce requirements for the mining operation throughout the project life are detailed below in Table 1.10.

TABLE 1.10 CASE I AVERAGE MINING WORKFORCE REQUIREMENTS

Category	2016-2020	2021-2025	2026-2030	2031-2040	2040-2057
Hourly Workers					
Contract Mining Labor	73	95	115	118	44
Management					
Mine Management	22	28	28	28	19
Total Employees	**95**	**123**	**128**	**146**	**63**

An initial evaluation of slope stability for the proposed surface mine was performed earlier this year. This analysis recommended mine slopes with overall angles of 45°. External waste piles are anticipated for this project, but the majority of the waste material will be disposed of in-pit. Additional geotechnical testing will be conducted as part of ongoing study efforts to provide additional data for mine facility design. Preliminary water management plans have been developed for surface water management, groundwater handling, and dust control for the pits and haul roads. Details of the surface water management plans, as well as groundwater management are presented in further detail in Section 16 of this report.

1.8 RECOVERY METHODS

The ROM ore will be processed, as envisioned, by crushing, wet grinding, reduction roasting, extracting SOP by leaching the calcine with water, solid/liquid separation, evaporation of brine, crystallization, and drying and packaging of SOP product for markets. Provisions have been made in the process plant to conserve energy and water through treatment and reuse of effluents and disposal of residues in an environmentally sound manner.

The proposed combination of unit operations in processing alunite ore is based on test results of investigations completed in 2012 and 2013 in support of process optimization at HRI, sedimentation and filtration studies at Pocock Industrial, Inc., in Salt Lake City, UT, and thermal processing systems modeling at FLSmidth in Bethlehem, PA.

Process design criteria for the major unit operations in the proposed integrated process plant complex are summarized below:

- ROM ore production rate is 10.4×10^6 short tpy at 2% moisture.
- Plant operation schedule is 330 days per year (dpy), 24 hours per day (hr/d).
- The nominal throughput capacity of the process plant is 1,313 short tons per hour (tph) and 1,500tph (maximum).
- Particle size of wet Grinding Circuit Cyclone overflow at P_{80} = 1000 µm to Filters.
- Moisture content of Filter Cake as feed to Dryer/Calciner/Roaster at 10%.
- Roasting temperature at 1022°F (550°C) and not to exceed 1112°F (600°C).
- Roaster off-gases are routed as feed to a 4,000tpd Sulfuric Acid Plant.
- Water Leaching of Calcine: 35% solids; 176°F; 60 minutes residence time; and 90% SOP extraction.
- Alumina/silicate leach residues pumped at 55% solids to the tailings facility.

Manpower requirements to operate the above described facility are summarized below in Table 1.11.

TABLE 1.11 AVERAGE WORKFORCE REQUIREMENTS FOR THE PROCESSING PLANT

Category	2016-2020	2021-2025	2026-2030	2031-2040	2040-2057
Hourly Workers					
Plant Labor	261	261	261	261	261
Management					
Plant Management	101	101	101	101	101
Total Employees	**362**	**362**	**362**	**362**	**362**

Primary Crushing

The ROM ore at minus 36in. delivered to the Gyratory Crusher Truck Dump Pocket by 150t capacity ore haul trucks will be crushed to minus 6in. The Gyratory Crusher is sized to operate 7,920 hours per year. The crushed ore will be discharged into a Crusher Ore Pocket and reclaimed by means Apron Feeders.

A Metal Detector installed on the Feeder Discharge Conveyor enroute to the coarse ore stockpile will shut down the conveyor and the systems upstream including the Gyratory Crusher if a significantly sized metal object is detected. A Belt Magnet suspended above the Feeder Discharge Conveyor head pulley will remove and dump magnetic metal scrap into a Trash Bin.

Ore Stockpile and Reclaim

The Feeder Discharge Conveyor delivers the crushed ore to the Coarse Ore Stockpile (COS) Stacker Conveyor, which in turn discharges the ore onto the Coarse Ore Stockpile. A Belt Scale located on the Stacker Conveyor provides tonnage information. A dust collection system will be designed to both minimize dust emissions and to recover fine product. A 50t capacity overhead crane is part of the design for maintenance of the Gyratory Crusher and auxiliary equipment at the primary crusher building.

The conical COS is approximately 292ft in diameter by 110ft high and covered with a geodesic dome and contains up to approximately 150,000t of ore for a four-day total ore supply to the grinding circuit. The reclaim system consists of four 7ft wide x 13ft long Reclaim Chutes. Four tractor-type Apron Feeders will discharge the ore to the Reclaim Conveyor delivering the ore to the Sag Mill Feed Conveyor. If an apron feeder shuts down for any reason, the standby Apron Feeder will start up to maintain the production tonnage to the grinding circuit.

Wet Grinding and Classification

Preliminary grinding tests indicate that a SAG Mill will be the only stage of grinding. The SAG Mill Feed consists of minus 6in ore from the stockpile delivered at a nominal rate of 1,313 dry short tons per hour (dstph), sufficient water is added to provide 65 wt.% solids density for the grind cycle and the SAG grinding ball charge of between 10% and 12% by volume in the SAG Mill. The undersize material (-0.5in.) from the SAG Mill Screen flows to the Grinding Mill Sump while the Screen oversize material will discharge to the Pebble Screen Discharge Conveyor. Approximately 10% of the SAG Mill product or up to approximately 132dstph will report to the pebble mill circuit.

The SAG Mill discharges to the Grinding Mill Sump and water is added to maintain the slurry pulp density at 56% solids. The re-pulped slurry is classified in a Grinding Cyclone Cluster. Approximately 75% of the total slurry flow to the Grinding Cyclone Cluster reports to the underflow. The cyclone underflow slurry has a density of 70% solids and flows to the SAG mill feed chute. The recirculating flow through the SAG milling circuit is estimated to be approximately 300%. The overflow from the Cyclone Cluster containing approximately 80% passing minus 1000 micron (1.0mm) particles and at approximately 35% solids will flow by gravity to the Grinding Circuit Thickener. The overflow from the Thickener is recycled to the Overflow Water Tank.

Solid/Liquid Separation

The underflow from the Grinding Circuit Thickener at an estimated 45 wt.% by solids and is dewatered in a battery of Horizontal Belt Vacuum Filters to produce Filter Cake as feed to the Dryer/Calciner/Roaster units. It is estimated that the moisture content of the Filter cake is approximately 10wt.%.

Drying/Calcining/Roasting

Filter Cake containing alunite and inert solids at approximately 90% solids (10% moisture) is dried, dehydroxylated, and roasted to decompose the alunite. The calcine produced contains a mixture SOP (K_2SO_4) and alumina (Al_2O_3). The energy required for the Flash Dryer circuit is provided by the exhaust gases from the Flash Calciner circuit. A start-up air heater is used during the system start-up to bring the Flash Roaster up to the auto-ignition temperature of the fuel. A portion of the heat energy in the off-gases is used in steam generation.

The calcine produced from the roasting step is quenched with water and the slurry is pumped to the Water leach Circuit. Calcine particles entrained in the Roaster off-gases are separated in a cyclone followed by an electrostatic separator (ESP). The dust collected in the ESPs is also recycled to the Water Leach Circuit.

The gamma-alumina phase occurs in a porous cubic structure, which can be leached with sodium hydroxide (NaOH). It reverts at high temperatures to the recalcitrant alpha form with hexagonal close-packed structures. The temperature limits on thermal processing are, therefore, required to assure that the gamma-alumina crystals are the end product of alunite roasting operations. Alunite decomposition reactions are, therefore, carried out at 1022^{o}F (550^{o}C). The maximum temperature in the Roaster shall not to exceed 1112^{o}F (600^{o}C).

Sulfuric Acid Plant

Air-to-gas heat exchangers are used to remove heat in the Roaster off-gases to ensure that the gas temperature entering the ESP is in the range of 525-575^{o}F (275-300^{o}C). Reducing conditions are created by injecting excess fuel at the Roaster to convert most of the generated SO_3 from the decomposition of aluminum sulfate to usable SO_2 in the production of sulfuric acid. The conversion of SO_3 to SO_2 will be regulated based on the amount of excess fuel delivered to the Roaster. An estimated 3,390stpd of sulfuric acid (H_2SO_4) is manufactured at the Project site from sulfur dioxide (SO_2) produced by the decomposition of alunite during the thermal processing of ROM ore.

Sulfur dioxide (SO_2) bearing off-gases from the Roaster are sent to the gas cleaning section of the Sulfuric Acid Plant and for further processing to recover 98% sulfuric acid (H_2SO_4). The highly-efficient Acid Plant has provisions for meeting the applicable emission limits for sulfur oxides (SO_x) in the stack gases discharged to the atmosphere. FLSmidth generally estimates the concentration of SO_2 and SO_3 in the flue gases from the thermal processing units at approximately 9.5+ vol. % as feed to the Acid Plant. This value is an estimate based on similar industry experience. No pilot plant testing has been completed by FLSmidth to date.

Emissions to the atmosphere from the Sulfuric Acid Plant at the Project site will be in compliance with the applicable Air Quality Permit requirements.

Calcine Water Leach

Based on bench-scale test results at HRI, it is proposed that the calcine discharged from the Roaster will be leached with water at a pulp density of 35% solids. After dissolving approximately 90% of the K_2SO_4 in the calcine, the leach residue slurry at approximately 33% solids is filtered in a battery of Belt Filters and the Filtrate (brine) containing approximately 10.5 to 11.0% dissolved K_2SO_4 is pumped to the Evaporator/Crystallizer circuit. The Filter Cake at about 90% solids is washed with water and the washed Filter Cake consisting of inert solids, Al_2O_3 and undissolved K_2SO_4 is re-pulped and pumped at approximately 55% solids to the tailings facility.

Additional tests are being conducted at HRI to optimize the pulp density of the slurry, and related leach cycle parameters and to evaluate the quality of the brine, and at Pocock Industrial to assess the rheological properties of the slurry, sedimentation rates, and filter cake form time and solids content.

Solid/Liquid Separation

The leached slurry, at the end of the leach cycle is filtered by using vacuum belt filters and consists of inert solids originally present in the ROM ore, Al_2O_3 produced in the Roaster, an aqueous solution containing approximately 90% of the K_2SO_4 extracted during leaching, plus approximately 10% of unleached K_2SO_4 solids. Solid/Liquid separation is one of the most significant unit operations at the Project site because of its importance in energy conservation both in the Dryer/Calciner/Roaster and Evaporator/Crystallizer circuits, in water conservation through reuse of effluents, and its impact on the size and type of water and wastewater treatment facilities.

Evaporation/Crystallization

Once the dissolved K_2SO_4 content of the Filtrate from the Calcine Leach circuit has reached an estimated 10.5 to 11.0% range, through recirculation of the filtrate, approximately 35% of the Filtrate will be pumped as feed to the Evaporator/Crystallizer circuit. The remaining 65% of the Filtrate and cake wash water, as required, will be recycled to quench the calcine solids discharged from the Roaster and to maintain the solids content of the slurry in the Leach Tanks at 35%.

Recovery of SOP crystalline product from the Filtrate containing K_2SO_4 in solution consists of an integrated Evaporator/Crystallizer system followed by separation of the SOP crystals from slurry, drying, sizing, and packaging of the product for shipment. The approach for evaporating water from the feed solution uses a multi-effect crystallizer with a four-effect design. Vacuum pumps are employed in the downstream stages to lower the pressure, which lowers the boiling point of the water being vaporized.

The dewatering section of each fourth effect evaporator is a gravitational thickener (settling chamber). The thickened slurry discharged from the "thickener" is the feed to a centrifuge where the crystals are separated from the liquid in the slurry. The wet crystals are transported to the Fluid Bed Dryer/Cooler. The centrate from the centrifuge is collected in a tank and returned to the Crystallizers.

Compactors are used to compact the fine particles into flakes, which will then be crushed and screened to obtain the granular-grade product. Rejects are recycled back to the Compactors. The proposed Compactor is 1000mm x 1000mm (nominal) roll size unit fitted with twin-screw force feeders and equipped with one motor and planetary reducer driving each roll. The two large rolls compress the fines under pressure to the point they fuse into larger particles. The dried crystals are compacted and bagged for the markets.

Anti-caking additives are added to all products in the Conditioning Drum before transfer to a Polishing Screen. From the Polishing Screen, - 2mm powder is directed to the powder load out. Oversize, +4mm, material is sent to a Cage Pactor and then combined with -4mm +2mm product to be conveyed to the standard granular product load out.

Power, Gas and Steam Requirements

The alternative to on-site power generation for the facility is to build a 138kV power transmission line which ties into the existing electrical grid. The only option for routing an electrical power transmission line is above ground. The routing for an electrical power transmission line will be approximately 46 miles.

The processing facility requires approximately 500,000lbs/hr of steam for use in the process. Currently no steam will be generated in the Sulfuric Acid Plant. The steam will be provided using packaged boilers fired with natural gas.

A substantial supply of natural gas will be required for the facility. This gas will be used to power the boilers to produce the steam required for the processes. It will also be used for general heating needs throughout the facility. An emergency generating system is needed for the facility to mitigate the adverse effects of a sudden loss of power on the mineral processing system.

Water and Energy Conservation Measures

Measures being evaluated and to be validated in bench-scale tests for conserving water use and for reducing energy consumption in the operations are as follows:

- Achieving 90% solids Filter Cake with 10% moisture, on a sustained basis, as feed to the Dryer/Calciner/Roaster system for savings in natural gas consumption and for increasing the volume of water reclaimed for reuse.

- Increasing the pulp density of slurry from 35% to 40 or 50% solids during leaching to conserve water, will reduce the size of Leach Tanks and downstream solid/liquid separation equipment.
- Based on desk-top simulation studies, recycling the filtrate to achieve an "equilibrium concentration" of approximately 10% K_2SO_4 and approximately 65% of the filtrate thereafter to the Leach Tanks, and pumping approximately 35% of the filtrate at approximately 10% SOP to the Evaporator/Crystallizer circuit.

1.9 Project Infrastructure

Project supporting infrastructure around the mine and processing plant site was developed for the following major components:

- Project Access
- Water Supply
- Water Treatment Plant
- Dust Control
- Power Supply
- Natural Gas Supply
- Mine Roads and Pads
- Mine Support Buildings
- Surface Water Management System
- Tailings Management
- Tailing/Seepage/Collection Pond
- Product Transportation
- Miscellaneous Support Infrastructure.

The Blawn Mountain Project will be accessed via upgrades to existing county roads Revenue Basin and Willow Springs off of State Highway 21. In addition to these road upgrades an existing county road will be relocated west of project area to provide a bypass for motorists and recreational users. The upgraded roads will convey traffic at 50mph with the exception of a 1.0 mile section that is designed for 30mph because of the terrain.

Water supply will be provided from the Wah Wah Valley. The site water supply system will include a well field to supply the required water, a series of booster pump station(s)/surge pond(s), and a pipeline to convey water from well field to mine site. A small portion of the water from the well field will be treated before continuing out to support process operation requirements (i.e. boiler, acid cooling, calcine cooling, crystallization,) and potable water.

The mine area is not expected to be a major dust producer. Within the process area, the tailings area and the transportation components of the operation can impact air quality. Dust will be managed by applying water as necessary.

New transmission and natural gas lines will be constructed to the site to provide power and gas to the project. Both of these lines will be owned and operated by the respective utility company.

Mine roads and pads and associated support buildings were designed to support a year-round mining operation. These facilities include:

- Access roads
- Mine truck shop
- Mine warehouse
- Administration building
- Reagent warehouse
- Fuel depot
- Explosives storage
- Equipment ready line
- Guard shack.

Storm water controls will be located downstream of all surface disturbances. These controls will consist of diversion ditches, sediment ponds, outlet control structures, and a combination tailings/runoff containment structure and a seepage/water collection pond.

As ore is processed, tailings are produced requiring storage. Tailings will be pumped from the processing plant to the tailings storage area. Based on the assumed gradation of this material, it is anticipated the tailings will be coarse grained sand that will be freely draining. A collection pond will be constructed to collect the drainage from the tailings and the runoff from the site and process facilities, mine haul roads and associated areas. Products, both SOP and sulfuric acid, will be transported via a short line rail system constructed onsite which will ultimately connect to the existing rail network. Storage and loadout facilities will be constructed near the processing plant which will connect to the rail system. Miscellaneous supporting infrastructure including a helipad, sanitary waste treatment, firefighting, man camp during construction and site transportation have also been considered for this PFS.

1.10 Markets and Contracts

The primary product, SOP produced from the Blawn Mountain Project will be marketed domestically and globally. The co-product sulfuric acid will be marketed to existing US phosphate producers, copper and gold miners, as well as mines under development in the region.

As the most commonly used alternative to Muriate of Potash (MOP) when the presence of chloride ions is undesirable, SOP sells at a premium over MOP. The SOP market in western United States is being served by a single producer leading to a supply constrained market. As a result, the high value crop growers in these markets pay a larger premium for SOP over MOP than premiums realized in other markets. For the period 2001 – 2010, SOP has commanded an average premium of 47% over MOP, ranging from 38% to 61%. In recent months, this premium has been as high as 98% in in the US.

Specialty crops best suited for SOP application account for approximately 40% of total crop revenues. SOP consumption in the US is approximately 350,000 tonnes per annum, with over 50% of this demand coming from California. California is the number one state in cash farm receipts, growing 58% of US-grown non-citrus fruits, nuts and vegetables and 100% of US almond production (the second highest commodity in value after milk). PRC believes the US market can absorb 485,000 additional tonnes of SOP per annum.

PRC intends to focus its marketing efforts on the economic value of SOP to growers of Premium Value Crops. California will be a key market given its large agricultural base of premium crops. Florida will be another key target. Currently, 100,000 tonnes per annum of SOP is imported into Florida from Europe and Chile, which can also be displaced given the transportation advantage over shipping from Europe. Outside of the US, China and Brazil with their growing populations and growing need for food are other key markets of focus for PRC.

The existing US Mountain West market for sulfuric acid in the Project region is 5.1 million tonnes per annum. In addition, there are new and planned mine developments and existing mine expansions having the potential to significantly increase this amount. These developments are combined with potential supply disruptions to existing sulfuric acid production in the region.

Product sales prices are discussed in greater detail below in Section 19. Table 1.12 summarizes average selling prices at the plant gate.

TABLE 1.12 PRICING SUMMARY USD

Pricing	Unit
Average SOP Selling Price – FOB Rail at Plant	\$649/ton
Average Sulfuric Acid Selling Price – FOB Rail at Plant	\$135/ton

1.11 ENVIRONMENTAL AND PERMITTING

Regulatory Environment

Mining and processing operations in the United States must comply with all applicable federal and state regulations. Mining operations in Utah require compliance with federal as well as state mining and environmental regulations. Utah has primacy over major environmental laws applicable to the project including mining, air and water permitting. The mine and processing plant is located on SITLA-controlled mineral and surface land, and is not expected to impact resources with federal oversight, and as such, federal site specific approvals and permits are not anticipated to be required for the mine and processing plant.

Environmental Setting

Environmental baseline conditions are being assessed for the following resources to assess permitting and regulatory requirements and to support permit applications:

- Air Quality
- Archeological Resources
- Wildlife Habitat including Threatened, Endangered, and Sensitive Species
- Vegetation including Threatened, Endangered, and Sensitive Species
- Soils
- Surface and Groundwater
- Wetlands and Waters of the US.

The results of these surveys and evaluations are discussed in detail in Section 20.

Major Operating Permit and Authorizations

The major permits and approvals that need to be obtained prior to the construction and start-up of the mine and processing plant are provided in Table 1.13.

TABLE 1.13 MAJOR REQUIRED PERMITS

Major Permits or Approvals	Issuing Agency
Exploration Permit	Utah Division of Oil, Gas and Mining
Large Mine Operation Approval	Utah Division of Oil, Gas and Mining
Water Appropriations	Utah Office of State Engineer
Groundwater Permits	Utah Division of Water Quality
Air Quality Permit	Utah Division of Air Quality
General Multi-Sector Industrial Storm Water Permit	Utah Division of Water Quality
Army Corps of Engineers Jurisdictional Waters Concurrence	US Army Corps of Engineers
County Conditional Use Permit and Other Permits	Beaver County

These permits are not meant to be all-inclusive and cover only the major permits required for the mine and processing plant. Permit details are discussed further in Section 20.

Social or Community Impact

A project of this scale represents a significant economic impact to Beaver County and the town of Milford and also to a lesser extent to adjacent Iron County. Representatives of Beaver County have expressed strong support for the project. Infrastructure and public services in Beaver County and to some degree Iron County will require upgrading and expansion to support the expanded population required for the project. The Utah "School and Institutional Trust Lands Management Act" requires SITLA to manage trust lands to optimize trust land revenues and increase the value of trust land holdings consistent with the balancing of short and long-term interests, so that long-term benefits are not lost in an effort to maximize short-term gains; and mandates the return of not less than fair market value for the use, sale, or exchange of school and institutional trust assets. The Blawn Mountain Project will assist SITLA in meeting these objectives.

1.12 CAPITAL AND OPERATING COSTS

Capital and operating costs were developed and analyzed for several of the mine plan cases discussed earlier. The PFS base case; which resulted in the most favorable economics, considered the following:

- Whole ore processing (calcining, roasting and leaching)
- 40 year mine life (approximately 500,000 to 860,000 tons of SOP annually)
- Mining ore grade cut-offs of roughly 3.5% K_2O for Area 1 and 3.25% K_2O for Area 2. Ultimately, all ore of approximately 2.5% K_2O or above is processed (later years stockpiling) resulting in an average of 3.5% K_2O over the project life
- Line power
- 1000 micron mill feed
- Wet grinding.

The total capital costs for the processing plant are summarized in Table 1.14.

TABLE 1.14 TOTAL PROJECT CAPITAL ESTIMATE (USD M'S)

	2016	2017	Total Construction and Development Capital	Sustaining Capital	Total Life of Project Capital
Project Infrastructure	$45	$45	$90	$3	$93
Processing Plant	$477	$477	$954	$153	$1,107
Product Storage and Handling	$15	$15	$30	$4	$34
Contingency	$25	$25	$50	$0	$50
Total	**$562**	**$562**	**$1,124**	**$160**	**$1,284**

The table and chart above do not include capital costs for the access road, the rail spur and loop, the gas pipeline, the acid plant, the water supply and treatment facility and the mining operations. These items are assumed to be either provided by a third party or financed through government programs. In either case, a provision has been made in the operating costs to account for these items. PRC has received indicative estimates from various parties with respect to the majority of these support assets. These capital costs will not be incurred by PRC. Table 1.15 below shows the capital cost of each item used to formulate the basis for the operating costs.

TABLE 1.15 THIRD PARTY PROJECT CAPITAL (USD M'S)

	Third Party Equipment and Infrastructure
Access Road	$53
Rail Spur and Loop	$76
Natural Gas Transmission Line	$83
Acid Plant	$280
Water Supply and Treatment System	$60
Initial Mine Capital	$89
Total	**$641**

Further discussion of these costs and the amounts included in operating costs are discussed in the Section 21 of this report.

Contingencies of 15% to 18% were added to the direct costs for various areas of the process plant depending on the design and basis for the cost estimates resulting in an average of 15.4%. It was not applied to turnkey quotes from vendors for the acid plant and calcining process as these included separate contingencies in the estimates.

Operating Costs

Average annual operating costs for the processing plant and mining operation are shown in Table 1.16. The total cash production cost summary is shown in Table 1.17. All costs in this report are stated in constant US 2013 dollars, there is no provision for inflation.

TABLE 1.16 AVERAGE ANNUAL PLANT AND MINE DIRECT OPERATING COSTS (USD$)

Direct Plant and Mine Cash Production Cost	Annual Average Cost($)/Ton SOP	Life of Plant Annual Average (000)
SOP Tons Sold		645
Sulfuric Acid Tons Sold		1,440
Mining Cost		
Mining (Contract Mine Operator Cost)	**$66**	**$42,381**
Processing		
Crushing & Grinding	$35	$22,322
Concentrate	$7	$4,317
Roasting	$199	$128,418
Acid Plant	$60	$38,474
Leaching & Crystallization	$14	$8,834
Drying and Compaction	$4	$2,553
Steam Plant	$23	$14,951
Tailings and Reclaim	$3	$2,108
Water and Tailings Thickeners	$0	$0
Product Storage and Loading	$4	$2,420
Total Processing	**$348**	**$224,396**
Credit for value of acid	**($302)**	**($194,348)**
Total Direct Operating Cost (Mining and Processing)	**$112***	**$72,430**

***ROUNDED**

TABLE 1.17 TOTAL CASH PRODUCTION COST SUMMARY

Total Cash Production Costs	Annual Average Cost($)/Ton SOP	Life of Plant Annual Average
SOP Tons Sold		645,000
Sulfuric Acid Tons Sold		1,440,000
Direct Plant and Mine Cash Production Cost	$414	$266,777
Credit for Value of Acid	($302)	($194,348)
Subtotal of Direct Plant and Mine Cash Production Cost	$112	$72,430
Royalties	$45	$28,704
Site G&A	$12	$7,932
Property Taxes	$11	$7,308
3rd Party Facility Charges	$34	$21,632
Corporate Overhead	$4	$2,500
Total Cash Production Cost	$218	$140,506*

*Rounded

1.13 ECONOMIC ANALYSIS

An economic analysis was completed for the Blawn Mountain Project. Production volume is planned at an average of 645,000t of SOP per year for the 40 year life of the project, ranging from 861,000t to 496,000t. As a result of the SOP production process, an average of 1.4Mt of sulfuric acid is also produced annually. Over the 40 year period, there are 26.4Mt of SOP and 59.0Mt of sulfuric acid produced. Pre-production cash outflows total $1.1 billion over the two year construction period. Cash flow is positive beginning in 2018. Payback occurs mid-way through 2022 which is approximately 7 years after the initial investment and 5 years after commissioning. Cash flow after payback averages $221M per year for a total net cash flow of $8.0 billion over the life of the project. Annual and cumulative cash flows are shown in Chart 22.2. The summary of cash flow for the project is presented in Table 1.18.

TABLE 1.18 PROJECT CASH FLOW SUMMARY

Project Cash Flow Summary	Life of Plant Annual Average $M
SOP Tons Sold	645
Sulfuric Acid Tons Sold	1440
Net SOP revenue FOB - Plant	**$419**
Net acid revenue FOB - Plant	**$194**
Total revenue FOB - Plant	**$613**
Direct Plant and Mine Cash Production Costs	$267
Royalties	$29
Site G&A	$8
Property Taxes	$7
Third Party Facility Charges	$22
Corporate Overhead	$3
Total Cash Production Costs	$336
Operating Margin	$277
Income Taxes	$53
Cash Flow from Operations	**$224**

The internal rate of return for the project is 20.5%. After tax net present values at 8%, 10%, and 12% are shown in Table 1.19.

TABLE 1.19 NET PRESENT VALUE RESULTS

Discount Rate	8%	10%	12%
After Tax Net Present Values	$1.5 billion	$1.0 billion	$0.7 billion

1.14 OTHER RELEVANT DATA AND INFORMATION

The Blawn Mountain Project schedule has been prepared in order to meet plant commissioning and production target dates in 2018. A high-level summary of the schedule is provided below:

- Exploration drilling of Areas 1 and 2 in support of the prefeasibility and feasibility studies was completed in August 2013.
- Project financing includes timing to cover five major categories for financing feasibility through production.
- Environmental permitting will span into second quarter 2016.
- Engineering studies and procurement started in January 2013 and will conclude in March 2017.
- Construction of third-party design build utilities will be in the final stages for the start of the plant commissioning in third quarter of 2017.
- Mining will start in December 2016. Ore will be stockpiled until the processing plant is online.
- Plant commissioning will occur from third quarter 2017 through second quarter of 2018 with full production occurring in fourth quarter 2018.

1.15 CONCLUSIONS AND RECOMMENDATIONS

Based on the results of the PFS, Norwest and ICPE have reached the following conclusions:

- There are sufficient mineable tons of ore at an average grade of 3.51% K_20 to produce an annual average of approximately 645,000 tons of SOP over a 40 year project life.
- No fatal flaws have been identified at this stage of project development.
- Pre-production capital costs estimated at $1.1 billion along with several third party build, own, operate arrangements will be required to bring this project into production.
- Cash costs of production per ton of SOP, after sulfuric acid credits and before royalties, is estimated at $173
- Based on the assumptions defined in this report, the project will generate positive cash flows and achieve an after tax IRR of 20.5%.

The overall conclusion is that the results of this study indicate positive economic results and the project should be continued to the next phase of development.

The recommendations for additional work have been discussed in detail in Section 26. A summary of the work and estimated costs are noted in Table 1.20 below.

TABLE 1.20 ESTIMATED COSTS OF ADDITIONAL WORK

Description	Range of Estimated Costs ($millions)
Mineral Processing and Metallurgical Testing	$1.5 to $3.0
Continued Evaluation of Recovery Methods	Included in Feasibility Study
Development Drilling	$0.75 to $1.0
Permitting	$2.0 to $3.0
Feasibility Study	$6.0 to $8.0
Total	**$10.5 to $15.0**

2 INTRODUCTION

Norwest Corporation (Norwest) has prepared this report on the Blawn Mountain Project. This report presents information pertaining to a Prefeasibility Study (PFS) performed at the request of Potash Ridge Corporation (PRC). The purpose of this report is to summarize the approach, findings and ultimately the evaluation of the potential economic viability of mining and processing ore from the areas controlled by PRC. PRC controls significant alunite deposits in the Blawn Mountain area of southwestern Utah. Mineral control is through a State of Utah Mining Exploration Agreement With Option To Lease (Exploration/Option Agreement), administered through the State of Utah School and Institutional Trust Lands Administration (SITLA) and two state mineral leases also managed by SITLA. The property is located approximately 30 miles southwest of the town of Milford, Utah and 30 miles east of the Nevada state border.

Several firms contributed to the creation of this document and additionally, several data sources have been utilized during the preparation of this report including:

- Results of exploration documented in various public reports and on recent drilling campaigns.
- Processing plant design and requirements performed by Intermountain Consumer Professional Engineers, Inc. (ICPE).
- Material balance information performed by ICPE and PRC.
- Marketing information and product sales prices provided by PRC and Serecon.
- Certain processing plant operating costs provided by ICPE and PRC such as plant manpower and some operating cost estimates on ancillary plant equipment and on transportation costs of the final products to the railcar.
- PRC's interpretation of the US Federal Tax regulations as they relate to the percentage depletion calculation used in the economic analysis, which was reviewed by independent tax specialist Wisan, Smith, Racker, & Prescott LLP of Salt Lake City, Utah (WSRP).
- Mine operating and portions of the plant operating costs were derived from the 2013 Western Mine and Mill Cost Estimating guide.
- Hatch Mott MacDonald prepared portions of Section 18 that relate to site access infrastructure.

The Blawn Mountain Alunite deposits were explored by Earth Sciences, Inc., (ESI), a mineral exploration and development company that was headquartered in Denver, Colorado, in the early 1970s and 1980s. ESI, which was a joint venture partner in The Alumet Company (Alumet), referred to the Blawn Mountain deposits as the NG Alunite deposits. The Blawn Mountain property subsequently came under PRC control in 2011 through the initial Exploration/Option Agreement with SITLA.

Alunite is a complex mineral containing alumina (Al_2O_3), potassium (K_2O), and sulfur (SO_3) all of which have important uses in commercial markets. PRC is pursuing development of the Blawn Mountain Project primarily for the manufacture of sulfate of potash (SOP). However, following initial development PRC anticipates multiple products from the alunite including sulfur products and potentially alumina.

Alunite is a naturally occurring mineral with the chemical composition of $KAl_3(SO_4)_2(OH)_6$. In pure state, alunite is comprised of 11.37% K_2O, 36.92% Al_2O_3, 38.66% SO_3, and 13.05% H_2O. At times sodium will replace a portion of the potassium altering the alunite to the mineral natroalunite. This is not common in the Blawn Mountain mineral deposit as drill hole cuttings typically assay at less than 1% Na_2O. Iron can replace some of the aluminum altering the alunite to the mineral jarosite. However, iron does not appear to occur at Blawn Mountain in significant quantities. Alunite occurs worldwide associated with hydrothermal alteration accompanying volcanic activity. Alunite can be present in some very large deposits (Hall, 1978) and the western United States contains some of the largest deposits known in the world. The Blawn Mountain deposit is one of these significant large deposits (Hall, 1978).

Alunite has been mined worldwide for centuries (Hall and Bauer, 1983). Mining of alunite in the United States has historically been for the production of potassium fertilizer. During War World I alunite was mined in the Mount Baldy mining district in Utah for production of potash fertilizer. The district was again mined during War World II for the alumina going to production of aluminum for the war effort. When potassium prices returned to normal levels following the two wars, alunite operations were no longer economically viable in the US primarily due to the size of operations. Alunite has long been known to have value for alumina, potassium, and sulfur, though three obstacles have often limited development:

- Adequate size of deposit
- Concentrations of commercial components
- Cost of building and operating a processing plant.

The size of most western alunite deposits were not known until the 1970s. Many of the western US alunite deposits are fairly large and make them a cost-effective source for potassium sulfate and a competitive alternative to bauxite for alumina.

Compared with other types of mineral deposits containing similar chemical compounds, alunite contains less potassium than sylvinite, approximately 5% versus 20 to 35% and contains less alumina than bauxite (about 18% compared to 45%). However, alunite can be mined in the US whereas all of the bauxite used in the US is imported. Sylvinite is more abundant and less expensive to process through solution mining and flotation. Key factors to the economics of processing alunite are that two valuable products are produced; potash and alumina. Also, production of potash from alunite is in the form of potassium sulfate which sells at a premium to the more commonly produced potassium chloride from sylvinite. Recent changes in world mineral economics (increased demand for minerals in Asia and higher mineral prices) have led to a re-examination of alternate sources of minerals like alunite. Recent increases in the commodity prices for potash and demand growth for bauxite have led to renewed interest in the Blawn Mountain deposit.

The Blawn Mountain Project contains the four mineralized areas previously defined by ESI and Alumet as the NG Alunite deposit. In June 2011, PRC acquired a collection of Alumet technical reports and correspondence from a third party that had received the data as part of another business transaction with ESI.

The authors of this PFS report have visited the project site on several occasions over the past two years. The authors certify that they have supervised the work as described in this report. The report is based on and limited by circumstances and conditions referred to throughout the report and on information at the time of this investigation. The authors have exercised reasonable skill, care and diligence to assess the information acquired during the preparation of this report.

The accuracy of resource and reserve estimates is, in part, a function of the quality and quantity of available data and of engineering and geological interpretation and judgment. Given the data available at the time this report was prepared, the estimates presented herein are considered reasonable. However, they should be accepted with the understanding that additional data and analysis available subsequent to the date of the estimates may necessitate revision. These revisions may be material. There is no guarantee that all or any part of the estimated resources or reserves will be recoverable.

3 RELIANCE ON OTHER EXPERTS

Norwest has prepared this report specifically for PRC. The findings and conclusions are based on information developed by Norwest and others available at the time of preparation and data supplied by outside sources.

PRC has supplied the appropriate documentation that supports the Exploration/Option Agreement and mineral leases it holds with the State of Utah (SITLA) to be in good standing. The existence of encumbrances to the agreement has not been investigated.

Norwest relied on ICPE to prepare Sections 13, 17 and the portion of Section 21 and 22 relating to the processing plant.

Norwest relied on Hatch Mott MacDonald to prepare portions of Section 18 that relate to site access infrastructure.

Norwest relied on transportation, marketing information and product sales prices provided by PRC. Norwest has not independently verified the market information and sales prices. Reliance on this information applies to Sections 19, 21, and 22.

Norwest relied on processing plant capital and operating costs provided by PRC and ICPE such as plant manpower and some operating cost estimates on ancillary plant equipment. Reliance on this information applies to Sections 21 and 22.

Norwest relied on PRC's interpretation of the US Federal Tax regulations as they relate to the percentage depletion calculation used in the economic analysis, which was reviewed by independent tax specialist Wisan, Smith, Racker, & Prescott LLP of Salt Lake City, Utah (WSRP). Our reliance on this information applies to Section 22.

Norwest relied upon ICPE and PRC to perform the material balance information. Reliance on this information applies to Section 17.

4 PROPERTY DESCRIPTION AND LOCATION

The Blawn Mountain Project is located in the southern Wah Wah Mountains of Beaver County, Utah about 180 air miles south-southwest of Salt Lake City, Utah (Figure 4.1). The property is situated west-southwest of Milford (30 air miles to the northeast) and west-northwest of Cedar City (55 air miles to the southeast). The property is located on the Wah Wah South 100,000-scale USGS topographic map and straddles four 24,000-scale maps: Lamerdorf Peak, Frisco SW, The Tetons, and Blue Mountain. The property occupies Township 29 South, Range 15 West, Sections 13-16, 21-29, 32-36 and Township 30 South, Range 15 West, Section 2 along the Blawn Wash and Willow Creek drainages that cover most of the historic NG Alunite property.

PRC controls the Blawn Mountain property through an Exploration/Option Agreement (ML 51983.0 OBA) and two mineral leases (ML52513, and ML 52364), administered through SITLA. The agreements consists of a main tract of land that covers 14,923.72ac and six individual 80ac tracts located 3.5 to 4.5 miles northeast of the main tract of land (Figure 4.2). Table 4.1 provides a legal description of the controlled area. The combined acreage for the three leases is 15,403.72ac. The Exploration/Option Agreement was issued to Utah Alunite, LLC, a 100% owned entity of PRC. In April 2012, Utah Alunite, LLC was merged into Utah Alunite Corporation (UAC), a 100% owned entity of PRC and the mineral lease was assigned by Utah Alunite, LLC to UAC.

UAC added five full sections adjacent to the north of ML 51983.0 OBA through the acquisition of SITLA lease ML 52513, effective June 1, 2013. The additional leased area was acquired to expand the amount of non-mineral-bearing property that would allow for siting of process tailings and potential alumina stockpiling. Additionally, the six individual 80ac parcels to the north of the mineral property were acquired from SITLA under lease ML 52364 on January 7, 2013 and are targeted for the development of water supply to the project.

There are two pre-existing mineral tracts consisting of a 40ac tract (ML48699.0 MC) along the western edge of the project area and a 155ac tract (ML 48698.0 MC) within the Blawn Mountain Project area. Another mineral tract of 640ac is located approximately one mile east of the Blawn Mountain Project. Remaining lands surrounding the Blawn Mountain Project are predominantly a mix of federal lands administered by the US Bureau of Land Management (BLM) and state lands administered by SITLA.

The Exploration/Option Agreement, ML 51983.0 OBA, is a combined metalliferrous minerals (includes sulfur) and potash exploration and an option to a mining lease agreement with the following stipulations:

- Three year lease
- Bonus payment of $200,000
- $6/ac each year ($69,300/yr).

The initial lease terms included a provision to provide a “Positive Prefeasibility Study” to SITLA documenting the project’s economic viability, possible markets, mining methods, and potential environmental issues and providing evidence showing the possible continuity and grade of the ore. In May 2013, SITLA provided a letter to PRC stating that the Preliminary Economic Assessment (PEA) completed in November 2012, satisfied the Positive Prefeasibility Study requirement and that PRC could proceed with exercising the option to convert the exploration agreement to a lease.

At the end of the agreement, March 31, 2014, an additional bonus payment of $1,020,000 is required for issuance of a combined metalliferrous minerals and potash lease. Primary term of the lease will be for 10 years with a provision to extend past the primary term provided the lessee is either in production of leased minerals or in diligent development of leased minerals. Annual rental rate for a combined mineral lease would be $1/ac as required by statute; in addition $4/ac advanced minimum royalty which would be increased at $1/ac commencing with the sixth lease year and each lease year thereafter. Combined lease will require a 4% gross royalty for metalliferrous minerals and a 5% gross royalty for potash and associated chlorides.

Mineral leases ML52513, and ML 52364, are standard metalliferrous mineral leases, each with a 10-year primary term and option to extend beyond the primary term. SITLA mineral leases carry an annual rental rate of $1/ac, a gross production royalty of 8% on fissionable minerals and 4% on non-fissionable minerals. Advance royalty payments equating to three times the annual rental rate were assessed at the time of issuance.

TABLE 4.1 LEGAL DESCRIPTION OF SITLA PROPERTY

T29S,R14W, SLB&M		**Acres**
Sec. 7	ALL	638.16
T28S,R14W, SLB&M		
Sec. 16	NW1/2NW1/4	80.00
Sec. 21	N1/2NE1/4	80.00
Sec. 22	N1/2NE1/4	80.00
Sec. 26	N1/2NE1/4	80.00
Sec. 27	N1/2NE1/4	80.00
Sec. 35	N1/2NE1/4	80.00
T29S,R15W, SLB&M		
Sec. 1	ALL	731.36
Sec. 2	ALL	725.00
Sec. 11	ALL	640.00
Sec. 12	ALL	640.00
Sec. 13:	ALL	640.00
Sec. 14:	ALL	640.00
Sec. 15:	ALL	640.00
Sec. 16:	E ½	320.00
Sec. 21:	ALL	640.00
Sec. 22:	ALL	640.00
Sec. 23:	ALL	640.00
Sec. 24:	ALL	640.00
Sec. 25:	ALL	640.00
Sec. 26:	ALL	640.00
Sec. 27:	ALL	640.00
Sec. 28:	ALL	640.00
Sec. 29:	ALL	640.00
Sec. 32:	ALL	640.00
Sec. 33:	ALL	640.00
Sec. 34:	ALL	640.00
Sec. 35:	ALL	640.00
Sec. 36	W½	320.00
T30S, R15W, SLB&M		
Sec. 2:	Lots 1(47.38), 2(47.32), 3(47.28), 4(47.22), S½SN½, S½(ALL)	669.20

There are four main zones of mineralization identified by PRC (Figure 4.3). Area 1 is located along a northeast trending ridgeline in the northwest portion of the property. Area 2 is located on another ridgeline, parallel to Area 1 that extends from the center of the property towards the northeast corner. Area 3 is located in the southwest corner of the property and Area 4 is located west of Area 3 and south of Area 2. Both Areas 1 and 2 are the primary focus of this report. Area 1 has been the primary focus of past exploration efforts and continues be a key area of this report.

The two existing mineral leases located within the PRC exploration agreement area (ML 48698.0 MC and ML 48699.0 MC) are metallic mineral leases that includes aluminum but not potash. PRC can explore and delineate potash resources on these leases. PRC is working to secure an agreement for the 155ac section that extends across Area 2, either through an agreement with the lessee or through an adjudication process through SITLA.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESSIBILITY

The Blawn Mountain Project is located about 20 miles west of the Union Pacific (UP) Railroad route, 15 miles south of Highway 21, and 50 miles west of Interstate 15, the main north-south travel corridor through Utah. The area is reached by traveling west from Milford 24 miles on Route 21 and then turning south onto a graveled secondary road and traveling approximately 17 miles. The coordinates for the approximate center of the property are 1,420,000 feet (ft) east and 587,000ft north, Utah State Plane, NAD 27, South Zone. All coordinates given and used in maps and plans are in feet and in the above referenced coordinate system.

5.2 ARCHAEOLOGY

Berge (1974) inventoried the archeological resources of ESI's proposed alunite mine and processing plant sites and located numerous archeological sites but none were determined eligible for the National Register (Perry, 1977). An archeological survey of the Blawn Mountain area was completed in summer 2013 and is discussed in Section 20.

5.3 CLIMATE

The Blawn Mountain area is semi-arid with hot, dry sunny summers of low humidity and cold winters. Based on climate data from the closest long-term weather station at Milford (US BLM, 1977) the climate can be describes as follows, "Average mean temperatures at Milford based on 30 years of observation range from 25.7°F in January to 74.3°F in July. Extremes range from a record low of -34°F to a record high of 105°F. Maximum temperatures in summer frequently exceed 90°F. Cold spells in winter with temperatures below 0°F occur from time to time but seldom last for more than a few days". Temperatures at the Blawn Mountain Project would likely be cooler throughout the year than at Milford because Blawn Mountain is at higher elevation. Average annual precipitation at Milford is 8.4in. with the wettest month being March and the driest month being July. Snow does not generally persist in the valleys but can blanket the mountains through the winter season (US BLM, 1977).

5.4 ENERGY CORRIDORS

Two energy corridors pass to the east of the Blawn Mountain Project both of which trend roughly north-south, as shown in Figure 5.1. The first, located 22 miles east of the property, contains the Utah Nevada (UNEV) Gas Pipeline, the Intermountain Power Project electric transmission line, and the federally designated, multimodal West-wide Energy Corridor (US Department of Energy, 2011). The second located approximately 25 miles east of the property contains the Kern River gas pipeline. The West-wide Energy Corridor follows State Highway 21, 12 miles north of the Blawn Mountain Property (US BLM, 2011[a]).

5.5 Grazing

A grazing allotment map (US BLM, 2011[b]) shows boundaries of cattle and sheep grazing allotments and boundaries of wild horse herd management areas (HMA) on the federal lands surrounding the Blawn Mountain Project. The entire Blawn Mountain Project is within grazing allotments administered by SITLA. The project area is not within a HMA but the Four Mile HMA adjoins the south boundary of the Blawn Mountain Project and covers more than 100 square miles.

5.6 Local Resources

Construction of a mining operation and processing plant at Blawn Mountain would require local resources of contractors, construction materials, employees and housing for employees, and energy resources. The Milford area offers construction material such as sand and gravel from several sources, crushed limestone from the Graymont Lime Plant in the Cricket Mountains, north of Milford, crushed stone from a railroad ballast quarry just north of Milford, and Portland cement from the Ashgrove Cement West Plant at Leamington approximately 90 miles away. The nearby towns of Delta, Milford, Fillmore, and Cedar City, and Beaver could provide mine and plant workers and furnish housing for company employees. There are two nearby electrical corridors and there is sufficient electricity being supplied within the region from coal, geothermal and wind power plants.

5.7 Physiography

Topographically, the Blawn Mountain Project is situated in a typical Basin and Range setting. The ranges, consisting of north-south trending mountains, are generally steep and rugged with mountaintop elevations up to 7,900ft above sea level. The ranges are separated by fault graben basins with deeply incised drainages. Pine Valley lies to the west of the Wah Wah Range and Wah Wah Valley lies to the east. The Blawn Mountain deposits occupy three of the smaller ridges in the southern Wah Wah Range. The mineral tracts include substantial low relief areas that have potential to support mine and plant facilities.

Seasonal runoff is channeled away from the Blawn Mountain Alunite deposits by two main drainages. Blawn Wash drainage carries runoff to the southeast toward Escalante Valley and Willow Creek drainage carries runoff into Wah Wah Valley to the northeast.

5.8 SEISMOLOGY

The Blawn Mountain Project area has low potential for occasional moderate earthquakes. Perry (1977) discussed the possibility of weak earthquakes in the Blawn Mountain area due to its proximity to the transition zone between the Colorado Plateau and Basin and Range physiographic provinces, an area termed the Intermountain Seismic Belt. Perry also mentions “a non-instrumented report of an earthquake with a modified Mercalli Intensity of III (nominally Richter 3.1), recorded October 26, 1885 between 0800 and 0900 hours near Frisco, about 12 miles northeast of the project area”. Pankow, Arabasz, and Berlacu (2009) refined the seismic history of the region and delineated an area of mildly anomalous seismic activity in the Escalante Valley. The most significant earthquake that is discussed for the area is the 1908 Milford earthquake of local Richter Scale magnitude (ML) 5±.

5.9 SURFACE OWNERSHIP

The Blawn Mountain Project is composed of Utah State-owned land managed by SITLA. The lands immediately around the property are predominantly federal lands managed by the BLM along with additional SITLA tracts.

5.10 TRANSPORTATION

The Blawn Mountain Project is accessed by secondary roads maintained by Beaver County and located near highway and rail transportation. State Highway 21 passes 12 miles to the north of the property, connecting Milford, Utah with Ely, Nevada to the northwest. State Highways SR-21 and SR-130 pass about 30 miles east of the property connecting Milford, Utah to Cedar City, Utah to the south. I-15 is located approximately 63 miles to the east southeast accessed via SR-21 and SR-130. The UP Railroad route connecting Salt Lake City, Utah to Las Vegas, Nevada passes approximately 20 miles to the east of the Blawn Mountain Project.

5.11 VEGETATION

The Blawn Mountain Project is located in the pinyon-juniper community as defined by the BLM (1977). This flora community is characterized by occurrence of Utah Juniper, single-leaf and double-leaf Pinyon Pine. Occasional patches of Mountain Mahogany, Gamble Oak, Ponderosa Pine, and Aspen occur at higher elevations with greater rain fall amounts. The valleys of the area have been extensively chained to remove Juniper and Pinyon and improve grass growth for grazing.

Vegetation in the valleys is mixed shrub-grass community characterized by seven shrubs: Big Sagebrush, Black Sagebrush, Big Rabbitbrush, Small Rabbitbrush, Greasewood, Winterfat, and Matchweed. Galleta, Indian Ricegrass and Cheatgrass are the most common grasses across the property. A survey of the Blawn Mountain Project completed in 2013 did not identify any federally protected threatened or endangered (T&E) species or potential habitat. The results of this survey are discussed further in Section 20.

5.12 WATER RESOURCES

Surface Water

The Blawn Mountain Project area is in an arid portion of the state. Located at the headwaters of two drainages, one flows to the Wah Wah Valley and one flows to the Escalante drainage, surface water flows are ephemeral. Runoff events from the project site are short lived. Generally these drainages ultimately discharge to salt lakes or playas without an outlet other than evaporation.

Discharges to the south from the project area flow into the Escalante Valley. Most of these flows infiltrate into the groundwater system. However, only a small percentage of flows from larger duration storms reach the main drainage channel of the valley. The Escalante Valley flows northward toward Sevier Lake. Limited surface water is available for water rights in the valley.

Flows into the Wah Wah Valley are collected first in the Wah Wah Valley Hardpan, which occupies the lower (northern) end of the Wah Wah Valley, and then if there are excess flows, the discharge flows north to Sevier Lake. A few shallow stock ponds along the flanks of the Wah Wah Valley have water rights to capture periodic runoff.

Similar conditions exist in the Pine Valley located to the west of the Wah Wah Mountains. Discharges from the upper Pine Valley are collected in the playa in lower Pine Valley located to the west of the Wah Wah Mountains and is called the Pine Hardpan.

Sevier Lake and Pine Hardpan hold large quantities of poor quality water in deep evaporite and clay deposits, and have occasional standing water. PRC completed a surface water monitoring program during the fall of 2013 to document the site specific flow conditions and water quality.

Groundwater

The project area is on the east edge of the Basin and Range Province, in which faults divide uplands from sediment-filled valleys (horst and graben structure). Deep bedrock is crystalline rocks (gneiss) overlain by Paleozoic carbonate rocks (limestone and dolomite). Carbonates and gneiss are commonly intermingled by faulting. Lower Tertiary volcanics, mostly tuff and andesite, cap the highlands. Quaternary alluvium filled the down-faulted valleys to depths of several hundred feet as they deepened.

The Blawn Mountain Project area has no perennial streams, indicating that near-surface groundwater in the project area is limited. PRC commissioned a spring and seep survey in the spring of 2013 to assess the occurrence of water sources in the project area. This study covered about 20 square miles and assessed surface and groundwater flows. A total of 50 spring and seep sites were identified (see Figure 5.2). Many of these sites did not have flowing water, but were either damp spots, salt stains/accumulations on the surface, or phreatophytic vegetation areas. A limited number of water sources were identified which physically had water with flows ranging from 0 to 1.4 gallons per minute (gpm). For these locations, field water quality data were collected for pH, specific conductance, total dissolved solids (estimated), and temperature. Table 5.1 presents a summary of the data from the water sources that had water.

TABLE 5.1 SUMMARY OF FLOWS AND FIELD WATER QUALITY INFORMATION

Point ID	Spring Data					Fall Data				
	pH	Temp (°C)*	SC (mS)**	TDS (ppt)***	Q (gpm)****	pH	Temp (°C)*	SC (mS)**	TDS (ppt)***	Q (gpm)****
BWSU 0	7.55	11.60	2.17	1.12	1.41	7.96	15.20	1.82	0.90	0.79
BWSU 1	7.72	14.70	2.13	1.06	0.51	7.48	15.90	2.28	1.14	0.55
BWSU 2	7.34	14.70	2.04	1.02	1.17	7.51	14.30	1.96	0.98	0.75
BWSL0	N/A	N/A	N/A	N/A	DRY	7.41	18.00	2.11	1.05	0.04
BWSL1	N/A	N/A	N/A	N/A	DRY	7.10	15.20	2.43	1.22	0.35
BWSL5	N/A	N/A	N/A	N/A	DRY	6.87	16.00	1.96	0.98	0.99
BWT 0	7.61	20.50	3.50	1.75	1.06	7.52	16.20	3.29	1.64	0.16
ROAD SPRING	7.78	15.50	3.62	1.81	0.10	N/A	N/A	N/A	N/A	DRY
SEEP 02	7.09	21.00	3.54	1.79	0.00	7.57	14.90	3.40	1.70	0.36
SEEP 03	N/A	N/A	N/A	N/A	DRY	8.38	14.80	4.04	2.00	<0.01
SEEP 04	6.86	17.40	11.36	5.65	0.00	N/A	N/A	N/A	N/A	DRY
SEEP 07	7.41	14.90	3.62	1.81	0.00	N/A	N/A	N/A	N/A	DRY
SEEP 08	7.19	14.30	5.17	2.56	0.00	N/A	N/A	N/A	N/A	DRY
SEEP 10	N/A	N/A	N/A	N/A	DRY	7.68	15.50	1.89	0.94	0.06
SEEP 15	7.59	26.50	4.65	2.12	0.00	N/A	N/A	N/A	N/A	DRY
WS 0	7.10	11.50	1.97	0.90	0.59	7.04	19.10	1.61	0.80	0.00
WS 1	7.34	15.90	3.63	1.66	0.00	7.52	23.10	1.58	0.79	<0.01
WS 2	7.04	10.30	4.37	1.95	0.58	7.11	17.80	1.81	0.90	0.90
WS MAIN 1	7.98	16.20	2.62	1.21	1.34	8.04	24.70	1.96	0.98	0.90
WS MAIN 2	7.96	15.70	2.90	1.33	1.34	8.22	22.80	1.99	0.98	0.90
WS MAIN 3	7.88	19.00	3.84	1.75	0.00	7.75	22.70	1.99	0.99	<0.01
Water Well	N/A	N/A	N/A	N/A	DRY	7.43	20.40	0.58	0.29	N/A

*Temperature – degrees Centigrade
**SC – Specific Conductance in milli Siemens
***TDS – Total Dissolved Solids (estimated) parts per thousand
****Q – Discharge in gallons per minute

To assess the potential for groundwater issues during mining, 10 monitoring wells have been installed in the mine area by PRC. The wells are located in the valleys surrounding the project area. The zones monitored are the volcanic tuffs and andesite flows. Two of the wells were dry, the remainder encountered water. Water levels encountered in these wells are close to the surface. Flow to the wells is limited. For the wells completed in the andesite flows, the flows are in the range of 5 to 10gpm. For the wells in the volcanic tuffs, the flows are in the range of 0 to 0.5gpm.

Groundwater hydraulic parameters were gathered for eight of these wells. Six of the wells in the andesite flows were evaluated for both pumping and recovering water levels. Due to the relatively slow response of the two wells in the volcanic tuff, the analyses for these wells were limited to using recovery water levels in the wells following pumping. The transmissivity values determined from these tests are presented in Table 5.2.

TABLE 5.2 SUMMARY OF AQUIFER PARAMETERS

Well ID	Lithology	Transmissivity (ft^2/d)
MW-1	Andesite	15.56
MW-2	Andesite	DRY
MW-3	Tuff/Clay	2.57
MW-4	Andesite	4.74
MW-5	Tuff/Clay	6.47
MW-6	Andesite	0.13
MW-9	Andesite	1.59
MW-10	Andesite	0.72
MW-11	Andesite	DRY
MW-13	Andesite	7.31

These wells show that groundwater is limited and no extensive groundwater source is found in these zones. Water to support mining and processing will need to be produced from groundwater in the adjacent areas or from deeper sources on site. Three water bearing formations can be described:

- Alluvium up to hundreds of feet thick, and from Pliocene to recent in age
- Tertiary volcanic rocks capping the highlands in the project area
- Carbonate rocks underlying, and fault-splintered into basement gneisses.

The region is described in the hydrology literature as part of the Sevier Desert and the Great Basin, with groundwater systems dominated by coupled alluvium and carbonate rocks. Alluvium is described generally as containing some fresher water in valley sides and heads, and salty water in closed valley playas. Clays usually reduce permeabilities, but high yields can be obtained from deeper wells near zones where coarser sediments exist.

Volcanic rocks include tuffs and andesite flows capping highland areas; the Blawn Mountain Project's economic deposits occur in altered volcanic rocks. Volcanic rocks capping ridges receive higher recharge than the regional average due to higher elevation. Groundwater in the area show higher salt content than expected and feeds springs at the base of these rocks and in gulches incised into them. These springs are generally documented as water rights held for stock and wildlife, have flows less than a few gpm, and some of them may be seasonal.

Carbonates include limestones and dolomites which have variable permeability and may have yields associated with solution channels. These strata typically have poorer water quality. Regionally, the carbonate strata constitute a slow-flowing, saline aquifer which discharges to some low basins such as Sevier Lake, and also on longer paths to Great Salt Lake. One potential target for water supply assessment would be the small areas of limestone located within highly faulted volcanics near the project site.

United States Geologic Survey (USGS) studies indicate substantial groundwater resources are present in the Wah Wah and nearby Pine Valley drainages. Stephens (1974 and 1976) evaluated the hydrology of both the Wah Wah and Pine Valleys. Based on his findings, the water balances for the Wah Wah and Pine Valleys indicate:

TABLE 5.3 WATER BALANCE WAH WAH VALLEY

Description	Source	Estimated Quantity (acft/yr)
Recharge		
	From Precipitation in Drainage Basin	7,000
	Subsurface Inflow from Pine Valley	3,000
	Total	**10,000**
Discharge		
Evapotranspiration from	Stream Channel alluvium	40
	Wah Wah Springs	600
Flow and Pumpage from wells and springs	Stream-channel Alluvium	50
	Older alluvium	2
	Extrusive Rocks	24
	Intrusive Rocks	24
	Quartzite and metasedimentary rocks	10
	Wah Wah Springs	800
	Total	**1,500**

TABLE 5.4 WATER BALANCE FOR PINE VALLEY

Description	Source	Estimated Quantity (acft/yr)
Recharge		
	From Precipitation in drainage basin	21,000
	Total	21,000
Discharge		
Evapotranspiration from:	Stream Channel alluvium	**5,500**
Flow and Pumpage from wells and springs:	Man usage	5
	Pine Valley Springs	650
	Subsurface outflow to Wah Wah Valley	3,000
	Total	**9,155**

Studies by ESI, conducted in the 1970s, demonstrated that the alluvial fill in the southern portion of the Wah Wah Valley was very productive and able to supply a majority of that projects water resources. Pumping rates ranged from 875gpm with a stabilized drawdown of 93.5ft to 1353gpm with a drawdown of 113ft. Following pumping, the water level recovered within 14 minutes to within 3.5ft of the pre-pumping static water level. PRC is in the process of evaluating these resources and securing water rights to support the Blawn Mountain Project.

5.13 WILDERNESS DESIGNATION

The Blawn Mountain Project has not been designated for study or inclusion for wilderness. In 1999 the BLM re-inventoried its lands for suitability for classification of US wilderness designation. Part of the Wah Wah Range north of the Blawn Mountain Project, the Central Wah Wah Mountains, met the wilderness re-inventory criteria (US BLM, 2011[c]). The southern boundary of the re-inventoried Central Wah Wah wilderness area is approximately five miles north of the northern border of the Blawn Mountain Project.

5.14 WILDLIFE

Deer, wild horses, antelope, cougar, raptors and other birds, coyote, bobcat, and fox all are common animals in the area. A survey of the Blawn Mountain Project completed in 2013 did not identify any T&E species or potential habitat. The results of this survey are discussed further in Section 20. A BLM map of wildlife management areas (US BLM, 2011[d]) shows no special management areas within the Blawn Mountain Project area.

5.15 CONFLICTING DEVELOPMENT

The Blawn Mountain Project area has a long history of mineral exploration, grazing, and outdoor recreation. No historical land use conflicts are known for the property and if the property is developed for mineral extraction, no future land use conflicts are anticipated. Recently, southwest

Utah has experienced extensive conventional energy, alternative energy and energy infrastructure development in the vicinity of Blawn Mountain Project area. The Cedar City Field Office of the BLM compiled a draft map for their Resource Management Plan (US BLM, 2011[e]) indicating tracts with geothermal, wind power, and solar energy potential would not conflict with Blawn Mountain Project's development.

6 HISTORY

The extensive hydrothermal alteration of the southern Wah Wah Range has long been known and most of the prospecting in the area has been for metallic minerals associated with the hydrothermal alteration. Whelan (1965) was the first known geological investigation that discussed production of the Blawn Mountain Project's alunite as a commodity. In the early 1970s ESI was simultaneously investigating deposits in Colorado, Arizona, Nevada, California and several deposits in Utah. In 1970 ESI started the first systematic exploration of Blawn Mountain which they called the NG Alunite property; results were encouraging. That same year ESI entered into a joint venture agreement with National Steel Corp. of Pittsburgh, Pennsylvania and the Southwire Company of Carrollton, Georgia to open an alunite mine as a source of alumina to supply the National Steel/Southwire's jointly-owned aluminum plant at Hawesville, Kentucky (Parkinson, 1974). The partnership was called The Alumet Company and was headquartered in Golden, Colorado. ESI owned 50% of the partnership and National and Southwire each owned 25%.

The NG Alunite deposit is a circular cluster of four alunite areas (Figure 6.1). These four areas were mapped, surface sampled, and drilled. While ESI continued investigating other deposits, they focused most of their resources on the NG Alunite deposits. Initial results convinced ESI to further focus development on Area C, now referred to as Area 1, with the intention of investigating it as their first mine site. Additional surface sampling, drilling, and collection of bulk samples (for pilot plant testing) at Area 1, were completed before April 1974. Seven test pits (ESI, 1989) were excavated in the north end of Area 1 for samples to send to the Alumet pilot plant in Golden, Colorado; the largest sample was a 3,000t (Krahulec, 2007) sample from a pit identified as number 5. The pilot plant (designed by HRI) had the capacity to process 12 to 18 tons per day (tpd) and operated for three years with occasional shutdowns to modify the process (ESI, 1989).

Alumet's concept was to build an integrated plant that would produce 500,000 tons per year (tpy) of alumina with by-products of 450,000tpy of sulfuric acid, 250,000tpy SOP, and aluminum fluoride (Parkinson, 1974). To achieve this level of production, Alumet planned to mine 4Mt of alunite per year for 25 years (Perry, 1977). Alumet acquired subsidiary mining properties and resources needed to support the alunite plant. Alumet acquired a phosphate property near Soda Springs, Idaho. Phosphate was to be mined and calcined in Idaho and shipped to the NG Alunite Plant where the by-product sulfuric acid would be used to make phosphate fertilizer. The Soda Springs, Idaho phosphate mine was also intended to produce by-product vanadium (Parkinson, 1974). Alumet also acquired a coal property on the Wasatch Plateau to the northeast in central Utah to provide fuel for the alunite plant. Local water rights were acquired and water wells were drilled and tested. Local aggregate sources were evaluated for use in construction of the plant.

During this time Alumet refined their resource calculations, commissioned feasibility and environmental studies, continued improving their metallurgical process, and commissioned design of an open pit mine on the northeast end of Area C with a plant and tailings pond adjacent to the northeast (Figure 6.2). Despite this advanced stage of development, plant construction and mining never occurred due to a challenging US economic environment in the 1980s and depressed pricing for alumina and potash.

Previous resource estimates are difficult to relate to the current assessment for several reasons. Historical estimates centered on alumina as the primary product with potash as a secondary product. Cut-off grades were based on Al_2O_3 grades versus K_2O. Previous reserve estimates for Area 1 are summarized in Table 6.1. ESI initially carried out resource estimates in 1972 to include Areas 1 to 4. Chapman, Wood, and Griswold Ltd. (CW&G) were retained to calculate a corresponding estimate. Pincock, Allen and Holt and Computer Associates Inc. (PAH/CAI) calculated the resources for the north end of Area 1 in 1975. None of these studies are deemed to be NI 43-101 compliant although reasonable methodologies were applied at the time.

Table 6.2 presents historical resource and reserve estimates for all four areas that were part of the NG Alunite project. Previous resource estimates did not specify potassium grades. Table 6.3 provides calculated K_2O and K_2SO_4 contents based on Al_2O_3 contents for the historical estimates in Table 6.2. In recent analytical work completed by PRC in a validation drilling program, Norwest has observed a direct linear correlation between K_2O and Al_2O_3 values. Based on this correlation, a multiplier of 0.2809 is applied to Al_2O_3 to derive K_2O content. Potassium sulfate, K_2SO_4, is calculated from K_2O using a factor of 1.8493.

TABLE 6.1 AREA 1 HISTORICAL RESERVE ESTIMATES

Category	ESI		CW&G		PAH/CAI	
	Tons	**%Al_2O_3**	**Tons**	**%Al_2O_3**	**Tons**	**%Al_2O_3**
Proven	119,900,000	14.3	89,000,000	13	129,400,000	14
Probable	22,700,000	12.8	62,800,000	13.2	17,700,000	14.8
Inferred	36,100,000	14.1	Not estimated		18,015,000	17.1
Total	**178,700,000**	**14.1**	**151,800,000**	**13.1**	**165,185,000**	**14.4**

Table 6.2 Historical Resource and Reserve Estimates for Blawn Mountain

Deposit	Ore (000 Tons)	Alunite (%)	Al_2O_3 (%)	Al_2O_3 (000 Tons)	Inventory Classification	Reference
Area 1	129,400	38.3	14.00	18,155	Proven	Couzens, 1975
Area 1	17,770	40.3	14.80	2,626	Probable	Couzens, 1975
Area 1	18,015	46.7	17.10	3,079	Inferred	Couzens, 1975
Area 1	165,185	39.4	14.50	23,869	Total	Couzens, 1975
Area 2	54,400	38.5	14.30	7,779	Indicated	Walker, 1972
Area 2	124,900	39.5	14.60	18,235	Inferred	Walker, 1972
Area 2	25,900	41.5	15.30	3,963	High-Grade Indicated	Walker, 1972
Area 2	179,300	39	14.50	25,999	Total	Walker, 1972
Area 3	11,600	44	16.20	1,879	Indicated	Walker, 1972
Area 3	281,400	44	16.20	45,587	Inferred	Walker, 1972
Area 3	7,300	47	17.30	1,263	High-Grade Indicated	Walker, 1972
Area 3	293,000	44	16.20	47,466	Total	Walker, 1972
Area 4	51,700	36.5	13.50	6,980	Indicated	Walker, 1972
Area 4	49,200	38	14.10	6,937	Inferred	Walker, 1972
Area 4	100,900	37	13.80	13,924	Total	Walker, 1972
Total	**738,385**	**41.1**	**15.0**	**111,175**	**Grand Total**	

Table 6.3 Calculated K_2O and K_2SO_4 Concentrations for Historical Resource and Reserve Estimates for Blawn Mountain

Deposit	Ore (000 Tons)	K_2O* (%)	K_2O (000 Tons)	K_2SO_4* (%)	K_2SO_4 (000 Tons)	Inventory Classification	Reference
Area 1	129,400	3.98	5,147	7.36	9,518	Proven	Couzens, 1975
Area 1	17,770	4.20	747	7.78	1,382	Probable	Couzens, 1975
Area 1	18,015	4.86	875	8.98	1,618	Inferred	Couzens, 1975
Area 1	165,185	4.12	6,804	8.98	14,840	Total	Couzens, 1975
Area 2	54,400	4.06	2,210	7.51	4,087	Indicated	Walker, 1972
Area 2	124,900	4.15	5,181	7.67	9,580	Inferred	Walker, 1972
Area 2	25,900	4.35	1,126	8.04	2,082	High-Grade Indicated	Walker, 1972
Area 2	179,300	4.12	7,386	8.04	14,412	Total	Walker, 1972
Area 3	11,600	4.60	534	8.51	987	Indicated	Walker, 1972
Area 3	281,400	4.60	12,951	8.51	23,950	Inferred	Walker, 1972
Area 3	7,300	4.91	359	9.09	663	High-Grade Indicated	Walker, 1972
Area 3	293,000	4.60	13,485	9.09	26,630	Total	Walker, 1972
Area 4	51,700	3.84	1,983	7.09	3,667	Indicated	Walker, 1972
Area 4	49,200	4.01	1,971	7.41	3,645	Inferred	Walker, 1972
Area 4	100,900	3.92	3,956	7.41	7,474	Total	Walker, 1972
Total	**38,385**	**4.29**	**31,675**	**7.93**	**58,577**		

*Calculated from Equivalent Al_2O_3 Concentrations, $3.52K_2O \Rightarrow Al_2O_3$; $1.8493K_2O \Rightarrow K_2SO_4$

In 2011, PRC initiated a validation drilling program on Area 1 primarily to validate the previous exploration efforts. A total of 34 holes were completed on Area 1 between October 2011 and February 2012. The drill holes were twinned to locations of previous ESI drill holes using coring and reverse circulation methods.

Norwest (2012) concluded that the PRC validation drilling program had adequately tested the Area 1 deposit, both spatially and in number of twinned drilling locations. A three dimensional geological block model (3DGBM) was constructed by Norwest for Area 1 using historic and PRC validation drilling data. Norwest subsequently used the 3DGBM for reporting of resources for Area 1 in accordance with CIM Standards on Mineral Resources and Reserves, effective date April 16, 2012 (Norwest, April 2012). At a 1% K_2O cut-off grade, the combined measured plus indicated resource for Area 1 was estimated to be 162Mt carrying an average grade of 3.23% K_2O and 13.90% Al_2O_3. The calculated potassium sulfate grade (K_2SO_4) at a 1% K_2O cut-off grade was estimated to be 5.98%. As of April 16, 2012 approximately 66% of the identified resource was classified as measured resource and 34% as indicated resource.

Between July and September, 2012, PRC completed a 50-hole infill drilling program in Area 2 to define the extent of alunite mineralization. Previous exploration in Area 2 was limited to 18 drill holes completed by ESI. Three (3) of the infill drill holes were completed using wire-line slim coring (HQ core) methods, an additional three (3) drill holes completed using wire-line large diameter coring (PQ core) methods and the remaining 44 drill holes completed using reverse-circulation (RC) methods. North American Exploration Company (NAE) managed logistics, logging, and sampling for the infill drilling program with Norwest providing QA/QC management of all procedures, data collection, sampling, and chain of custody.

Following completion of the 50-hole infill drilling program in Area 2, Norwest believed there was sufficient geologic and analytical data to support a resource estimate for Area 2 in addition to Area 1. A 3DGBM was constructed by Norwest for Area 2 using the infill drilling data and ESI drill hole data. The ESI data was only used as guide for mapping of potential mineralization because only lithologic descriptions were available for these holes with no supporting assay grade data. Norwest subsequently used the Area 2 3DGBM for reporting of resources for Area 2 in accordance with CIM Standards on Mineral Resources and Reserves, effective date October 30, 2012 (Norwest, November 2012). The Area 2 historic resources were reported by Norwest as part of a PEA that included both Area 1 and Area 2. The measured plus indicated historic resources and average grades reported by Norwest at this time, using a 1% K_2O cut-off grade, are outlined in Table 6.4.The minor difference in in situ tonnes for Area 1 outlined in the Table 6.4 when compared to previous estimates (Norwest, April 2012) are attributed to the removal of 2ft of material on the surface to account for potential weathering on near-surface mineralization.

TABLE 6.4 PRIOR RESOURCE ESTIMATE (EFFECTIVE DATE OCTOBER 30, 2013)

Area	IN Situ (Mt)	K_2O (%)	K_2SO_4 (%)	AL_2O_3 (%)
Area 1	156.3	3.22	5.96	13.90
Area 2	464.4	3.07	5.68	13.16

Source: Norwest, November 2012

An open pit conventional truck/shovel method was identified as the preferred mining method for Area 1 and Area 2 in the PEA (2012). Using a target K_2SO_4 requirement of 750,000tpy, a total of 121.6Mt ROM alunite in Area 1 and 387.9Mt of ROM alunite in Area 2 were identified by Norwest as potential extractable over a 30 year LOM.

There has been no known production of alunite or any other mineral resource from the Blawn Mountain Property.

7 GEOLOGICAL SETTING AND MINERALIZATION

The Blawn Mountain Project is located in the southern Wah Wah Mountains, of the eastern Basin and Range province, in an area characterized by a thick Paleozoic sedimentary section that was:

- Thrust faulted during the Sevier orogeny
- Buried under a thick layer of regionally distributed Oligocene volcanic rocks and locally derived volcanic rocks
- Extended to the west by the Basin and Range event
- Altered by H_2S rich hydrothermal alteration related to a postulated shallow laccolithic intrusive which domed, and altered the overlying calc-alkaline volcanic rock (Hofstra, 1984)
- Affected by continual erosion of the ranges contributing to colluvial and alluvial deposition in the valleys.

Blawn Mountain is located along the Blue Ribbon lineament (Rowley and others, 1978) within the Pioche mineral belt (Shawe and Stewart, 1976), a tectonic, structural, and igneous zone that contains a large number of metallic mineral mining districts with almost two dozen associated alunite vein and replacement deposits.

Figure 7.1 shows a diagrammatic cross-section through the Wah Wah Range centered on Blawn Wash and Figure 7.2 presents a diagrammatic cross-section through Area 1 at Blawn Mountain.

7.1 REGIONAL STRATIGRAPHY

Regional rock strata underlying the Wah Wah and Blawn Mountain areas are Proterozoic to Cenozoic Era in geologic age. Rock strata consist of varying types of volcanic tuffs, rhyolites, mafic flows, basalts, quartzites, limestones, dolomites, sandstones and shales. Also present are brecciated zones associated with volcanic and faulting activity.

The sedimentary and volcanic stratigraphy of the region is summarized in Table 7.1.

TABLE 7.1 REGIONAL STRATIGRAPHY

Eras	Periods	Epochs	Groups	Formations	Members
Cenozoic	Quarternary				Alluvium And Colluvium
	Tertiary	Pliocene		Steamboat Mountain	Basalt
		Miocene	Quichapa	Blawn	Rhyolite
					Tuff
					Bauers Tuff
					Mafic Flow
					Garnet Tuff
		Oligocene		Isom	Bald Hills Tuff
				Bullion Canyon Volcanics	Three Forks Tuff
			Needles Range	Lund	
				Wah Wah Springs	
				Cottonwood Wash Tuff	
				Escalante Desert	
				Tuff Of Towers Point, Volcanic Breccia	Conglomerate
		Paleocene - Eocene		Claron	
Mesozoic	Jurassic			Temple Cap	
				Navajo Sandstone	
	Triassic			Chinle	Petrified Forest
					Shinarump
				Moenkopi	
Paleozoic	Permian			Gerster Limestone	
				Pympton Limestone	
			Oquirrh	Kaibab Limestone	
				Ely Limestone	
	Pennsylvanian			Callville Limestone	
				Woodman	
	Mississippian			Gardison Limestone	
				Fitchville	
	Devonian			Pinyon Peak Limestone	
				Simonson Dolomite	
				Sevy Dolomite	
				Laketowm Dolomite	
	Silurian			Ely Springs Dolomite	
	Ordovician			Eureks Quartzite	
				Kanosh Shale	
				Juab Limestone	
				Wah Wah Limestone	
				Fillmore Limestone	
				House Limestone	
				Notch Peak	
	Cambrian			Orr	
				Wah Wah Summit	
				Tripple Limestone	
				Pierson Cove	
				Eye Of Needle Limestone	
				Swasey Limestone	
				Whirlwind	
				Dome Limestone	
				Peasley Limestone	
				Chisholm Shale	
				Howell Limestone	
				Pioche	
				Prospect Mountain Quartzite	
Proterozoic	Precambrian			Mutual	

7.2 ALUNITE OCCURRENCES

Hofstra (1984) postulates the presence of a relatively shallow laccolithic intrusion as the source of the hydrothermal fluids that created the alunite deposits, based on radial doming of the extrusive Miocene and Oligocene volcanic strata over an area of 6 miles north-south and 3 miles east-west. The laccolith may have intruded along a zone of weakness such as the Blue Mountain thrust. The high temperature H_2S rich fluid associated with the laccolith rose along the fracture zones created in the overlaying strata by the intrusion. The fluid then penetrated into the Miocene and Oligocene volcanic layers where it encountered and boiled the groundwater. With the presence of oxygen that was transported in the groundwater, the H_2S was oxidized into super-heated aqueous solutions of H_2SO_4 and the resulting solution altered the volcanic rock along fracture zones associated with normal faulting and in zones of higher porosity/permeability. The more porous the fracture zones and strata, the more mineralization occurred. The alunite alteration has been K-Ar age dated at 22.5M years ago (Hofstra, 1984).

7.3 STRUCTURAL GEOLOGY

The Blawn Mountain Project lies within the eastern Basin and Range province. During the Late Cretaceous Sevier orogeny the region was subjected to thrust faulting and folding. Major thrust faults are the Wah Wah, Teton, Dry Canyon and Blue Mountain. The Wah Wah thrust emplaced upper Proterozoic and overlying Cambrian strata over Ordovician to Pennsylvanian strata. The Teton thrust emplaced Ordovician and Silurian strata over Silurian and Devonian carbonates and the Dry Canyon thrust emplaced Silurian and Devonian carbonates over Pennsylvanian and Mississippian strata. The Blue Mountain thrust emplaced Cambrian and younger age carbonates over Jurassic strata.

Regionally there are four sets of normal faults that relate to Basin and Range block faulting. These faults generally trend west-northwest, northeast, northwest and north-south. The Blawn Wash area is a graben bounded by west-northwest and northeast faults and the bounding volcanic ridges that host the alunite mineralization.

Within the project area are several minor normal faults that offset the alunite deposits. Figure 7.3 depicts the location of these local normal faults as well as the mapped surface geology.

7.4 PROPERTY GEOLOGY

The Wah Wah Range is partly composed of a thick section of marine, Paleozoic and Triassic quartzites and carbonates (Miller, 1966) deposited in the miogeocline of the western continental shelf. This area was covered by ocean until the Jurassic Period when it was uplifted during the Sonoma orogeny. The first major deformation of this area was during the Cretaceous/Tertiary Sevier orogeny which thrusted older basement rocks over younger rocks along both the Wah Wah and Blue Mountain thrusts, contributing to the folding of the sediments associated with the upper thrust plate (Ordovician to Pennsylvania Age strata).

Regional volcanism deposited a thick layer of calc-alkaline volcanic rocks across the area presently occupied by the southern Wah Wah Mountains. The Basin and Range extensional event created much of the current topography of the area by stretching the region about 40 miles westward; creating mountains with intervening valleys separated by range-bounding, normal faults that rotate at depth into a regional decollement. Local bimodal (calc-alkaline and basaltic) volcanism also occurred in the southern Wah Wah Mountains, associated with Basin and Range extension which began about 26M years ago. The sedimentary and volcanic stratigraphy of Blawn Mountain is summarized in Table 7.2 below.

NORWEST CORPORATION

Table 7.2 Stratigraphy of the Blawn Mountain Area from Krahulec (2007) as Modified From Hofstra (1984) and Abbott and Others (1983)

Eras		Periods	Epochs	Groups	Formations	Members
Cenozoic		Quaternary				Alluvium And Colluvium
		Tertiary	Pliocene		Steamboat Mountain	Basalt
			Miocene	Quichapa	Blawn	Rhyolite Tuff Bauers Tuff Mafic Flow Garnet Tuff
			Oligocene		Isom Bullion Canyon Volcanics	Bald Hills Tuff Three Creeks Tuff
				Needles Range	Lund Wah Wah Springs Cottonwood Wash Tuff Escalante Desert	
						Conglomerate
Paleozoic	Upper Plate of Wah Wah Thrust	Cambrian			Orr Wah Wah Summit Trippe Limestone Pierson Cove Eye Of Needle Limestone Swasey Limestone Whirlwind Dome Limestone Peasley Limestone Chisholm Shale Howell Limestone Pioche Prospect Mountain Quartzite	
Proterozoic					Mutual	
Paleozoic	Lower Plate of Wah Wah Thrust	Pennsylvanian			Callville Limestone	
		Mississippian			Woodman Gardison Limestone	
		Devonian			Fitchville Pinyon Peak Limestone Simonson Dolomite Sevy Dolomite	
		Silurian			Laketown Dolomite	
		Ordovician			Ely Springs Dolomite Eureka Quartzite Kanosh Shale Juab Limestone	

7.5 MINERALIZATION

Alunite mineralization is found on four ridges that occur within the Blawn Mountain Project. Acid sulfate alteration associated with a shallow, possibly laccolithic intrusion altered the silicic-alkalic rhyolite porphyries, flows and tuffs belonging to the Miocene Blawn Formation and the Oligocene Needles Range Group. Alteration tends to be in linear bodies reflecting the role of normal faults in controlling the mineralization. Alteration is zoned away from the point of hydrothermal fluid upwelling. The mineralized ridges are erosional remnants of a once larger altered area. The alteration zoning types as described by Hofstra (1984) are summarized in Table 7.3.

TABLE 7.3 MINERAL ALTERATION ZONES OF ACID SULFATE ALTERATION AT BLAWN MOUNTAIN ALTERATION INTENSITY INCREASES FROM TOP TO BOTTOM IN THE LIST

Zone Name	Mineral Assemblage	Rock Texture Destroyed?
Low Propylitic	chlorite-calcite ± quartz	No
High Propylitic	quartz-epidote-montmorillonite-sericite ± pyrite ± kaolinite± quartz ± calcite ± illite	No
Hematite-Clay	hematite-kaolinite-chlorite-montmorillonite ± alunite ± sericite	No
Quartz-Alunite	quartz- alunite ± kaolinite ± pyrophylite ± cristoballite ± hematite	Mostly
Silica Cap	quartz ± opal ± cristoballite ± tridymite	Yes
Quartz-Sericite-Alunite	quartz-sericite-pyrite ± alunite	Yes

(Modified from Hofstra, 1984)

Krahulec (2007) described the appearance of rocks from the silica cap and quartz-alunite zone as follows, "The Silica Cap is a zone of intense silicification believed to be the near-surface manifestation of the hydrothermal channelways. The silica is typically buff, dense, and massive but may be quite porous and vuggy locally and resemble a siliceous sinter . . . . On the surface the Quartz-Alunite alteration zones are composed of white to cream to buff to gray to pink, generally fine grained, punky to dense, intermixed alunite and silica with only minor amounts of other impurities, mainly iron . . . . Alunite also occurs locally as coarse (>0.5in.), lathy, typically pink crystals in veins. Kaolinite becomes increasingly important, at the expense of alunite, in the Quartz-Alunite zone near the boundary with the Hematite-Clay zones and also where the Quartz-Alunite zones are cut by faults (Walker, 1972). Dickite (a high-temperature member of the kaolinite group) is reported by Whelan (1965) and Thompson (1991) in the Quartz-Alunite zone".

Figure 7.4 depicts mapping by Hofstra of the alteration facies in the Blawn Mountain area and its effect on topography. The extremely erosion resistant Silica Cap forms the tops of peaks and the underlying highly erosion resistant Quartz-Alunite facies forms the steepest parts of the ridges. In cross section the alteration zones have two basic forms, a nested-cone geometry and a relatively flat-lying form, as shown in Figure 7.5. Krahulec gives the following description of the two geometries, "The cone-shaped (narrow end at the base) zones are interpreted as the primary area of strong hydrothermal upwelling . . . . and the adjoining flat-bottomed zones are recognized as permeability-controlled areas above the paleo-ground-water table where steam-heated H_2S is oxidized to H_2SO_4. Only the central portion of Area C (Area 1) at Blawn Mountain is clearly a funnel-shaped zone. The other flat bottomed alunite zones are strongly controlled by higher porosity and permeability of the host volcanic rocks, while the hydrothermal cones are largely independent of these factors (Hofstra, 1984)". Krahulec continues this discussion by quoting Hofstra, ". . . . The control of permeability on the degree of alteration intensity is most important near the margins of Quartz-Alunite altered zones. Alteration is pervasive and unaffected by variations in the permeability of the host rocks". The alteration zones tend to be thicker in cone-shaped areas than in flat-lying areas. It is possible that there were more cone-shaped feeder zones but they were eroded or are buried under valley fill.

Figure 7.6 shows the geometry of the Area 1 alunite alteration zone and Figure 7.7 shows the geometry of the Area 2 alunite alteration zone. Both figures are derived from the block model used in the resource calculations presented in this report.

8 DEPOSIT TYPES

There is no known formal industrial mineral ore deposit model for alunite. The characteristics for a model and some exploration criteria are derived from three publications: Hall (1978), Hall and Bauer (1983), and Hofstra (1984).

The local alunite deposit has been described, in the above mentioned publications, as hydrothermal alteration of calc-alkaline volcanic rocks.

9 EXPLORATION

The Blawn Mountain area was first evaluated by ESI as part of a nationwide alunite exploration program in 1969 which included literature searches, aerial reconnaissance for the bleached alunite zones, and field studies. In 1970 ESI started the first systematic exploration of the Blawn Mountain Project which they referred to as the NG Alunite property. Initial exploration focused on four separate mineralized zones located on along three ridges. All four of these mineralized zones are completely within the current PRC lease holding. ESI conducted mapping, surface sampling and drilling before focusing its attention on the northwest trending ridge now referred to as Area 1. Figure 9.1 shows the rotary drill locations completed by ESI at Blawn Mountain.

After acquiring the mineral leases in 2011, PRC initiated a validation drilling program on Area 1 primarily to validate the previous exploration efforts. Under the guidance of NAE, a combination of 19 core holes and 16 reverse circulation holes were completed on Area 1 between October 2011 and February 2012. During Norwest's first site visit in February, additional recommendations were made to the validation drilling program that included the two final reverse circulation holes and some adjustments to the sample preparation procedures. All 35 drill holes were twinned to locations of previous drill holes completed by ESI.

A second drilling program was initiated by PRC in July of 2012. The drilling program included 17 additional holes on Area 1, 50 holes on Area 2, two holes on Area 4, and 21 holes on the ridgeline extending southwest of Area 1 now referred to as the Southwest Extension. A total of 90 drill holes were completed including 74 reverse circulation holes, eight HQ core holes, and eight PQ core holes. PQ core holes were completed to collect material for metallurgical testing. A total of 32,392ft were completed in the reverse circulation and core drilling program. In addition to the exploration drilling, PRC completed 10 groundwater monitor wells in valley fill material to begin baseline characterization of near-surface hydrology. The 10 monitor wells represent a total drilling footage of 2,400ft.

A third reverse circulation drilling program was conducted in January and February of 2013. The program included two holes on Area 1 and 17 holes on Area 2 for a combined total of 8,310ft. The primary purpose of the drilling was to further increase geologic assurance for resource assessment.

In addition to drilling, Norwest has conducted additional field investigations to:

- Further assess faults and fractures on Area 2 and the role they play in controlling mineralization and silicification.
- Investigate a prominent fault trending northwest from the north side of Area 1 as a possible exploration target for additional mineralization.
- Reconnaissance of Area 3 for future exploration.

10 DRILLING

10.1 HISTORICAL DRILLING

ESI company records indicate a total of 320 drill holes were completed on their NG Alunite deposit. Within Area 1, 287 holes were completed, 18 holes at Area 2, 12 holes at Area 3, and 3 holes on Area 4. Six of the drill holes located in Area 2 are located within a 155ac tract where surface rights are jointly shared between SITLA and a third-party. Additionally, the rights to metallic resources for this tract are owned by a third party, with PRC controlling the rights to potash and other minerals.

ESI used air track percussion drilling and conventional rotary drilling in its exploration efforts. Air track drilling was primarily used as a prospecting tool to test the ground where there were poor bedrock exposures. Rotary drilling was used to define subsurface geology and collect samples for analysis.

There are numerous drill site locations where multiple holes have been drilled. This was due to:

- Air track drilling being first used at several sites where there were poor surface exposures to identify sites to be followed with rotary drilling.
- Adverse drilling conditions were encountered at several sites that required abandoning a drill hole, moving over a few feet on the drill pad and making another attempt.
- Several locations where holes were re-entered or drilled a second time to collect additional information.

ESI completed its drilling in three stages:

- Reconnaissance drilling in 1971 completing 10 holes for a total of 2,650ft. Three holes were completed on Area 1, four holes at Area 2, and three holes at Area 3.
- Exploration drilling in 1972 completing an additional 42 drill holes. Sixteen (16) holes were completed at Area 1 for a total of 4,438ft, 14 holes were completed at Area 2 for a total of 2,865ft, 9 holes were completed at Area 3 for a total of 2,590ft, and 3 holes were completed on Area 4 for a total of 740ft.
- Development drilling in 1973 and 1974 on Area 1. Drilling was roughly aligned to a 300 (NW-SE) by 500 (NE-SW) grid pattern oriented to the ridgeline. A total of 268 air track and rotary holes were completed for a total footage of 46,267ft.

ESI did not maintain complete records for most of the air track drill holes and some of the abandoned holes. Complete records were only maintained for holes with assays. Norwest has geologic logs for all holes, but is missing coordinates for the air track holes and some abandoned holes.

10.2 PRC DRILLING

PRC completed a validation drilling program on Area 1 between October 2011 and February 2012. All drill sites were twinned to locations of previous drill holes completed by ESI and were oriented to provide adequate spatial representation of the deposit. Nineteen (19) of the PRC holes were drilled using wire-line slim coring methods, continuously collecting HQ core. A total 6,764ft of drilling was accomplished through core drilling with an average recovery of 91%. The remaining 15 drill holes were completed using RC drilling equipped with either a down-hole hammer or deep-hole bit. A total of 8,050ft were completed with RC drilling.

NAE managed logistics, logging, and sampling for the PRC program. Two different drilling contractors were used in the RC drilling. It was quickly recognized by NAE that the first drilling contractor was having difficulty recovering sufficient sample volumes in the RC drilling. After several measures were employed to improve sample returns, NAE brought in a second drilling contractor to complete the RC drilling. The second contractor did not experience the same problems and was able to deliver adequate sample volumes and complete the drilling program. The first drilling contractor completed seven RC holes for a total of 4,210ft. None of the samples from these seven holes have been used or incorporated by PRC in their evaluation of the Blawn Mountain Project. The second drilling contractor completed 8 holes for a total of 3,840ft. Samples and data from these holes are being used by PRC in their evaluation of the deposit.

PRC completed a second substantial drilling program at Blawn Mountain in the summer of 2012. Drilling was accomplished using RC, wire-line coring for HQ and PQ core, and conventional rotary methods. The program included a total of 90 holes in Areas 1, 2, and 4 plus 10 groundwater monitor wells.

The second drilling program accomplished several goals:

- Further delineate resources on Area 1 with nine RC holes and five HQ core holes.
- Explore and define potential alunite resources on Area 2 with 44 RC holes and three HQ core holes.
- Explore and define potential alunite resources on the ridge extending southwest from Area 1 with 19 RC holes and 2 HQ core holes. This area is referred to as the Southwest Extension.
- Complete five PQ core holes on Area 1 and three PQ core holes on Area 2. Core from the PQ holes was used to develop bulk metallurgical samples.
- Perform resource reconnaissance in Area 4.
- Complete 10 widely-spaced rotary holes in the alluvial/colluvial areas surrounding the alunite deposits to collect samples for overburden testing and observe groundwater conditions. All 10 rotary holes were subsequently converted to monitor wells to observe and sample groundwater conditions.

All core drilling was completed by a contractor using a track-mounted LF-70 core drill. A total of 2,804ft of HQ core drilling was accomplished with an average core recovery of 91% and a total of 4,054ft of PQ size core drilling with a recovery rate of 95%. All RC drilling was completed by a separate contractor utilizing up to three RC rigs simultaneously on the project area. All three RC rigs used on the program were track-mounted Schramm DLD 1000 rigs, equipped with 4in. pipe and wet rotary splitters for sample collection. All RC drilling was done with foam injection to minimize dust and water consumption.

A third drilling program totaling 21 holes at 18 locations was completed by PRC in January and February of 2013. Two RC holes were completed on Area 1 and 16 RC holes were completed on Area 2. All drilling was to improve resource delineation and geologic assurance. Six of the drill holes on Area 2 were drilled as angle holes, specifically targeting potential resources that could not be accessed with roads and vertical holes. Difficult drilling conditions required abandoning and restarting drill holes at three of the angle hole sites. A total of 480ft of drilling was completed on Area 1 and 7,830ft on Area 2.

A fourth drilling program was conducted in the spring/summer of 2013 in the southern end of the Wah Wah valley. Three test bores were drilled for a total of 3,340ft. This evaluation was to assess depth to bedrock and to assist in determining the presence of groundwater. In the fall of 2013, two of the bore holes were completed as observation wells to assist in assessing the potential to develop a well field in the area.

Throughout the second and third drilling programs field operations and geologic logging was completed by NAE. Norwest provided an onsite QA/QC manager who oversaw all data collection and sampling for the programs for the two drilling programs.

Table 10.1 summarizes the drilling that has been completed by PRC from October 2011 through July 2013. Figure 10.1 shows the locations of all drilling completed by PRC on the main tract of land (excluding ML 52364).

TABLE 10.1 PRC DRILLING SUMMARY

Phase 1 - Area 1						
Drill Hole ID	UTM83-12			Depth (Ft)	Completion Date	Type
	Easting	Northing	Surface			
BM1	279,764	4,240,018	2,235	230	10/25/2011	HQ
BM10	279,089	4,239,436	2,273	536	11/20/2011	HQ
BM11	279,309	4,239,629	2,239	610	11/26/2011	RC
BM11A	279,309	4,239,629	2,239	457	11/30/2011	HQ
BM12	280,161	4,239,820	2,264	300	12/4/2011	HQ
BM13	279,836	4,239,759	2,230	200	11/30/2011	HQ
BM14	279,533	4,239,655	2,237	370	11/15/2011	HQ
BM14A	279,533	4,239,655	2,237	650	11/30/2011	HQ
BM14B	279,533	4,239,655	2,237	651	12/06/2011	HQ
BM15	279,505	4,239,575	2,196	380	12/05/2011	RC
BM16	280,031	4,239,773	2,249	228	12/05/2011	HQ
BM16A	280,031	4,239,773	2,249	200	12/07/2011	RC
BM17	279,433	4,239,573	2,201	470	12/04/2011	RC
BM17A	279,433	4,239,573	2,201	820	12/07/2011	RC
BM18	279,937	4,239,817	2,259	198	12/10/2011	HQ
BM19	279,443	4,239,658	2,241	650	12/14/2011	HQ
BM2	279,270	4,239,337	2,211	146	10/26/2011	HQ
BM20	280,026	4,239,957	2,315	400	01/03/2012	RC
BM21	279,356	4,239,488	2,205	810	12/16/2011	RC
BM22	279,804	4,239,834	2,264	338	12/12/2011	HQ
BM23	279,490	4,239,746	2,272	198	12/14/2012	HQ
BM24	279,644	4,239,838	2,266	420	12/19/2011	RC
BM25	279,357	4,239,676	2,242	790	12/19/2011	RC
BM26	279,913	4,239,978	2,289	410	01/05/2012	RC
BM27	280,032	4,239,872	2,293	300	01/07/2012	RC
BM28	279,535	4,239,791	2,270	700	01/11/2012	RC
BM3	279,779	4,239,928	2,277	280	10/16/2011	HQ
BM3A	279,779	4,239,928	2,277	200	10/30/2011	RC
BM4	279,473	4,239,807	2,258	368	11/01/2011	HQ
BM5	279,498	4,239,891	2,250	440	11/03/2011	RC
BM6	279,144	4,239,376	2,258	301	11/14/2011	HQ
BM7	280,000	4,240,069	2,262	133	11/14/2011	HQ
BM8	279,392	4,239,776	2,247	800	11/10/2011	RC
BM8A	279,392	4,239,776	2,247	800	11/20/2011	RC
BM9	279,537	4,239,711	2,270	477	11/21/2011	HQ

TABLE 10.1 PRC DRILLING SUMMARY, CONTINUED

Phase 2 - Area 1						
Drill Hole ID	UTM83-12			Depth (Ft)	Completion Date	Type
	Easting	Northing	Surface			
PDH 1-03	280,500	4,239,928	2,186	124	08/27/12	HQ
PDH 1-04	280,222	4,240,002	2,280	240	10/11/12	RC
PDH 1-05	280,152	4,239,948	2,330	404	08/30/12	HQ
PDH 1-06	280,089	4,239,680	2,192	520	10/07/12	RC
PDH 1-07	279,655	4,239,963	2,241	100	09/02/12	RC
PDH 1-08	279,655	4,240,028	2,220	220	09/22/12	RC
PDH 1-09	279,576	4,239,959	2,245	480	09/23/12	RC
PDH 1-10	279,086	4,239,564	2,232	600	09/25/12	RC
PDH 1-11	278,953	4,239,338	2,236	394	08/27/12	HQ
PDH 1-12	279,576	4,239,959	2,245	270	09/26/12	RC
PDH 1-13	279,086	4,239,564	2,232	200	09/19/12	RC
PDH 1-14	279,174	4,239,507	2,272	560	09/21/12	RC
PDH 1-15	278,953	4,239,338	2,236	300	10/05/12	RC
PDH 1-16	278,827	4,239,266	2,244	450	08/24/12	RC
PDH 1-17	278,773	4,239,183	2,259	324	08/24/12	RC
PDH 1-18	278,474	4,239,177	2,288	200	09/22/12	RC
PDH 1-19	278,469	4,239,037	2,298	400	09/23/12	RC
PDH 1-20	278,709	4,238,893	2,259	400	10/06/12	RC
PDH 1-22	279,040	4,238,878	2,184	140	10/08/12	RC
PDH 1-23	278,375	4,239,104	2,293	370	09/15/12	RC
PDH 1-24	278,317	4,238,842	2,281	299	08/11/12	HQ
PDH 1-25	278,592	4,238,879	2,245	350	10/04/12	RC
PDH 1-26	278,427	4,238,797	2,243	300	08/11/12	RC
PDH 1-27	278,265	4,238,676	2,293	600	09/02/12	RC
PDH 1-30	278,171	4,238,591	2,298	400	10/11/12	RC
PDH 1-31	278,050	4,238,557	2,276	400	09/13/12	RC
PDH 1-32	277,991	4,238,463	2,289	499	09/22/12	HG
PDH 1-33	278,214	4,238,548	2,288	570	08/21/12	RC
PDH 1-35	278,768	4,239,010	2,215	100	10/08/12	RC
PDH 1-36	278,584	4,238,980	2,281	340	10/06/12	RC
PDH 1-37	278,455	4,238,938	2,280	240	10/04/12	RC
PDH 1-38	277,884	4,238,365	2,280	355	08/30/12	RC
PDH 1-39	279,425	4,239,700	2,252	1,066	09/14/12	PQ
PDH 1-40	279,602	4,239,825	2,269	669	09/19/12	PQ
PDH 1-41	279,726	4,239,883	2,275	454	09/22/12	PQ
PDH 1-42	279,853	4,239,960	2,282	404	09/25/12	PQ
PDH 1-43	280,114	4,239,945	2,332	559	10/04/12	PQ
PDH 1-44	277,761	4,238,284	2,247	340	08/27/12	RC

TABLE 10.1 PRC DRILLING SUMMARY, CONTINUED

Phase 2 - Area 2						
Drill Hole ID	UTM83-12			Depth (Ft)	Completion Date	Type
	Easting	Northing	Surface			
PDH-2-01	282,709	4,239,824	2,096	200	07/15/12	RC
PDH-2-03	282,578	4,239,596	2,137	220	07/14/12	RC
PDH-2-04	282,670	4,239,709	2,101	160	07/20/12	HQ
PDH-2-05	282,628	4,239,356	2,106	280	08/24/12	RC
PDH-2-06	282,680	4,239,279	2,091	620	07/19/12	RC
PDH-2-07	282,572	4,239,271	2,114	360	08/26/12	RC
PDH-2-08	282,579	4,239,134	2,097	320	07/19/12	RC
PDH-2-09	282,350	4,238,998	2,136	400	08/03/12	RC
PDH-2-10	282,485	4,238,937	2,110	400	08/01/12	RC
PDH-2-11	282,209	4,238,783	2,129	330	08/07/12	RC
PDH-2-12	282,418	4,238,809	2,106	350	08/24/12	RC
PDH-2-13	282,843	4,239,378	2,106	220	07/15/12	RC
PDH-2-14	281,725	4,239,078	2,116	220	09/03/12	RC
PDH-2-15	282,102	4,238,813	2,159	320	08/03/12	RC
PDH-2-16	282,212	4,238,893	2,133	420	07/18/12	HQ
PDH-2-17	282,318	4,238,716	2,126	300	08/21/12	RC
PDH-2-18	281,874	4,238,954	2,110	100	02/02/13	RC
PDH-2-19	282,055	4,238,578	2,150	500	08/06/12	RC
PDH-2-20	282,207	4,238,571	2,111	360	08/12/12	RC
PDH-2-22	281,729	4,238,725	2,233	220	08/08/12	RC
PDH-2-23	282,080	4,238,419	2,113	500	07/31/12	RC
PDH-2-24	281,484	4,239,032	2,124	270	09/04/12	RC
PDH-2-26	281,597	4,238,737	2,241	370	08/26/12	RC
PDH-2-27	281,791	4,238,615	2,215	400	07/20/12	RC
PDH-2-28	281,876	4,238,532	2,198	590	08/22/12	RC
PDH-2-29	281,354	4,238,905	2,130	200	09/13/12	RC
PDH-2-30	281,442	4,238,691	2,240	420	09/02/12	RC
PDH-2-31	281,595	4,238,550	2,189	160	09/02/12	RC
PDH-2-32	281,741	4,238,473	2,168	570	08/09/12	RC
PDH-2-33	282,001	4,238,334	2,111	680	07/20/12	RC
PDH-2-34	281,130	4,238,827	2,135	140	09/14/12	RC
PDH-2-36	281,429	4,238,504	2,215	504	08/01/12	HQ
PDH-2-37	281,497	4,238,341	2,163	360	08/01/12	RC
PDH-2-38	281,769	4,238,257	2,120	440	08/03/12	RC
PDH-2-39	281,023	4,238,619	2,121	200	09/15/12	RC
PDH-2-40	281,205	4,238,475	2,178	220	09/12/12	RC
PDH-2-42	281,580	4,238,162	2,157	360	07/24/12	RC
PDH-2-43	281,245	4,238,051	2,121	300	09/16/12	RC
PDH-2-44	281,428	4,237,902	2,101	520	09/16/12	RC

TABLE 10.1 PRC DRILLING SUMMARY, CONTINUED

PDH-2-45	281,068	4,238,191	2,167	300	09/12/12	RC
PDH-2-46	282,288	4,239,662	2,154	355	07/12/12	RC
PDH-2-47	282,482	4,239,647	2,124	370	07/13/12	RC
PDH-2-48	281,149	4,238,596	2,182	120	09/03/12	RC
PDH-2-49	281,343	4,238,456	2,219	300	09/15/12	RC
PDH-2-50	281,912	4,238,248	2,109	620	07/17/12	RC
PDH-2-51	281,811	4,238,096	2,144	380	07/21/12	RC
PDH-2-52	280,965	4,238,058	2,149	200	09/17/12	RC
PDH-2-53	281,325	4,238,347	2,205	329	08/04/12	PQ
PDH-2-55	281,884	4,238,747	2,195	245	08/06/12	PQ
PDH-2-56	282,488	4,239,033	2,117	329	08/08/12	PQ

Phase 2 - Area 4						
Drill Hole ID	**UTM83-12**			**Depth (Ft)**	**Completion Date**	**Type**
	Easting	**Northing**	**Surface**			
PDH 4-01	281,077	4,234,846	1,977	200	09/20/12	RC
PDH 4-02	281,480	4,234,931	2,005	500	09/20/12	RC
Phase 2 - Monitor Wells and Groundwater Test Bores						
Drill Hole ID	**UTM83-12**			**Depth (Ft)**	**Completion Date**	**Type**
	Easting	**Northing**	**Surface**			
MW-1	277,499	4,238,236	2,122	240	10/06/12	ROT
MW-2	281,004	4,241,119	1,996	200	10/04/12	ROT
MW-3	282,521	4,240,855	1,937	240	10/08/12	ROT
MW-4	279,286	4,237,129	2,064	260	10/05/12	ROT
MW-5	282,223	4,237,288	2,028	260	10/07/12	ROT
MW-6	279406	4238275	2109	320	06/19/13	ROT
MW-9	280385	4239246	2051	60	06/21/13	ROT
MW-10	279859	4240683	2078	260	07/18/13	ROT
MW-11	284897	4242302	1883	280	07/28/13	ROT
MW-13	282770	4243461	1852	280	07/20/13	ROT
TB-03	290260	42491160	1632	980	03/28/13	ROT
TB-03	290184	4247778	1648	1205	04/09/13	ROT
TB-03	291235	4247638	1649	1155	04/16/13	ROT
Phase 3 - Area 1						
Drill Hole ID	**UTM83-12**			**Depth (Ft)**	**Completion Date**	**Type**
	Easting	**Northing**	**Surface**			
A1-03	280,250	4,239,817	2,254	300	02/14/13	RC
A1-04	280,356	4,240,057	2,212	180	02/13/13	RC

TABLE 10.1 PRC DRILLING SUMMARY, CONTINUED

Phase 3 - Area 2						
Drill Hole ID	UTM83-12			Depth (Ft)	Completion Date	Type
	Easting	Northing	Surface			
A2-03	282,020	4,238,906	2,106	300	02/04/13	RC
A2-04	282,038	4,238,748	2,173	440	01/16/13	RC
A2-05	281,845	4,238,454	2,184	850	02/04/13	RC
A2-06	281,640	4,238,369	2,157	800	01/23/13	RC
A2-07	281,700	4,238,134	2,151	560	01/18/13	RC
A2-08	282,701	4,239,495	2,130	590	01/12/13	RC
A2-09	281,377	4,237,969	2,111	180	02/08/13	RC
A2-10	281,242	4,238,225	2,190	300	02/07/13	RC
A2-11	281,134	4,238,332	2,164	200	02/06/13	RC
A2-12	281,378	4,238,601	2,245	250	02/07/13	RC
A2-12	281,378	4,238,601	2,245	600	02/11/13	RC
A2-13	281,064	4,238,730	2,122	260	02/11/13	RC
A2-14	281,266	4,238,848	2,136	300	02/10/13	RC
A2-15	281,603	4,239,062	2,106	300	02/14/13	RC
A2-16A	281,520	4,238,719	2,243	80	01/28/13	RC
A2-16B	281,520	4,238,719	2,243	800	02/01/13	RC
A2-17A	281,673	4,238,724	2,243	680	01/21/13	RC
A2-17B	281,673	4,238,724	2,243	720	01/24/13	RC
A2-18	281,440	4,238,140	2,137	380	01/30/13	RC

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 HISTORICAL WORK

From 1969 through 1974, ESI collected samples from rotary drilling on 10ft intervals. ESI also collected extensive outcrop and trench samples. For drilled samples, the material penetrated (alunite, clay, dolomite, non-alunite) was reported in 10ft increments along with analytical results (data column headings were: % Al_2O_3 by SO_3 determination, % soluble Al_2O_3, % Al_2O_3 by K + Na determination, % K_2O, and % Na_2O). In some drill holes, lab analysis was only performed on samples at every 30 to 50ft or on composite samples from four 10ft intervals. For surface samples, the alumina analysis of the sample was typically plotted by location on a resource plate.

ESI determined both the elemental and mineralogical content of a large number of samples. Some of the mineralogy was done by X-ray diffraction (XRD). The most critical analytical number for ESI was the Al_2O_3 content of the alunite and was determined by three methods simultaneously:

- Indirectly by measuring the SO_3 content through a LECO furnace determination of the sulfur content
- By determining the soluble Al_2O_3 content, presumably by wet chemical methods
- By indirectly determining the Na and K content.

ESI also measured K_2O and Na_2O by an unspecified method. ESI documentation provides results achieved by different techniques and different analytical laboratories. Laboratories listed were ESI, Alumet, HRI, Skyline Labs, and NAS. Though ESI did evaluate their internal analytical testing with outside labs and the results are available in the documents PRC has obtained, there is little information relating to actual sample procedures or quality control methods.

11.2 SAMPLING METHOD AND APPROACH

For PRC's validation drilling program logistics, logging, and initial sample preparation was managed by NAE following recommendations made by Norwest. NAE maintained chain of custody for all samples from the time of collection at the drill sites through initial sample preparation to delivery of samples at the ALS Minerals facility in Winnemucca, NV where they have undergone further preparation for analysis. For PRC's validation drilling program, NAE collected samples on 10ft intervals for core holes and on 5ft intervals for RC holes. Geologic logs have been maintained for all drill holes and include descriptions for lithology, alteration, and recovery. In addition, core logs provide detail on fractures and orientations. Following logging, core was transported to a preparation facility set up by NAE where the core was cut longitudinally into half and quarter-core sections. Core samples submitted for analyses are comprised of 10ft quarter-core sections. Each sample weighs approximately 10 to 11lbs. The remaining half and quarter-core sections are stored in traditional waxed cardboard core boxes, in a secure storage facility in Milford. For RC drilling, samples were collected on 5ft intervals. Cuttings coming up through the central return discharge hose, pass through a cyclone and then through a Jones splitter. The splitter is set to a 50/50 split with one split being retained. Samples were collected continuously at 5ft intervals. Each 5ft sample weighs approximately 18 to 24lbs.

For the second and third drilling programs, adjustments were made to the RC sampling. Sample intervals were changed from 5ft intervals to 10ft intervals based on analysis of analytical variability observed in the validation drilling. Because all RC drilling in the second and third drilling programs were completed using foam injection, adjustments were made to collect between 18 to 24lbs of material directly from the rotary splitter, eliminating use of the Jones splitter.

11.3 Sample Preparation, Analyses and Security

HQ core and RC samples from PRC's validation and infill drilling programs were shipped directly by NAE personnel to the ALS Minerals sample preparation facility in Winnemucca, NV. To date NAE has delivered 944 slim core samples and 4,541 RC samples from the three drilling programs. This includes 335 blind duplicate and 150 reference samples to evaluate analytical precision.

At the ALS Minerals sample preparation facility samples are prepared through the following steps:

- Samples are initially weighed and entered into the ALS tracking system
- Samples are completely crushed to 70% < 2mm
- Samples are then passed through a riffle splitter to create a 1000 gram (g) representative sample
- The 1000g samples are then pulverized to 85% < 75µm
- Prepared samples are then forwarded onto the ALS Minerals laboratory in Vancouver, B.C. for geochemical analysis.

All reject material following splitting is saved and returned to PRC for potential future testing. For the validation drilling program PRC selected two analytical packages to use on all samples. The first package is a whole rock analysis for major oxides using Ion Couple Plasma- Atomic Emission Spectroscopy (ICP-AES) following a lithium metaborate fusion. Under this procedure determinations are made for SiO_2, Al_2O3, Fe_2O_3, CaO, MgO, K_2O, Cr_2O3, TiO_2, MnO, P_2O_5, SrO, BaO, and LOI (loss on ignition). Reporting levels are to 0.01%. The second analytical package is an ICP-AES package for major, minor, and trace elements using a four acid digestion. Determinations in the second analytical package include Al, Ca, Fe, K, Mg, Na, S, and Ti reported to 0.01% levels and Ag, As, Ba, Be, Bi, Cd, Co, Cr, Cu, Ga, La, Mn, Mo, Ni, P, Pb, Sb, Sc, Sr, Th, Tl, U, V, W, and Zn all reported in parts per million (ppm) concentrations.

For the second drilling program completed between July and October of 2012, the ICP-AES whole rock analytical package completed was used on all samples from the HQ core drilling and the RC drilling. The HQ core samples were also tested for minor and trace elements using the four-acid ICP-AES procedure. Both analytical packages were determined by ALS Minerals. ALS Minerals also prepared duplicate pulps of the HQ core samples that were subsequently sent to DCM Science in Denver, CO for mineral analysis using XRD.

For the third drilling program completed in January and February of 2013, the ICP-AES whole rock analytical package was used on all samples. In addition, sulfate determination by carbonate leach and gravimetric analysis was completed on all of the drilling samples plus a selected group of pulp samples from the previous two drilling programs.

Geotechnical logging and testing have been completed on two holes from Area 1, PDHC-1-09 and PDHC-1-11. Geotechnical work was completed by Seegmiller International located in Salt Lake City, Utah. Geotechnical tests completed on samples from the two core holes include: Point Load Testing (axial and diametric), Uniaxial Compression, Elastic Modulus, Direct Shear, and Bulk Density.

Five PQ core holes were completed during the second drilling program on Area 1 and three more were completed on Area 2. The PQ holes were drilled to collect representative material for metallurgical testing and explained in further detail in Section 13 of this report.

12 DATA VERIFICATION

Norwest has conducted numerous site visits to the Blawn Mountain property in support of the drilling and alunite resource characterization. Mr. Kerr first performed a site visit on February 9, 2012 and has made several site visits to the property since that time. Mr. Kerr's last trip to the property was on April 11, 2013. Mr. Henchel has made two site visits to the property, the last visit on January 21 and 22, 2013. The site visits have confirmed the location and access routes of previous and current exploration activities. During the first site visit PRC's validation drilling program was still in progress with both the core and RC rigs operating. Norwest was able to observe drilling, logging and sampling procedures at the drill sites. Norwest also visited and observed the core cutting procedures and sample storage facilities being employed by NAE in Milford. At the time of the first site visit, none of the drill samples had yet been shipped to ALS Minerals for sample preparation and analysis. At the request of Norwest blind duplicate samples of core were added into the sample sequence as one step of quality control.

During subsequent site visits Norwest has been able to observe and confirm both alunite and non-alunite lithologies, alterations, geologic contacts, and observe several of the major structures that bound the alunite deposits. Norwest has maintained an onsite presence throughout the second and third drilling programs ensuring logging, data collection and sampling procedures are being followed in a consistent manner and maintaining a chain of custody.

A search of the SITLA online database confirms the mining leases PRC has with the State of Utah for the Blawn Mountain Project. PRC has valid mineral control through the Exploration/Option Agreement and the two mineral leases for a combined total of 15,403.72ac.

The drill program carried out by PRC in 2011 and 2012 for Area 1 was designed to validate the previous drilling data collected by ESI between 1969 and 1974. All of PRC drill hole locations were twinned to ESI drill holes. Table 12.1 identifies the ESI holes that are twinned by the PRC holes.

TABLE 12.1 PRC VALIDATION DRILLING

PRC Validation Holes	Drill Type - Driller	Twin ESI Drill Hole
BM1	Core - Layne	C159
BM2	Core - Layne	C103 (C207)
BM3	Core - Layne	C12A
BM3A	RC - Gardner	C12A
BM4	Core - Layne	C178
BM5	RC - Layne*	C145
BM6	Core - Layne	C196
BM7	Core - Layne	C11
BM8	RC - Layne*	C197
BM8A	RC - Gardner	C197
BM9	Core - Layne	C125
BM10	Core - Layne	C194
BM11	RC - Layne*	C180
BM11A	Core - Layne	C180
BM12	Core - Layne	C162 (C120)
BM13	Core - Layne	C79
BM14	Core - Layne	C9
BM14B	Core - Layne	C9
BM15	RC - Layne*	C168 (C208)
BM16	Core - Layne	C13
BM16A	RC - Gardner	C13
BM17	RC - Layne*	C170A
BM17A	RC - Gardner	C170A
BM18	Core - Layne	C163
BM19	Core - Layne	C172
BM20	RC - Gardner	C7
BM21	RC - Layne*	C175
BM22	Core - Layne	C164
BM23	Core - Layne	C88
BM24	Core - Layne	C157
BM25	RC - Layne*	C171
BM26	RC - Gardner	C156
BM27	RC - Gardner	C182
BM28	RC - Gardner	C130

* Samples not used due to poor recovery

Norwest has examined and compared the K_2O and Al_2O_3 values from 27 of the Area 1 PRC holes with their respective twin ESI holes. The comparison covers 639 assay intervals or 6,390ft of drilling. On an interval per interval basis there is poor correlation for K_2O and Al_2O_3 concentrations between the two sets of data. However, composite intervals for each hole show that the PRC holes have concentrations that range from 9 to 19.2% higher than the ESI data.

Table 12.2 summarizes the composite values for the twinned intervals. Poor correlation between the two sets of data can be attributed to different drilling methods and most likely different analytical techniques. ESI used conventional rotary drilling methods. Rotary samples tend to be prone to dilution and wall-rock contamination compared to core and RC drilling. Though it is not specified in the ESI documents, K_2O was most likely determined by traditional spectrometry such as atomic absorption or flame photometry versus the ICP-AES analyses completed by ALS Minerals.

TABLE 12.2 COMPOSITE VALUES FOR TWINNED VALIDATION DRILLING

PRC Drill ID	ESI Twin ID	Composite Interval		PRC		ESI	
		From	To	K_2O (%)	Al_2O_3 (%)	K_2O (%)	Al_2O_3 (%)
BM1	C159	10	230	4.84	18.39	4.11	15.62
BM2	C103 (207)	Various		1.91	15.19	2.25	12.60
BM3 (3A)	C12A	30	170	3.65	17.69	3.32	14.29
BM4	C178	90	360	3.30	14.81	2.93	12.18
BM6	C196	Various		2.58	14.28	2.53	12.41
BM7	C11	0	130	3.83	16.38	2.76	11.36
BM8 (8A)	C197	10	780	2.43	16.78	2.70	15.19
BM10	C194	Various		1.75	18.25	0.90	14.90
BM11 (11A)	C180	0	450	3.97	15.32	3.33	12.83
BM12	C162	Various		2.85	13.89	2.74	11.95
BM14 (14B)	C9	0	640	3.87	16.31	3.84	15.53
BM16 (16A)	C13	10	220	1.78	15.02	2.24	14.01
BM17 (17A)	C170A	0	810	3.12	15.04	2.86	12.25
BM18	C163	40	170	2.72	14.51	2.65	12.63
BM19	C172	0	620	5.18	19.79	5.02	18.43
BM20	C7	10	400	3.14	16.05	2.96	14.53
BM22	C164	0	330	3.72	15.81	3.57	14.70
BM24	C157	10	400	1.93	13.23	3.69	17.10
BM26	C156	0	410	3.25	15.32	3.51	14.82
BM27	C182	0	280	3.36	14.81	3.01	13.06
BM28	C130	0	650	5.01	18.89	4.18	15.45

Although there were no targeted twin-hole drilling for Area 2, Norwest has compiled in Table 12.3 comparative average grade data for K_2O and Al_2O_3 values from historic versus current drill holes collared less than 100ft apart. These comparisons suggest that the current versus historic K_2O and Al_2O_3 grade data is similar for Area 2 despite different drilling methods (rotary versus RC) discussed earlier and most likely different analytical techniques.

TABLE 12.3 COMPOSITE VALUES FOR ADJACENT AREA 2 DRILLING

PRC Drill ID	ESI Drill ID	Distance Apart (ft)	Composite Interval		PRC		ESI	
			From	To	K_2O (%)	Al_2O_3 (%)	K_2O (%)	Al_2O_3 (%)
PDH-2-42	B14	30	10	360	2.67	10.63	2.73	12.66
PDH-2-03	B9	100	Various		2.76	12.49	3.96	14.64

A set of 12 sample pulps was forwarded to ACT Labs for comparative analysis (Table 12.4) for the Area 1 twin hole program. For this set of 12 samples there are two sets of analyses from ALS Minerals, original and duplicates, plus the one set of analyses from ACT Labs. ACT Lab analyses compare very closely to ALS Minerals for the 12 samples. Correlation between the two sets of analyses for K_2O and Al_2O_3 exceed 98%.

TABLE 12.4 ANALYTICAL COMPARISON BY LABORATORY FOR AREA 1

Sample ID	Al_2O_3 (%)			K_2O (%)		
	ACT Labs	ALS Original	ALS Duplicate	ACT Labs	ALS Original	ALS Duplicate
949922	19.09	19.10	18.50	4.52	4.37	4.51
949937	23.61	23.90	24.00	6.76	6.74	7.04
949947	10.78	10.95	10.85	3.08	2.96	3.15
949957	22.87	23.20	22.90	6.38	5.84	6.49
949967	22.39	22.30	22.60	6.22	5.55	6.30
949977	19.84	19.90	20.10	5.06	4.61	5.28
949987	14.36	14.50	14.70	2.99	2.87	3.08
949992	14.10	14.15	14.35	3.21	3.10	3.34
949997	16.65	16.60	16.95	4.00	3.88	4.12
978252	16.71	16.55	17.05	4.50	4.36	4.65
978257	14.80	14.95	15.20	2.99	2.96	3.10
978262	15.41	15.30	15.00	3.22	3.11	3.28

A comparison made during the PRC validation drilling program in Area 1 was to evaluate analytical results between core and RC drilling. Two RC holes, BM3A and BM16A, are twinned to two of the core holes, BM3 and BM16. Between the two twinned locations there are 340ft of analyses to compare between the two types of drilling. There is a 75% correlation for K_2O between matched sets data between the core and RC data. Al_2O_3 has a lower correlation of 50%. Core generally returns slightly higher grades for K_2O and Al_2O_3 then drill cuttings for the respective intervals.

PRC incorporates duplicate samples in its analytical sampling program as a check to track analytical precision. To date, 274 duplicate samples have been included in the samples submitted to ALS Minerals. Figure 12.1 compares the K_2O values of duplicates to original samples. There is a strong correlation of values between the duplicate and original samples.

Beginning with the second phase of drilling in 2012, PRC began submitting two reference samples into the stream of samples being submitted to ALS Minerals. Both reference samples were prepared from bulk sample material previously collected for metallurgical testing. Original testing of the two reference samples and preparation of pulps for submittal in the drilling program was completed by ACT Labs in Ontario, Canada. Table 12.5 compares the results from the initial testing of the reference samples to results from the past two phases of drilling.

TABLE 12.5 REFERENCE SAMPLE COMPARISON

Sample ID	SiO_2%	Al_2O_3%	K_2O%			
ACT Labs Reference Sample A	25.58	25.57	7.52			
ACT Labs Reference Sample B	51.01	17.52	4.52			
Sample ID	**Assay Item**	**Number of Samples**	**Minimum**	**Maximum**	**Average**	**Standard Deviation**
ALS Labs Reference Sample A	SiO2%	64	21.30	26.90	23.93	1.35
	Al2O3%	64	24.70	29.80	26.60	0.94
	K2O%	64	7.06	8.45	7.54	0.28
ALS Labs Reference Sample B	SiO2%	49	50.00	54.70	52.88	1.02
	Al2O3%	49	16.75	18.75	17.67	0.44
	K2O%	49	4.22	4.79	4.52	0.12

Reference and duplicate samples show a strong continuity in the dataset without any significant anomalies. Norwest believes the sufficient steps have been taken to validate the analytical data. The authors are of the opinion that the data used in this report adequately depicts the geology and mineral content. The data is sufficient for resource estimation.

Norwest is satisfied with the procedures established by NAE in data collection and sampling. The duplicate samples and comparative analyses returned favourable results that would indicate reliable analyses from ALS Minerals for the validation drilling program. While the ALS Minerals results show higher concentrations than previously indicated in the ESI drilling data, the ALS Minerals analyses confirm the presence of mineralization and indicate grades determined from the ESI drilling data will be conservative estimations.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 INTRODUCTION

Bench- scale comminution, beneficiation, flotation, calcination, leaching, crystallization, and solid/liquid separation tests were performed on composites of drill core and rotary drill cuttings from the exploration drilling program. A large bulk sample collected from Test Pit No. 5 was processed through a pilot plant at the HRI facilities in Golden, Colorado. The testing program was successful and design criteria were established for design of the full scale process facilities. Block Flow Diagrams (BFDs) and description of major unit operations in the proposed methods of processing alunite ore for the production of SOP product are given in Section 17.

Product Specifications

Table 13.1 summarizes the three main parameters: purity, particle size and chloride content that determine SOP product quality.

TABLE 13.1 TYPICAL MARKET PRODUCT GRADES

Standard SOP	Low Chloride SOP	Granular SOP	Soluble SOP
Purity: 50 wt.% K_2O (92.5 wt.% K_2SO_4) Particle Size: 70 to 10 Tyler mesh Chloride Content: < 1.0%	Purity: 51 wt.% K_2O (94 wt.% K_2SO_4) Particle Size: 70 to 10 Tyler mesh Chloride Content: < 0.5%	Purity: 50 wt.% K_2O (92.5 wt.% K_2SO_4) Particle Size: 20 to 6 Tyler mesh Chloride Content: < 1.0%	Purity: 52 wt.% K_2O (96 wt.% K_2SO_4) Particle Size: 150 to 48 Tyler mesh Chloride Content: < 0.5%

13.2 RECENT AND HISTORICAL METALLURGICAL TEST RESULTS

On April 27, 2011, PRC acquired from a third party certain historical information pertaining to the NG alunite property, including data on drilling results, resource estimates, pilot plant testwork, mine plan, and a feasibility study and engineering work performed and/or commissioned by ESI.

In 2011, PRC contracted HRI to provide conceptual metallurgical testing to support a preliminary economic assessment and subsequently in 2012 to perform the metallurgical testing program for the prefeasibility study. The test programs included process development-related bench-scale and pilot scale investigations primarily to recover K_2SO_4 and secondarily Al_2O_3 from alunite. The proposed process flowsheet for the extraction of SOP, H_2SO_4 and alumina, consists of a number of integrated unit operations for processing either ROM whole ore, or alunite concentrate produced by rejecting silica by froth flotation. Based on the results of an extensive metallurgical testing program at HRI, processing ROM ore is being considered as candidate technology.

The following sections describe the extent of metallurgical testing performed to date, which includes size reduction by crushing, grinding and classification, flotation to upgrade the concentration of alunite as feed to downstream processes, drying, calcination, and roasting of whole ore or concentrate to decompose the alunite, leaching the calcine to extract the soluble SOP, solid/liquid separation to recover the brine (filtrate) from the leached slurry, evaporation of brine, and crystallization to recover the product for markets.

Recent Developments in Metallurgical Testing

Since 2011, the historical information acquired on the alunite deposit for recovering alumina as the primary product and SOP as by-product is being supplemented and refined by PRC. These efforts by PRC in support of the Blawn Mountain Project consist of additional exploration, mine planning, comprehensive metallurgical testing, and hydrological and environmental studies, with a focus on recovery of SOP as the primary product, concentrated sulfuric acid as a by-product, and alumina as a potential co-product.

The comminution circuit will be designed for single-stage SAG milling, based on input provided by Outotec, and to meet the design criteria that the feed to the FLSmidth Dryer/Calciner/Roaster system will be P_{80} at 1000 µm (1.2mm maximum).

Concurrent with bench-scale and pilot plant testing at HRI, PRC has enlisted the services of several laboratories, turnkey proprietary technology suppliers, and equipment vendors to assist in process optimization through additional testing of ROM ore samples, pilot plant test residues, and desktop process simulation and modeling studies. Equipment vendors have provided recommendations on selection of equipment, based on their world-wide experience in engineering and construction of similar commercial SOP production facilities.

13.3 Historical Metallurgical Test Results

Results of metallurgical experiments from the 2013 test program at HRI (T. J. Salisbury, June 24, 2013) including testwork subsequent to the June report, from the 2011 and 2012 HRI test program (R. J. Mellon, May 21, 2012) as well as those from 1972 for Earth Sciences Inc., ESI, also by HRI (F. J. Bowen, et al. April 12, 1973) are summarized in the following sections.

13.4 Metallurgical Testing From 2011 through 2013

PRC contracted HRI to perform a process development study to validate and identify process design criteria for the proposed flowsheet consisting of a number of integrated unit operations. For this effort, two sources of ore were selected:

- Bulk material excavated from Test Pit No. 5 located in Area 1. This was the same test pit used by ESI in the 1970's. The first shipment of 2 tons, identified as Bulk 1, was used in experiments conducted by HRI in June 2012.
- An additional 20t of bulk material from Test Pit No. 5, identified as Bulk 2, were received by Hazen in March 2013.
- Also in 2013, eight PQ size drill holes were drilled for the purpose of collecting metallurgical test samples representing the main exploration areas. Core samples identified as PQ were shipped to HRI from the two different regions of the deposit, designated as Area 1 and -2.

The PQ core samples were stage-crushed and master composites labeled Master Composite A (MC-A), Master Composite B (MC-B), and Master Composite C (MC-C) composed of two to three drill cores were prepared for bench scale and pilot plant test work. The large sized PQ core is required for comminution testing including crushing and JK drop weight testing for SAG mill sizing.

HRI has performed confirmatory testwork on bulk samples collected from Test Pit No. 5 and on PQ core composite samples collected from Blawn Mountain. Test Pit No. 5 was the deepest test pit and was located near the center of the envisioned starter pit for the mine. The objective of the test programs were to confirm the results of the original testwork by HRI and to develop new design criteria for process development.

The test programs included the following tests:

- Ore characterization
- Particle size analysis
- Head sample chemical analysis
- Comminution testwork including JK Drop-weight tests, SMC tests, and Bond crushing, rod and ball mill work indices and abrasion indices
- Calcination
- Water leach testing
- Evaporation and crystallization
- Solid-liquid separation
- Alumina processing.

13.5 Material Characterization

Table 13.2 describes Bulk 1-Bulk 2 surface samples and the PQ core samples used by HRI in process development work.

TABLE 13.2 DESCRIPTION OF SAMPLE MATERIALS*

HRI Project	HRI Identification	Short Description	Use
11468	HRI 53201	Bulk 1	Scoping flotation experiments, start-up material for pilot plant
11687	HRI 53456	PQ Core samples	Flotation, comminution, variability tests, pilot plant feed material
11687	MC-A MC-B MC-C HRI 53466-01	39, 40 (Area 1, South) 41, 42, 43 (Area 1 North) 53, 55, 56 (Area 2) Bulk 2	Mai pilot plant feed material

*T. J. Salisbury, June 24, 2013

Table 13.3 provides a summary of how the cores from drill holes were combined to generate the master composites samples. The sampling interval was 8ft linear down the drill hole, which is equivalent to two core boxes per sample. The average core density was 9.4lb/ft.

TABLE 13.3 DESCRIPTIONS OF MASTER COMPOSITES*

Master Composite	Area	PQ Hole	Number of Samples
MC-A	Area 1 (South) Area 1 (South)	PDH1-39 PDH1-40	104 53
MC-B	Area 1 (North) Area 1 (North) Area 1 (North)	PDH1-41 PDH1-42 PDH1-43	59 52 71
MC-C	Area 2 Area 2 Area @	PDH2-53 PDH2-55 PDH2-58	36 25 38

*T. J. Salisbury, June 24, 2013

During 2013, a head sample of each material described in Table 13.2 was characterized by mineralogical studies and chemical and physical assays. Table 13.4 summarizes the results of XRD studies.

TABLE 13.4 RESULTS OF XRD ANALYSIS OF MINERALS*

Sample		Mineral Constituents (Estimate based on peak heights)			
HRI	ID	Major	Subordinate	Minor	Trace
53021	Bulk 1	Alunite	Quartz	Hematite	Kaolinite Muscovite
53466	Bulk 2	Quartz Alunite			Kaolinite Hematite
53456	MC-A	Quartz	Alunite		Kaolinite Hematite
53456	MC-B	Quartz	Alunite		Kaolinite Hematite
53456	MC-C	Quartz	Alunite		Kaolinite Hematite

Each sample was also analyzed using HRI's Bruker S8 Tiger XRF spectrometer (WDXRF) and a sample of Bulk 1 was sent to The Mineral Lab (Golden, CO) for a comparison analysis.

Table 13.5 summarizes the analyses for major elements, typically reported as oxides. Table 13.6 summarizes the trace metal analysis for Bulk 1 sample. The composite samples were not analyzed for trace elements using WDXRF methods.

TABLE 13.5 XRF RESULTS OF ANALYSIS FOR MAJOR ELEMENTS*

Constituent	Analysis, wt.%				
	Bulk 1		MC-A	MC-B	MC-C
	The Mineral Lab	HRI			
Na_2O	0.28	0.15	0.04	0.14	0.30
MgO	<0.05	0.15	0.05	0.11	0.27
Al_2O_3	21.0	21.0	14.1	15.6	15.3
SiO_2	43.4	39.5	58.7	57.4	61.3
P_2O_5	0.28	0.27	0.14	0.14	0.11
S	9.02	NA	NA	NA	NA
SO_3	NA**	19.2	NA	NA	NA
Cl	<0.02	<0.01	NA	NA	NA
K_2O	5.83	5.05	3.34	4.05	3.03
CaO	0.08	0.33	0.09	0.13	0.16
TiO_2	0.59	0.54	0.58	0.48	0.39
MnO_2	<0.01	<0.02	<0.01	<0.01	<0.01
Fe_2O_3	1.32	1.48	2.75	1.62	2.33
BaO	0.07	0.07	NA	NA	NA

*T. J. Salisbury, June 24, 2013.

TABLE 13.6 XRF RESULTS OF ANALYSIS FOR TRACE ELEMENTS*

Constituent	Element, ppm	
	The Mineral Lab	HRI
V	88	100
Cr	124	<100
Co	<10	196
Ni	<10	180
W	<10	NA
Cu	97	280
Zn	<10	60
As	23	<100
Sn	<50	NA
Pb	35	<100
Mo	<10	<100
Sr	706	<100
U	<20	NA
Th	51	60
Nb	12	<100
Zr	155	170
Rb	19	<100
Y	13	30

*T. J. Salisbury, June 24, 2013; **NA = Not Analyzed.

13.6 Chemical Analysis of Master Composites

The first shipment of approximately two tons of Bulk 1 material in June 2012, used in the initial experiments (HRI 53021, Project 11468), was analyzed only for alunite constituents (aluminum, sulfur and potassium) and for silicon as the primary gangue mineral. Table 13.7 summarizes the results of Bulk 1 head assays.

Table 13.7 Bulk1 Head Assays*

Analysis, %			
Al	K	Si	S
10.3	4.7	21.5	7.83

*T. J. Salisbury, June 24, 2013.

Chemical analysis for 28 elements were performed on the Bulk 2 sample and the three master composites using a Perkin Elmer Optima 7300 DV inductively coupled plasma-optical emission spectroscopy (ICP-OES) analyzer. These samples were also analyzed for sulfur content using a LECO carbon-sulfur analyzer. Table 13.8 presents the results of the ICP-OES analyses for the major constituents of interest such as potassium, sulfur, aluminum, sodium, iron, and titanium.

Table 13.8 ICP-OES Chemical Head Assays of Major Constituent Elements*

Sample ID	Analysis, %					
	K	S	Al	Na	Fe	Ti
Bulk 2	4.47	NA	10.2	0.190	0.820	0.284
MC-A	2.74	5.47	6.82	0.388	1.88	0.306
MC-B	3.31	5.47	7.50	0.221	1.13	0.245
MC-C	2.48	4.12	7.25	0.343	1.57	0.201

* Adopted from T. J. Salisbury, June 24, 2013.

To confirm sodium, potassium, iron, and silicon analyses, flame atomic absorption (AA) was used. A second ICP-OES analyzer was used to confirm the aluminum assay results. Table 13.9 summarizes the confirmatory assay results.

Table 13.9 Confirmation Assays*

Sample ID	Analysis, %				
	Flame AA				ICP
	Na	K	Fe	Si	Al
Bulk 2	0.141	4.46	0.790	22.0	10.0
MC-A	0.357	2.95	2.01	27.3	7.32
MC-B	0.184	3.59	1.23	28.5	7.78
MC-C	0.330	2.38	1.68	28.9	7.64

* T. J. Salisbury, June 24, 2013.

A loss on ignition (LOI) test also was conducted at 1000^{o}C (1,830^{o}F). Table 13.10 contains a summary of LECO sulfur and LOI assay results.

TABLE 13.10 LECO SULFUR AND LOI ASSAY RESULTS*

Sample ID	Analysis, %	
	LECO	LOI
	S	at 1000oC
Bulk 2	7.35	23.1
MC-A	5.67	17.6
MC-B	5.66	18.4
MC-C	4.30	16.3

* T. J. Salisbury, June 24, 2013

Table 13.11 summarizes the calculated elemental conversion factors from the chemical assays to determine the alunite content of a given sample using the molecular formula of alunite, $(K_2SO_4.Al_2(SO_4)_34Al(OH)_3$.

TABLE 13.11 CALCULATED ALUNITE CONTENT OF SAMPLES FROM ELEMENTAL ANALYSIS*

Sample ID	Analysis						
	ICPA			LECO	ICPb	Flame AA	Average Alunite
	S	Al	K	S	Al	K	
Bulk 1	NA**	NA	NA	50.5	52.7	49.7	51.0
Bulk 2	NA	51.7	47.4	47.4	51.2	47.3	49.0
MC-A	35.3	34.9	29.0	36.6	37.5	31.2	34.1
MC-B	35.3	38.4	35.1	36.5	39.8	38.0	37.2
MC-C	26.2	37.1	26.3	27.2	39.1	25.2	30.3

*****T. J. Salisbury, June 24, 2013; **NA = Not Analyzed; [a]Building 7 ICP-OES analyzer; and [b]Building 1 ICP-OES analyzer

Table 13.12 summarizes the silica analyses for the five samples.

TABLE 13.12 SILICA ANALYSES*

Sample ID	Analysis,a %SiO2
Bulk 1	46.0
Bulk 2	47.1
MC-A	58.4
MC-B	61.0
MC-C	61.8

*****T. J. Salisbury, June 24, 2013; [a]Flame AA

Table 13.13 is a summary of the average values of angle of repose for various size fractions of the Bulk 2 and MC-A, MC-B, and MC-C composite samples.

TABLE 13.13 AVERAGE ANGLE OF REPOSE MEASUREMENTS OF SIZE FRACTIONS*

Sample		Size Fraction	Average Measurement (°)
HRI	ID		
53466	Bulk 2	¼ in. 6 mesh 10 mesh	30.3 29.3 34.5
53456	MC-A	1.5 in. 0.5 in. 6 mesh	31.8 28.3 34.5
	MC-B	3 in. 1 in. 0.5 in. 6 mesh 10 mesh	43.8 32.3 29.3 34.8 38.8
	MC-C	3 in. 1 in. 0.5 in. 6 mesh 10 mesh	40.8 34.3 32.0 27.5 34.3

13.7 SAMPLE PREPARATION

During 2013, HRI crushed and sampled two shipments of ore:

- On January 30, 2013, HRI received a shipment of 28 pallets (29,000lb) of PQ core boxes from the drilling campaign from the summer of 2012. The large diameter core was drilling specifically for metallurgical testing and represents the main mineralization in Area 1 and Area 2, which are PRC-designated regions at the mine site. These samples were logged in under HRI 53456. The core samples were stage-crushed to different sizes and the samples were used in comminution, flotation, and variability tests.
- On March 6, 2013, 20 one-ton supersacks of ROM material taken from Test Pit No. 5 in Area 1 on Blawn Mountain were delivered to HRI and logged in under HRI 53466. This material was stage-crushed, blended, then split to generate samples used during pilot plant testing.

In 2012, HRI received 4,800lb of ore as part of a 20t sample from Pit No. 5, which was stored in Utah for testing as follows:

- First, set aside 20 pieces of samples 2 x 3in. size for comminution testing.
- Stage-crushed the as received material to passing 1in, cone-and-quartered the crushed material to obtain three representative lots of 200lb.
- Further stage-crushed each of the 200lb lots to obtain 0.75 x 0.50in., minus .50in., and minus 6-mesh lots for studying the effect of particle size in the calcination step.

In 1972, HRI performed tests on a core composite sample, Composite NGC-101, which was prepared by coning, quartering, and splitting 110 bags (2,750lb) of ore from core holes, which varied in depth from 10ft to 400ft. Assay heads were prepared on the as-received and minus 65-mesh composite.

13.8 Comminution Tests And Work Indices

Reduction of particle size in mineral processing is an energy-intensive operation. Work Index relates power consumption in crushing and grinding to the feed and product size distribution. During 2013, HRI completed a comprehensive comminution testing program utilizing the PQ core composite samples, which included the following tests:

- JK drop-weight index (DW_i)
- Abrasion index (A_i)
- Bond ball mill work index (BW_i)
- Bond rod mill work index (RW_i)
- Bond impact work index (CW_i)
- Semi-autogenous grinding (SAG) Mill Comminution (SMC) testing.

The JK drop-weight test measures the breakage parameters of a rock sample, which are required to analyze or predict SAG mill performance. Table 13.14 summarizes the drop-weight breakage evaluations.

Table 13.14 Drop-Weight Breakage Evaluations*

Parameter	MC-A	MC-B	MC-C
Specific gravity	2.54	2.48	2.36
A - Maximum Breakage	66.3	64.1	63.8
b – Relation between energy and impact breakage	2.65	1.41	2.31
A x b = Overall AG-SAG hardness	175.7	90.4	147.4
T_a = Abrasion parameter	1.36	0.77	0.84
Resistance to Impact breakage	Very Soft	Soft	Very Soft
Resistance to Abrasion Breakage	Very Soft	Very Soft	Very Soft

*T. J. Salisbury, June 24, 2013; AG = Autogenous Grinding

The Abrasion Index is used to determine steel media and liner wear in the comminution circuit. Table 13.15 provides a summary of test results of abrasion index and work indexes.

TABLE 13.15 SUMMARY OF AI, BWI, RWI, AND CWI RESULTS*

PRC ID	Ai, g	BWi, kWh/t	RWi, kWh/t	CWi, kWh/t
MC-A	0.4057	12.0	10.6	10.3
MC-B	0.4132	14.7	13.3	10.0
MC-C	0.4838	14.5	10.9	10.2

*T. J. Salisbury, June 24, 2013.

Table 13.16 is a summary of the Semi-autogenous grinding Mill Comminution test results.

TABLE 13.16 SUMMARY OF SMC TEST RESULTS*

Parameter	MC-A	MC-B	MC-C
Specific gravity	2.60	2.49	2.32
A - Maximum Breakage	79.8	70.9	75.4
b – Relation between energy and impact breakage	1.07	1.25	1.64
A x b = Overall AG-SAG hardness	85.4	88.6	123.7
DW_i = drop-weight index, kWh/m^3	3.04	2.8	1.88
DW_i %	17	15	8
M_{ia} = Coarse particle component, kWh/t	10.7	10.5	8.2
M_{ih} = High-pressure grinding roll (HPGR) component, kWh/t	6.8	6.5	4.7
M_{ic} = Crusher component, kWh/t	3.5	3.4	2.4
T_a = Low-energy abrasion component of breakage	0.85	0.92	1.38

*T. J. Salisbury, June 24, 2013; AG = Autogenous Grinding

Comminution Testing - 2012

Table 13.17 presents the results of tests performed in 2012 by HRI (D. W. Gillespie, March 1, 2012) on the bulk sample from Test Pit No. 5, which was assigned Hazen number 53021, to evaluate BWi, and CWi in kilowatt hours per ton (kWh/t), and Ai in grams (g).

TABLE 13.17 WORK INDICES BWI, CWI AND AI FROM HRI TEST RESULTS*

Parameter	BWi, kWh/t	CWi, kWh/t	Ai, g
HRI Sample # 53021	5.9	7.06	0.2391

*D. W. Gillespie, March 1, 2012

- Twenty pieces of ore ranging from 2 to 3in set aside were used to determine CW_i.
- Two kilograms of the .75 x .50in material saved were used to determine the A_i.
- Twelve kilograms of the 6-mesh material was used to determine BW_i.

13.9 Mineralogical Analysis

Historical Mineralogical Analyses – 1970s

HRI performed mineralogical examinations on a series of screen fractions (from plus 4-mesh to minus 400-mesh) of Composite NGC 101 core (F. J. Bowen, et al. April 12, 1973). The following is a summary of the findings:

- The particle size of alunite varies from coarse-grained to fine-grained (from 200 x 50µm to 50 x 10µm) aggregates in a matrix of microcrystalline quartz.
- Coarse alunite aggregates contain inclusions of about 10 micron rounded quartz particles and alunite, when fine-grained, is intimately intergrown with quartz.
- Accessory minerals are iron oxide (hematite), leucoxene (fine-grained altered titanium mineral, rutile or TiO_2), zircon (zirconium silicate), and microcline ($KAlSi_3O_8$) or potassium-rich feldspar.
- Voids up to about 1mm diameter and filled with opal-like substance with minor amounts of calcite ($CaCO_3$) on microfractures.
- XRD studies indicate the absence of aluminosilicates such as clay.
- Microscopic examination of calcines obtained from 750°C to 900°C roasting step and their water-leach residues shows:
  - An amorphous, intimate mixture of alumina and dehydrated alum phase.
  - Optical evidence suggests that leachable K_2SO_4 might be entrapped in this phase.
  - The presence of amorphous alumina on the exterior of the alum-alumina particles could prevent diffusion of soluble K_2SO_4 from the interior of the mixture.

Mineralogical Analyses - 2013

Table 13.18 summarizes the mineral composition of the Bulk 2 Composite and Master Composite A, B, and C samples.

Table 13.18 Mineral Composition of the Composite Samples

Minerals	Bulk 2 Composite	Master Composite A	Master Composite B	Master Composite C
Alunite	47.1	40.8	31.7	25.8
Quartz	49.5	52.9	58.3	62.4
Hematite	1.15	3.84	2.21	1.98
Kaolinite (Clay)	0.69	0.21	2.65	3.89
Sulfide minerals	0.09	0.24	0.10	0.11
Jarosite	0.00	0.51	0.07	0.01
Rutile/Anatase	0.69	0.80	0.76	0.54
Micas	0.19	0.20	2.04	2.75
Others*	0.59	0.49	2.33	2.45
Total	**100**	**100**	**100**	**100**

Notes: *Others includes Feldspars, Fe-Al silicates, Pyroxene, Calcite, Chalcopyrite, Pyrite, and trace amounts of other minerals.

13.10 Chemical Analysis 2012

In January 2012, HRI received 4,800lb of alunite ore from Pit No. 5, which was a collection of various sizes of rocks taken from 24 super sacs (R. J. Mellon, May 21, 2012). The sample was characterized by performing chemical and physical analyses, mineralogy, and thermal analyses.

Results of analyses of two pulverized head splits for the major constituents of the ore are:

- 10.6wt.% Al (aluminum)
- 4.6wt.% K (potassium)
- 7.6wt.% S (sulfur)
- 20.5wt.% Si (silicon).

The remaining components were each <1.0 wt.%.

The XRD analyses of the head splits confirmed the major components of the ore as quartz and two forms of alunite:

- Quartz (SiO_2) and
- Two forms of alunite:
  - $K_{0.805}Na_{0.0132}(H_2O)_{0.063}Al_3(SO_4)_2(OH)_6$ and
  - $KAl_3(SO_4)_2(OH)_6$.

Analytical results of 2012 of 400g of the 6-mesh material, pulverized to produce two head samples (Sample A and Sample B) are given in Table 13.19.

Table 13.19 Ore Analyses*

Analyte	Analysis, wt.%		
	Sample A	Sample B	Sample B (Repeat)
Al	**10.5**	**10.5**	**10.8**
Fe	0.87	0.90	0.89
K	**4.51**	**4.65**	**4.62**
Mg	0.018	0.018	0.018
Na	0.16	0.17	0.16
Si	**21.6**	**19.6**	**20.2**
S	7.64	**NA	**NA
Ti	0.30	0.31	0.31

* R. J. Mellon, May 21, 2012; **NA = not available

HRI also determined particle size distribution (PSD) for the 1in material and fractions were submitted for x-ray fluorescence semi-quantitative (XRF-SQX) analysis to determine the analyte distribution within a given size fraction for the purpose of selecting an enriched fraction after crushing.

The XRF-SQX analysis is particularly suited for a high silica sample which is difficult to dissolve for Inductively Coupled Plasma Mass Spectrometry (ICPMS). XRF is a more economical test for samples where composition down to 0.01% (100ppm) is desired, the major advantages being the sample does not need to be put into solution as in ICPMS and no standards are required for semi-quantitative (SQX) analysis.

Table 13.20 and Table 13.21 summarize distributions of the major and minor species within each size fraction. The potassium and aluminum contents show no trend (within analytical precision) among the particles sizes.

TABLE 13.20 XRF RESULTS OF MAJOR CONSTITUENTS BY PARTICLE SIZE DISTRIBUTION*

Analyte	Analysis, wt.%					
	¾ x ½ in.	½ in. x 6-mesh	6 x 20-mesh	20 x 35-mesh	35 x 65-mesh	Minus 65-mesh
Al_2O_3	**19.7**	**18.7**	**19.3**	**18.5**	**19.8**	19.7
BaO	0.07	0.07	0.07	0.07	0.07	0.07
CaO	0.05	0.06	0.07	0.22	0.22	0.24
Cl	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02
Fe_2O_3	1.08	1.08	1.24	1.44	1.58	1.54
K_2O	**5.65**	**5.42**	**5.58**	**5.50**	**5.86**	**5.90**
MgO	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05
MnO_2	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01
Na_2O	0.23	0.22	0.22	0.22	0.23	0.22
P_2O_5	0.25	0.26	0.28	0.26	0.25	0.25
S	**8.22**	**7.96**	**8.25**	**7.93**	**8.33**	**8.30**
SiO_2	**42.2**	**44.7**	**43.8**	**45.8**	**41.5**	**39.8**
TiO_2	0.57	0.54	0.56	0.60	0.52	0.50
Distribution, wt.%	37.1	30.1	13.8	6.8	7.4	4.839.8

*R. J. Mellon, May 21, 2012

TABLE 13.21 XRF RESULTS OF MINOR CONSTITUENTS BY PARTICLE SIZE DISTRIBUTION*

Analyte	Analysis, wt.%					
	¾ x ½ in.	½ in. x 6-mesh	6 x 20-mesh	20 x 35-mesh	35 x 65-mesh	Minus 65-mesh
As	28	27	25	25	37	43
Co	<10	<10	<10	<10	<10	12
Cr	106	82	89	110	80	91
Cu	23	19	41	31	40	59
Mo	<10	<10	<10	10	14	26
Ni	<10	<10	<10	<10	<10	<10
Pb	21	31	39	38	32	35
Sn	<50	<50	<50	<50	<50	<50
Sr	691	655	697	696	662	705
U	<20	<20	<20	<20	<20	<20
W	<10	<10	<10	<10	<10	<10
Y	91	81	88	93	91	97
Zn	<10	<10	<10	<10	<10	<10

*R. J. Mellon, May 21, 2012

13.11 THERMAL ANALYSIS

HRI performed (R. J. Mellon, May 21, 2012) Thermal Gravimetric Analysis (TGA) to understand the behavior of the ore as it is heated. TGA uses a highly sensitive balance to monitor and measure the weight loss of a sample of material as a function of temperature and time, in a controlled atmosphere.

In this analysis, a sample of ore placed in the instrument was heated to 1000°C at a rate of 10°C/min and held at 1000°C. Time and weight loss with temperature, and temperature, weight loss, and rate of weight change were determined. The results indicate:

- Initially, the weight loss commenced at 505°C and produced a weight loss of 5.6% of the sample.
- The weight loss slowed between 560 and 720°C, at which point the total loss was 14%.
- An additional 8.3% weight loss began at 735°C and continued to 780°C.
- The final weight loss at the end of analysis was 25%.
- The first weight loss occurred in a typical range when hydroxyls decompose and water is evolved.
- The weight loss at 735°C is associated with the generation of sulfur oxides, which was confirmed through the SO_2 analyzer during the batch kiln runs.

According to HRI, the TGA confirmed that the proposed lower calcining temperature of 800°C would be sufficient to drive off the water and decompose the alunite.

During drying tests performed in 1973 by HRI, the as-received ore was found to be moist and became air-dried during sample preparation. The moisture content of five 20lb bag samples averaged 0.5% H_2O when dried at 100°C before stage-crushing to 65-mesh.

Table 13.22 summarizes observed weight loss during drying tests.

TABLE 13.22 SAMPLE DRYING TEST RESULTS*

Drying Temperature, °C	Weight Loss, %
100	0.9
200	1.2
400	1.2
600	8.0
800	18.5

* F. J. Bowen, et al. April 12, 1973.

The drying tests indicated:

- Interstitial water was lost during drying through 400°C.

- At 600°C, both interstitial and combined water were lost along with small quantities of sulfur in off-gases.
- At 800°C, sulfur loss in off-gases was significant.

Microscopic examination of calcines obtained from 750°C to 900°C roasting step and their water-leach residues shows:

- An amorphous, intimate mixture of alumina and dehydrated alum phase.
- Optical evidence suggests that leachable K_2SO_4 might be entrapped in this phase.
- The presence of amorphous alumina on the exterior of the alum-alumina particles could prevent diffusion of soluble K_2SO_4 from the interior of the mixture.

13.12 CALCINATION EXPERIMENTS

During 2013, in addition to flotation concentrate, HRI evaluated whole ore as an alternative feed material to the roasters in a parallel path of process development.

HRI used the flotation concentrate generated from bench-scale and pilot plant as feed in the roasting experiments to convert the potassium and sulfate values in the alunite mineral into a leachable form as K_2SO_4. Previous investigations have shown that high-temperature roasting at greater than 1472°F (800°C) produced calcine with potassium extraction ranging from 67 to 87% during leaching with water. However, high temperature roasting results in changing the crystal structure of Al_2O_3 to the alpha form, which is not amenable to leaching with NaOH.

A benchmark of successful roasting experiment is significant extraction of the SOP into the leachate along with the formation of gamma- Al_2O_3, which is amenable to extraction from the leach residue by a caustic (NaOH) leach. The goals of the roasting process, therefore, were as follows:

- Roast the alunite ore at temperature less than 1292°F (700 °C).
- Produce a calcine material that can be leached to a target 90% K extraction.
- Produce the gamma form of alumina that can be leached with KOH.

HRI used the 4in diameter quartz batch kiln to minimize the amount of feed needed for roasting and still generating sufficient amount of calcine for leaching. A total of 38 batch kiln campaigns were completed using both single-stage and three stage roasting flowsheets.

The results of the calcining and water leach testing of the calcine at particle sizes of 80% passing 80 and 1,000 microns are presented in Table 13.23.

TABLE 13.23 RESULTS OF ROASTING AND WATER LEACHING TESTS

Roast ID	Roast Conditions						Results			
	Type	Particle Size P_{80}, µm	Temp, °C	Gas	Type	Mass, g	Parameter	Al	K	Leachable K[a]
BK-28	Whole Ore	<80	575/575/575	Air/Natural Gas/Air	4 in	400.2	Extraction, %	0	50	85
							Accountability, %	110	108	91
BK-28	Whole Ore	<80	575/575/575	Air/Natural Gas/Air	N/A	N/A	Extraction, %	0	59	NC
							Accountability, %	113	103	NC
BK-29	Whole Ore	<1500	575/575/575	Air/Natural Gas/Air	None	None	Extraction, %	0	68	NC
							Accountability, %	97	100	NC
BK-30	Whole Ore	<1000	575/575/575	Air/Natural Gas/Air	None	None	Extraction, %	0	76	NC
							Accountability, %	96	108	NC
BK-31	Whole Ore	<500	575/575/575	Air/Natural Gas/Air	None	None	Extraction, %	0	71	NC
							Accountability, %	97	102	NC
BK-32	Whole Ore	<300	575/575/575	Air/Natural Gas/Air	None	None	Extraction, %	0	70	NC
							Accountability, %	101	105	NC
BK-33	Whole Ore	<150	575/575/575	Air/Natural Gas/Air	None	None	Extraction, %	0	67	NC
							Accountability, %	97	104	NC
BK-30	Whole Ore	<1000	575/575/575	Air/Natural Gas/Air	None	None	Extraction, %	0	71	NC
							Accountability, %	91	101	NC
BK-34	Whole Ore	<1000	575/575/575	Air/Natural Gas/Air	None	None	Extraction, %	0	90	100
							Accountability, %	100	104	100
BK-35	Whole Ore	<1000	575/575/575	Air/Natural Gas/Air	None	None	Extraction, %	0	54	92
							Accountability, %	97	103	95
BK-36	Whole Ore	<1000	575/575/575	Air/Natural Gas/Air	None	None	Extraction, %	0	71	NC
							Accountability, %	102	104	NC
BK-1A	Whole Ore	1 mm (crush)	575	N_2/Natural Gas	None	None	Extraction, %	0	85	NC
							Accountability, %	100	106	NC
BK-2A	Whole Ore	<0	575	N_2/Natural Gas	None	None	Extraction, %	0	69	NC
							Accountability, %	97	106	NC

During single-stage roasting at the low temperature of less than 1292°F (700°C), the off-gas composition was primarily a dense white phase with low (ppm) SO_2 concentration. The white phase was assumed to be SO_3, which was not detected by the array of gas analyzers, but was collected in the ESP. When powdered sulfur was added to the kiln, the concentration of SO_2 in the off-gas was outside the 0 to 25% analyzer range.

Potassium extraction was marginal in the initial water leaching experiments of calcine from single-stage roast tests BK-1 and BK-13 conducted in 2013. Based on literature reviews, HRI identified an alternative method of roasting the alunite in a three-stage process. The first stage of roasting, or dehydroxylation, removes the hydroxyl (OH^-) groups associated with the alunite minerals.

After the dehydroxylation, the calcine was cooled and elemental sulfur was added to the calcine to reduce the sulfate (SO_4^{2-}) groups and convert the potassium into a leachable form while maintaining the alumina in a non-alpha phase. This reaction is given by the equation:

$$4KAl(SO_4)_2 + 3S \Rightarrow 2K_2SO_4 + 2Al_2O_3 + 9SO_2(g)$$

When the reduction reaction is complete, the excess sulfur is removed as SO_2 in an oxidation stage by the reaction:

$$S + O_2 \Rightarrow SO_2$$

During the 3rd quarter of 2013, FLSmidth Minerals Pyrometallurgy group in Bethlehem, PA was contracted to evaluate the most energy-efficient method of drying, calcining and roasting either alunite flotation concentrate or whole ore filter cake with an estimated moisture content of 10wt.% and P_{80} of 1000μm. FLSmidth, based on desktop simulation studies and accrued information from similar projects, identified a Dryer/Calciner/Roaster system consisting of the following major components:

- <u>Flash Dryer Circuit</u>: The flash dryer portion of the system supports removal of the free moisture present in the alunite cake using a combination of flash calciner off gases and additional hot gas provided by a natural gas fired air heater.
- <u>Flash Calciner Circuit</u>: The dried material from flash dryer is delivered to flash calciner for further heating to remove all the chemically bounded water and to decompose the alunite to potassium and aluminum sulfates. The material produced is transfer to a flash roaster so that it serves to complete the aluminum sulfate decomposition.
- <u>Flash Roaster Circuit</u>: The pre calcined material from the flash calciner is delivered to the flash roaster for further heating and to decompose all aluminum sulfate to aluminum oxide and SO_3. It also serves to convert most of the generated SO_3 to SO_2 at reduced conditions.
- <u>Flash Roaster Off Gas Circuit</u>: Air-to-gas heat exchanges cool the process gas leaving the flash roaster cyclone to a temperature level suitable for the downstream ESP. The ESP removes non-condensable particulate from the final system process off gas stream and an induced fan delivers excess gases to acid plant after some amount of gases are recycled back to the flash roaster.
- <u>Cyclone Cooling Circuit</u>: Four cyclones operating in series transfer heat from the flash roaster circuit product to the incoming ambient air stream.
- <u>Excess Air from Cooling Circuit</u>: The excess air from the cyclone cooling circuit is delivered to a boiler to generate steam and cool down the gases to a temperature that is suitable for handling in the downstream bag house and induced draft fan.

Preliminary process design criteria for the Dryer/Calciner/Roaster circuit, developed by FLSmidth and based on simulation and modeling of available information, are as follows:

- Solids content of Filter Cake as feed to Dryer = 90%
- Moisture content of Filter Cake as feed to Dryer = 10%
- Number of thermal processing units = 4
- Name plate capacity of each thermal processing unit = 330tph
- Alunite decomposition reactions are carried out at 1022°F (550°C).
- The maximum temperature in the Calciner shall not to exceed 1112°F (600°C).
- The residence time in the Calciner is about two seconds.

Roasting Studies – 2012

HRI conducted 18 calcination experiments (R. J. Mellon, May 21, 2012) in a batch quartz kiln system. The kiln, 44in. overall length, consisted of a 4in. diameter by 18in. long quartz reactor section, four rows of lifters pressed into the sidewall at 90° intervals for mixing the charge, a direct current motor for controlling the rotation of the kiln between 1 and 5rpm, a Type K thermocouple that resided within the bed material, a programmable temperature controller that varied the furnace electrical input for a given zone, and a gas handling train:

- Typically, the kiln loaded with 1 to 2kg of alunite ore was placed inside an electrically-heated furnace and the thermocouple positioned into the material.
- In the first group of 12 experiments, the ore was ground to minus 0.50in. and the retention time (30, 60, 90, and 120 minutes) and temperature (800, 850, and 900°C) were varied to study their effects on extraction of potassium and aluminum during leaching with water.
- In the remaining six kiln experiments, three were performed on the ore crushed to minus 6-mesh and three on the ore sized to 0.75 by 0.50in. to study the effect of ore particle size on potassium extraction.
- In all experiments, the O_2 concentration in the exhaust gas consistently increased or decreased as the concentration of SO_2 increased or decreased.
- Peak concentration of SO_2 was as high as 31vol.% (dry basis), but at the end of the experiment typically was less than 2 vol.% SO_2. It is inferred that the concentration of O_2 at 21vol.% in off-gases during the air purge of the kiln resulted in the observed peak concentration of SO_2.

The above data suggest that in all probability sulfur trioxide (SO_3) evolved during calcination of alunite is partially decomposed in the hot kiln to SO_2 and O_2 according to the following reaction:

$$SO_3 \Leftrightarrow SO_2 + \tfrac{1}{2} O_2$$

Off-gases from the kiln were cooled downstream from the furnace to remove water vapor. Downstream from the condenser, a slipstream of gas was directed to CO_2, CO, O_2, and SO_2 analyzers. Downstream of this equipment, gases passed through bubblers containing NaOH solution to remove SO_2 evolved from heating the ore. Gas analyses and thermocouple readings were continuously recorded to a digital file on a computer.

Roasting Studies – 1970's

In July 1973, HRI performed roasting studies (F. J. Bowen, April 12, 1973) on Composite NGC-101 ore in a rotary Vycor Retort and in a 6in diameter Screw Reactor to:

- Determine the solubility of potassium and sulfate in the calcine produced at approximately 600°C (low) and at approximately 800°C (high) calcining temperatures.
- Produce calcine for investigating alumina solubility and/or flotation studies.

Only high-temperature roasting tests were conducted in the Vycor Retort using 200 and 300g charges. Evaluation of calcine quality was, therefore, based on the solubility of potassium and sulfate during leaching with water.

The charge was placed in the electric furnace and the temperature was raised and held at the desired level. The retort was rotated at about 1rpm and a stream or air or nitrogen was used to purge the evolved gases from the retort. Temperature measurements were taken inside the retort just above the ore bed and recorded continuously. Off-gases were analyzed to determine SO_2, SO_3, and O_2 content. At the end of the roasting cycle, the furnace was shut off and the calcine was allowed to cool.

After cooling, the calcine was weighed to determine the weight loss due to roasting, a head calcine sample was split, and all or a portion of the remaining sample was sent to leach for evaluation of roasting efficiency based on the solubility of potassium and sulfate.

The Screw Reactor was an externally (gas) heated, 6in diameter and 110in long unit equipped with a variable speed screw for moving the ore at a rate of 5 to 10lb per hour along the reactor. The externally heated roasting zone measured about 50in length and the variable residence time was from 30 to 60 minutes. Temperature was measured at four points along the roasting zone and recorded.

The calcine leaving the roasting zone passed through a cooling zone and was discharged to a container. During roasting, air or nitrogen was circulated co-current or countercurrent to the flow of the ore. The off gases leaving the reactor were analyzed to determine SO_2, SO_3, and O_2 content.

Previous roasting studies (F. J. Bowen, April 12, 1973) during the months of March through May of 1972 at HRI have shown that:

- Relatively coarse ore can be roasted at about 600°C to produce a calcine in which the potassium and sulfate values are converted to a soluble form suitable for extraction in aqueous solution containing ammonia, NaOH, or KOH and thus separating them from aluminum oxide and silica.
- Roasting at 800 to 900°C eliminates the sulfate which is combined with the aluminum in the ore as volatile SO_2 and SO_3 and leaves the potassium sulfate as a water-soluble compound which can be leached from the alumina (Al_2O_3) and silica (SiO_2).

TABLE 13.24 OFF-GAS ANALYSES FROM 6-INCH SCREW REACTOR ROASTING TESTS*

Parameter	Roasting Temperature, °C				
	500°C	550°C	600°C	800°C	850°C
Test No.:	**SR-9**	**SR-10**	**SR-11**	**SR-6**	**SR-13**
Feed:					
lb/hr.	7.3	7.3	7.3	5.3	5.2
Residence time, min.	45	45	45	30	60
Size, mesh	65	65	65	65	8
Purge Air, cfm	0.6	0.6	0.6	0.6	0.6
Off-Gas Analysis:					
SO_2, %	0.	0	0	16.1	5.5
SO_3, %	1.1	0.6	1.1	5.0	1.0
O_2, %	18.4	18.0	17.4	15.1	15.6
% K Dissolution	23.5	50.5	92.5	92.3	86.6
% SO_4 in Calcine**	21.2	23.4	21.1	12.3	12.5

*F. J. Bowen, et al. April 12, 1973; **SO_4 in raw ore = 20.5%.

13.13 LEACHING POTASSIUM FROM THE ORE

Historical SOP Leach Studies - 1970s

In 1973, Hazen conducted water leach tests at 25°C on calcines produced at 800°C in a rotary Vycor Retort by agitating the slurry containing 50 wt.% and 17 wt.% solids for one hour to dissolve the potassium sulfate. The leaching cycle consisted of leaching with water as the lixiviant followed by filtration and washing the filter cake with water. The water leach tests showed that:

- About 94% of the potassium and 88% of the sulfate in the calcine can be dissolved in water under the best roasting conditions. The solubility of alumina was 3 or 4%. The leach residue assayed about 0.3%K and 1.5%SO_4.

- As much as 2.5 percentage points more of potassium sulfate was dissolved at 85°C than was dissolved at 25°C.
- No significant difference was seen in the leachability of calcine produced from 3-, 8-, 28-and 65-mesh materials.

Recent Leach Studies – 2012

During 2012, extraction of potassium sulfate was investigated by HRI using a two-step approach consisting of calcining the ore followed by leaching the calcine with water. The ore was calcined in a kiln under various temperature and residence time conditions to determine the effects of these variables on extraction of potassium in the subsequent leach step (Mellon, Robert J. May 21, 2012). The proposed calcination reaction at 1472 to 1652°F (800 to 900°C) is as follows:

$$2KAl_3(SO_4)_2(OH)_6 \rightarrow K_2SO_4.3Al_2O_3 + 3SO_3 + 6H_2O$$

Water and sulfur (as SO_3) are driven off during calcination and water-leachable K_2SO_4 is left behind. The aluminum is converted into an oxide that can be recovered in subsequent processing.

HRI performed eighteen calcination experiments in a batch quartz kiln system in which the retention time (30, 60, and 90 minutes) and temperature (800, 850 and 900°C) were varied. Approximately 200g of sample from each of the calcines produced in the batch kiln were leached with water. The leach equipment consisted of a 2L kettle, a condenser, a heating mantle, a variable speed mixer, a thermometer, a vacuum flask, a Buchner funnel with Whatman paper (#5 or #50), and sample bottles.

The test conditions and results for three sized samples calcined and leached with water in 2012 are summarized in Table 13.25.

TABLE 13.25 SUMMARY OF CALCINING AND LEACHING CONDITIONS AND RESULTS*

Conditions	¾ by ½ in.	Minus ½ in.	Minus 6-mesh
Calcine temperature, °C	850	900	900
Residence time, minutes	60	60	90
Potassium extraction, %	87	86	85
Aluminum extraction, %	1.3	0.4	0.2
Sulfur evolution, %	63	70	70

*R. J. Mellon, May 21, 2012

HRI evaluated the effectiveness of calcination by evaluating the recovery of potassium during leaching. Potassium recovery was calculated by analyzing the primary filtrate and wash solutions. A mass balance was performed for each analyte around the calcination and leach operations and the closure of the balance is the "mass accountability," which is labeled as "Bal" in Table 13.26.

Analytical precision and unavoidable material losses can cause deviations from 100% accountability. Table 13.26 summarizes the results of eighteen (18) batch kiln calcination and calcine leach experiments.

TABLE 13.26 OF RESULTS FROM BATCH KILN CALCINATION TESTS*[A]

Expt. No.	Calcination			Calcine, %				Leach, %	
	Ore Size	Time, min	Temp. °C	Potassium		Aluminum		K	Al
				Loss[B]	Bal[C]	Loss[B]	Bal[C]	Ext[D]	Ext[D]
1	-1/2 in.	30	800	0.1	93.4	0.01	92.7	82.7	9.3
2	-1/2 in.	30	850	0.1	97.1	0.02	91.4	84.2	7.2
3	-1/2 in.	60	800	0.9	93.1	<0.01	93.0	85.3	2.7
4	-1/2 in.	90	800	1.3	103	0.01	92.1	82.6	2.1
5	-1/2 in.	120	800	1.4	99.6	0.01	88.8	81.3	1.2
6	-1/2 in.	60	850	1.4	99.3	<0.01	102	82.4	0.6
7	-1/2 in.	90	850	1.4	97.7	<0.01	94.0	81.0	0.3
8	-1/2 in.	120	850	1.3	83.5	<0.01	94.8	86.8	0.2
9	-1/2 in.	60	900	1.3	94.3	<0.01	99.4	85.8	0.4
10	-1/2 in.	30	900	1.4	96.5	<0.01	94.8	84.9	0.4
11	-1/2 in.	120	900	1.3	89.1	<0.01	91.7	82.7	<0.01
12	-1/2 in.	90	900	1.3	93.6	<0.01	93.5	83.6	<0.01
13	¾ x ½ in.	30	800	0.5	77.5	0.01	74.7	67.4	1.9
14	- 6 mesh	30	800	0.7	98.6	0.2	101	81.8	12.4
15	- 6 mesh	60	850	0.8	100	0.3	104	83.3	1.1
16	- 6 mesh	90	900	0.7	91.8	0.01	92.3	85.0	0.2
17	¾ x ½ in.	60	850	0.5	101	0.01	102	86.6	1.3
18	¾ x ½ in.	90	900	0.5	99.9	0.01	98.6	86.3	0.2

Notes: Mellon, Robert J. (May 21, 2012)

All leaches were conducted at 90°C; 60 min. residence time; 20% solids; agitated.

When evaluating the water leach test results, both potassium and aluminum extractions must be taken into consideration. An inspection of test results shows:

- Calcine produced at 850°C resulted in one of the highest potassium extractions of 87% and aluminum extraction of 1.3%.
- Calcining the ore at 900°C typically resulted in comparable potassium extraction in the 85 to 86% range with the added benefit of lower aluminum extraction of 0.2 to 0.4%.
- Processing the ore at relatively high temperature (900°C) with resulting lower aluminum extraction is beneficial in the recovery of purified K_2SO_4 in downstream unit operations.

Recent Leach Testing 2013

During 2013, HRI completed a total of 22 leaching experiments with calcine generated from the roasting process to determine the staged roasting concept, establish operating conditions for the leaching stage, and produce brine or leach liquors for the crystallization experiments.

During 2013, a series of water leach tests were completed on calcines generated using reductants such as sulfur, carbon monoxide/carbon dioxide, hydrogen sulfide, and natural gas.

- The results of the most recent tests run at 80 and 1000um are presented in Table 13.23. Sulfur and natural gas were found to give the best overall calcine-leach results.
- The results were used to calculate the extraction of “extractable” potassium, which is associated with the alunite. Some potassium has been identified to be associated with minerals other than alunite, such as K-feldspar (Orthoclase) and kaolinite clays.
- It is estimated that more than 90% of the potassium associated with alunite can be extracted by leaching the calcine with water.
- Natural gas has been identified as the reductant of choice because of its performance over sulfur.
- A series of seven calcine leach tests were performed using Master Composite B to evaluate the effect of particle size of the feed on calcine-leach performance. The tests were performed with natural gas as the reductant. The feed size varied between 100 and 1500 µm in five batch kiln tests, followed by leaching the calcine with water.
- The results of the seven tests were used to select the optimal feed size and a single test was then run on Master Composite A and Master Composite C to confirm the results.
- Preliminary test results indicated that potassium can be leached from calcine feed with a particle size less than 80% passing 1.0mm without a reduction in leach recovery.
- Master Composite B and Master Composite C have a higher percentage of potassium and aluminum associated with non-alunite minerals, including K-feldspar and kaolinite clays than master Composite A.
- During September 2013, HRI completed two tests on Master Composite A and Master Composite B without the oxidation step and leaching the calcine with water at 35wt.% solids. The results achieved are similar to those in the previous experiments.

Leaching Fresh Calcine with Recycled Leach Liquor

Process design for the commercial plant envisions recycling the Filtrate to leaching to build up the concentration of the Filtrate pumped to the Evaporator/Crystallizer circuit, which is an effective method of conserving both energy and water. Additionally, the wash water from Filter Cake wash cycle also will be re-used for quenching the calcine and/or in the Leach circuit.

HRI conducted an experiment in 2012 on a portion of the calcine (Table 13.34) generated from Experiment No. 4 (calcining at 800^{o}C and 90 min. leaching) to determine the effect of recycling leachate to leach fresh calcine.

- Fresh calcine was leached with the recycled leach liquor at 90^{o}C, 20% solids, and 90min. residence time to determine the changes in concentration of the ions in the freshly generated leachate.
- Four cycles of contacting fresh calcine with recycled leachate were performed. During each cycle, an aliquot of the leach liquor was analyzed for potassium, aluminum, titanium, phosphorus, silicon, sodium, iron, magnesium, and sulfate.
- After the fourth cycle, potassium and sulfate concentrations increased from the initial leach by a factor 3- and 3.5-fold, respectively.

Table 13.27 summarizes the effect of recycling the leach liquor on extraction of potassium and aluminum.

TABLE 13.27 POTASSIUM AND ALUMINUM EXTRACTIONS BY CYCLE*

Cycle	Extraction, %		K/Al Mass Ratio
	K	Al	
1	83	2.2	17.2
2	81	2.0	17.1
3	89	3.2	18.8
4	87	3.4	17.9

*Mellon, Robert J. May 21, 2012.

- A ratio of potassium to aluminum is an indicator of increases in aluminum concentration as a function of the cycle.
- The data in Table 13.26 indicate that in successive contacts of the leachate with fresh calcine, up to four cycles, do not appear to affect the potassium extraction.
- Based on the mass ratio of potassium to aluminum, the concentration of aluminum is not increasing with respect to potassium over the four cycles.
- The columns labeled “Ext” in Table 13.26 indicate potassium or aluminum extractions from leached calcines and are calculated by accounting for all masses of species in solution and dividing it by that mass in solution plus unleached mass reporting to the leach residue.
- During calcination, potassium losses ranged from 0.1 to 1.4% as dust entrained in the off-gases from the kiln.
- In the FLSmidth Gas Suspension Calciner (GSC), or in any Dryer/Calciner/Roaster system, the off-gases will be cleaned by a train of cyclones and electrostatic precipitators (ESPs) and must be dust-free before being ducted to the Sulfuric Acid Plant. The dust collected and containing potassium and/or alumina will be recycled to the Water Leach circuit, thus minimizing loss of values.

Effect of Roasting and Leaching Conditions on Potassium Sulfate Recovery

The 2012 HRI experiments also investigated the effect of leaching temperature other than 90°C on extraction. Potassium extraction was only 61% when calcine generated at 800°C was leached at 25°C (room temperature) for 30 minutes at 20% solids. Aluminum extraction was 4%. Comparable leaches conducted at 90°C achieved 83% extraction for potassium and 9% for aluminum (Table 13.28).

The 1973 calcining and leaching tests results also compare favorably with those from the 2012 experiments:

- Significant increases in solubility of potassium and sulfate (SO_4) did not occur until the alunite ore was roasted at 750°C.
- The solubility of potassium peaked when roasting was at 800°C.
- At roasting temperature of 900°C, the solubility of potassium (84.7%) was about 5% less than that obtained (89.4%) at 800°C roasting temperature.

For a historical perspective, the 1973 roast/leach test results are summarized in Table 13.28. The 1973 historical test results compare favorably with those from the 2012 test results.

TABLE 13.28 EFFECT OF ROASTING AND LEACHING CONDITIONS ON PERCENT EXTRACTION OF POTASSIUM AND SULFATE*

Test No.	Roasting Temperature, °C	Leaching Temperature, °C	Roasting Weight Loss, %	% Dissolved	
				K	SO_4
Series 1					
T-2	510	25	5.0	4.3	2.9
T-3	750	25	12.8	47.9	30.2
T-4	800	25	18.2	89.4	87.1
T-5	850	25	19.3	86.4	85.9
Series 2					
T-14	0	85	0.0	0.5	<0.1
T-7	800	85	18.0	92.5	87.1
T-8	850	85	19.3	88.6	83.9
T-9	900	85	20.0	84.7	89.2

***** F. J. Bowen, et al. April 12, 1973.

Metallurgical tests in 1973, based on filtrate analyses, suggest that the amount of alumina and sulfate dissolved is dependent on the duration of the leach cycle.

The results of influence of leaching time on percentages of dissolved alumina and sulfate in the filtrate from water leaching of high temperature calcine produced at 800°C at 50wt.% solids and

80 to 90°C are given in Table 13.29. The results indicate that the alumina fraction in the calcine dissolves in water rapidly, but is re-precipitated with time.

The soluble alumina will report as alum during the subsequent step of potassium sulfate recovery from the leach liquor.

TABLE 13.29 INFLUENCE OF LEACHING TIME ON ALUMINA AND SULFATE DISSOLUTION*

Leaching Time, Minutes	% Dissolved	
	Al_2O_3	SO_4
15	5.2	80.0
60	3.3	80.5
180	2.1	60.0

Table 13.30 and Table 13.31 summarize the results of leaching tests at HRI in 1973 for establishing the influence of leaching parameters on percent extraction of potassium.

TABLE 13.30 PERCENT POTASSIUM EXTRACTION AS A FUNCTION OF PERCENT SOLIDS, LEACHING TEMPERATURE AND LEACHING TIME*

Hazen Leaching Test Results			
Temperature, °C	Leaching Time, Minutes	Solids, %	Potassium (K) Extraction, %
Room Temperature			
90	60	50	65.2
100	60	50	80.5
	60	17	74

TABLE 13.31 INFLUENCE OF LEACHING TIME ON WATER LEACHING OF HIGH TEMPERATURE (800°C) CALCINE*

Leaching Time, Minutes	Wt.% Dissolved	
	Al_2O_3	SO_4
15	5.2	80.0
60	3.3	80.5
180	2.1	60.0

***** F. J. Bowen, et al. April 12, 1973.

The effects of both calcining temperature and residence time on potassium and aluminum extractions should be considered when selecting a commercial calciner and evaluated in terms of associated capital and operating costs.

13.14 LEACHING ALUMINA

Extraction of SOP is the primary objective of the Blawn Mountain Alunite Project. The tailings produced at the end of the calcine leach cycle contain alumina, a potential resource for aluminum.

HRI conducted during 2012 two scoping studies to determine the extractability of aluminum from the tailings:

- The hot (80^oC) water-leach residue, derived from leaching material calcined at 850^oC for 60 minutes, was leached in 25wt% sulfuric acid at 90^oC for 60 minutes, cooled to 50^oC, and filtered through a Whatman 541 filter paper. Approximately 70% of the aluminum was extracted from the tailings. H_2SO_4 as a lixiviant shows promise for extracting aluminum from alumina-bearing tailings.
- The tests confirm that the alumina contained in the SOP leach residue produced by low temperature calcination, is soluble in hot caustic solutions.

HRI conducted a series of leach tests, during 2013, to confirm that the alumina contained in the SOP leach residue is in a form that is soluble in hot sodium hydroxide solutions. The SOP leach residues were produced from low temperature calcining of master composite samples. The results of the preliminary SOP Leach residue, alumina, leach tests are summarized in Table 13.32.

TABLE 13.32 RESULTS OF THE SOP LEACH RESIDUE, ALUMINA ,LEACH TESTS

Master Composite	SOP Leach Residue Sample No.	Particle Size, 80% Passing (um)	NaOH Concentration (g/L)	Temperature, Degrees C	Aluminum Recovery, %
MC-B	BK–31	500	200	60	77
MC-A	BK-34	1000	200	60	91
MC-C	BK-35	1000	200	60	63
MC-B	BK-36	1000	200	60	72

13.15 EFFECT OF PH ON LEACHING CALCINE

HRI selected minus 8-mesh calcine (F. J. Bowen, et al. April 12, 1973) produced in the 6in Screw Reactor during high temperature Test SR-13 to study the effect of pH of the lixiviant (leaching medium). Table 13.33 summarizes the characteristics of the calcine produced during high temperature Test SR-13.

TABLE 13.33 CHARACTERISTICS OF THE CALCINE FROM TEST SR-13*

Parameter	Test SR-13
Roasting: Temperature, ^{O}C Time, minutes Purge gas Feed, mesh	 850 60 Air 8
Leaching Temperature, ^{o}C	85
Calcine Analysis, wt.%: K SO_4 Al_2O_3	 4.01 12.5 19.1
Leach Weight Loss, wt.%	14.4
Dissolution, wt.%: K SO_4 Al_2O_3	 86.6 76.6 12

*F. J. Bowen, et al. April 12, 1973

Three parallel leaching tests were performed by Hazen at 85^{o}C using the calcine from high temperature Test SR-13. In these leaching tests, the pH of the leach solution was raised from its normal level of pH 4.0 to pH 6.0, 8.0, and 9.0. The pH adjustment was made using ammonium hydroxide (NH_4OH). The leaching test results show that:

- When leaching the calcine with water at 17wt.% solids, the potassium-rich liquor had a terminal pH of about 4.0.
- The solubility of potassium and sulfate increased by about seven percentage points when pH was raised from 4.0 to 6.0 with ammonium hydroxide.
- Little difference in the solubility of both potassium and sulfate when leaching at pH 6.0, 8.0, or 9.0.
- At all pH-adjusted levels, about 94% of the potassium and about 95% of the sulfate were dissolved.

Sulfate in Potassium Leach Residues

HRI performed in 1973 high (≈800^{o}C) and low (≈600^{o}C) temperature roasting studies on alunite composite sample NGC-101 followed by leaching the calcines with water (F. J. Bowen, et al. April 12, 1973) to determine the solubility of potassium and sulfate from the resultant calcines. The conclusions based on assays are follows:

- Water leached residues of calcines produced in the Vycor Retort contain sulfur in the sulfate form.
- The water-leached residues assayed at least 1.2% and as much as 3.4% SO_4 regardless of the potassium solubility over the range 85 to 94%.

- Approximately 18% of the SO_4 originally present in the calcine was present in the water-leached residue.
- Water-leached residues of calcines produced in the 6in Screw Reactor and from which about 90% of the potassium was dissolved assayed 1.8 to 2.7% SO_4, which represented about 18% of the SO_4 originally present in the calcine.

13.16 EFFECT OF LEACHING TEMPERATURE

Leaching of calcines with water experiments during 2012 had been conducted at 90°C. To evaluate the effect of leaching temperature on extraction, a sample of calcine generated at 800°C and 30 minutes residence time was leached at room temperature (25°C).

- Leaching at 25°C produced a potassium extraction of only 61% and 4% for aluminum.
- Comparable leaches conducted at 90°C resulted in 83% extraction for potassium and 9% for aluminum.

Additional leaching experiments between 25°C and 90°C will be required to evaluate the effects of leaching temperature on extraction of potassium and aluminum as well as other impurities.

13.17 RECENT SOLID LIQUID SEPARATION TESTING 2013

Leach residue or tailings samples were generated from a series of batch calcine leach tests for solid liquid separation testing. Fifteen kilograms samples of each composite were shipped to Pocock Industrial in Salt Lake City, Utah for sedimentation and filtration studies. The results of the tests are included in the appendices for this section.

Tests Performed on Calcine Leach Slurry

In October 2013, Pocock Industrial performed preliminary Conventional Thickening and Rheology tests on Composite A-, Composite B-, and Composite C Leach Slurry samples. The results of these tests are shown on Table 13.34.

TABLE 13.34 STATUS SUMMARY OF TESTS PERFORMED

Test Performed	Sample Tested		
	Composite A Leach	Composite B Leach	Composite C Leach
Sample characterization	Yes	Yes	Yes
Flocculant screening	Yes	Yes	Yes
Static thickening tests	Yes	Yes	Yes
Dynamic thickening tests	Yes	Yes	Yes
Pulp rheology	Yes	Yes	Yes
Vacuum filtration	Yes	Yes	Yes
Pressure filtration	Yes	Yes	Yes

Physical Properties of Calcine Leach Slurry

Table 13.35 is a summary of the Physical properties of the samples tested.

TABLE 13.35 PHYSICAL PROPERTIES OF SAMPLES

Property	Sample tested		
	Composite A Leach	Composite B Leach	Composite C Leach
Liquid specific gravity	1.04	1.05	1.05
Solids specific gravity	2.76	2.78	2.90
pH	5.5	5.6	5.2

The flocculant screening tests indicate Hychem AF 304, a medium-to-high molecular weight, 15% charge density, anionic polyacrylamide as the reagent of choice in solid/liquid separation operations. The tests were performed by diluting the pulp to a solids concentration likely to be encountered in the feed to the Thickener. The tests were based on the amount or dosage of flocculant required to initiate pinpoint floccule formation, effectiveness in capturing fines, the stability of the floccule, and resultant quality of the supernatant.

Flocculant Screening Tests for Calcine Leach Slurry

Table 13.36 is a summary of the results of flocculant screening tests.

TABLE 13.36 RESULTS OF FLOCCULANT SCREENING TESTS

Parameter	Sample Tested		
	Composite A Leach	Composite B Leach	Composite C Leach
pH	5.5	5.6	5.2
Temperature, ^{o}C	20	20	20
Initial Solids Concentration of Slurry Tested, wt.%	30%	20%	30%
Maximum effective dosage range, g/MT	20 – 30	25 – 35	30 – 40
Flocculant concentration*, g/L	0.1	0.1	0.1
Flocculant selected	Hychem AF 304	Hychem AF 304	Hychem AF 304

*Note: Concentration of flocculant in solution before contact with pulp

Thickening Calcine Leach Slurry

Conventional (static) thickening tests of calcine leach slurry at various flocculant doses and feed solids concentration (pulp density) were performed to select the parameters for conventional-type thickener design and to determine the operating conditions for high-rate (dynamic) thickening tests. Table 13.37 summarizes the thickening test results.

TABLE 13.37 THICKENING TEST RESULTS

Parameter	Sample Tested		
	Composite A Leach	Composite B Leach	Composite C Leach
Flocculant	Hychem AF 304	Hychem AF 304	Hychem AF 304
Flocculant dose, g/MT	20 – 30	25 – 35	30 – 40
Maximum Thickener Feed Solids, wt.%	25 – 35 (Conventional)	30 – 40 (Conventional)	20 – 30 (Conventional)
Maximum Unit Area for Conventional Thickener Sizing, m^2/MTPD	0.125	0.160 – 0.205	0.125
Estimated Underflow Density for Standard Thickener, %	69 - 73	61 – 65	69 – 73

- The static thickening test results (Unit Area basis) are used in the design of Conventional Thickeners.
- The dynamic thickening test results (hydraulic Net feed loading basis) are used in the design of High-rate Thickeners.
- Flocculant should be diluted to 0.1 – 0.2g/L with thickener overflow before contacting with pulp.

Pulp Rheology

Viscosity tests were performed in October 2013, to examine the rheological behavior of the thickened pulp across a specific shear range to correlate the relationship between apparent viscosity (Pa.sec) and shear rate (sec^{-1}), and shear stress (N/m^2) and shear rate (sec^{-1}) at operating temperatures, grind size, solids concentration, residual flocculant, and pH.

- Pulp viscosity data were collected using the Fann (Model 35A) Viscometer. The long-chain molecular structure of the flocculant was destroyed by shearing the underflow materials in a laboratory mixer before viscosity tests were performed.
- The pre-sheared data collected using the Fann Viscometer provides necessary information required to determine the maximum design underflow densities (% solids concentration) for standard conventional and high-rate thickeners.
- "Pseudoplasic Fluids" display decreasing apparent viscosity with increasing shear rate, or "shear-thinning" flow behavior. The thickener underflow pulps examined for the solids concentration range tested exhibited this behavior and, therefore, identified as belonging to the pseudoplastic class of "non-Newtonian fluids."

- In a Newtonian fluid, the relation between the shear stress and the shear rate is linear. In non-Newtonian fluids, the viscosity or the flow behavior changes with stress and is dependent on shear rate.
- The maximum recommended operating underflow pulp densities for ***Standard Thickener sizing*** based on the rheology data are summarized below:
  - Composite A Leach: 69% to 73%
  - Composite B Leach: 61% to 65%
  - Composite C Leach: 69% to 73%

Horizontal Belt Vacuum Filter

Vacuum filtration design criteria common to samples Composite A, Composite B and Composite C from tests performed in October 2013, are summarized below.

- Filter feed solids (Total Suspended Solids) = 71.3%
- pH of Slurry = 7.2
- Temperature of Feed Slurry = 20^{o}C
- Vacuum level = 67.7 kPa (9.8psi)
- Single stage countercurrent wash.

Table 13.38 summarizes the Horizontal belt Vacuum Filter test results for the three samples under four different operating conditions.

TABLE 13.38 HORIZONTAL BELT VACUUM FILTER TEST RESULTS

Parameter	Sample Tested		
	Composite A Leach	Composite B Leach	Composite D Leach
Test #1:			
Flocculant, g/MT	None	None	None
Cake thickness, mm	10	10	10
Cake moisture, %	17.8	17.7	17.8
Dry bulk density, kg/m^3	1834.8	1648.22	1799.39
Wet bulk density, kg/m^3	2233.29	2001.84	2190.13
Total cycle time, min.	1.48	3.51	3.74
Production rate, kg/m^{2}hr	593.7	225.39	231.02
Test #2:			
Flocculant, g/MT	None	None	None
Cake thickness, mm	15	15	15
Cake moisture, %	18.7	18.6	19.0
Dry bulk density, kg/m^3	1834.8	1648.22	1799.39
Wet bulk density, kg/m^3	2258.21	2025.63	2222.04
Total cycle time, min.	2.71	7.27	7.79
Production rate, kg/m^{2}hr	486.89	163.18	166.41
Test #3:			
Flocculant, g/MT	70	75	70
Cake thickness, mm	10	10	10
Cake moisture, %	20	22.1	20.9
Dry bulk density, kg/m^3	1345.68	1385.93	1357.10
Wet bulk density, kg/m^3	1682.95	1779.96	1715.05
Total cycle time, min.	0.59	0.56	0.55
Production rate, kg/m^{2}hr	1089.30	1185.09	1189.22
Test #4:			
Flocculant, g/MT	70	75	70
Cake thickness, mm	15	15	15
Cake moisture, %	21.1	23.5	22.5
Dry bulk density, kg/m^3	1345.68	1485.93	1357.10
Wet bulk density, kg/m^3	1706.38	1810.90	1751.50
Total cycle time, min.	0.71	0.64	0.61
Production rate, kg/m^{2}hr	1363.37	1564.18	1607.67

Automatic Pressure Filter

Two sets of Automatic Pressure Filter tests were performed on each of Composite A, Composite B and Composite D samples. The results are summarized for the three samples, respectively, in Tables 13.39 through 13.41.

TABLE 13.39 AUTOMATIC PRESSURE FILTER—COMPOSITE A SAMPLE OVERHEAD BEAM GHT – 2000MM – P/10

Parameter	Test #1	Test #2
Feed solids, %	69.7	69.7
pH	7.2	7.2
Air blow	Yes	Yes
Air blow & Squeeze	None	Yes
Feed temperature, ^{o}C	20	20
Feed pressure, kPa	551.6 kPa (80 psi)	551.6 kPa (80 psi)
Cake thickness, mm	60 (Full Cake)	60 (Full Cake)
Dry bulk density, kgm^3	1227.65	1472.02
Wet bulk density	1401.59	1662.40
Moisture content, %	**12.4**	**11.5**
Sizing basis, m^3/MT	1.018 (Note #1)	0.849 (Note #1)
Filtration rate based on Total cycle time, kg/m^2hr	92.4	92.4
Filtration rate based on 30 mm Cake + Dry time, $kg/m^2.hr$	463.9	509.6
Cake form time, min	0.81	1.16
Wash time, min.	0	0
Air blow/Squeeze time, min.	3.0	3.0
Miscellaneous time, min	10.0 (Note#2)	10.0 (Note #2)
Total cycle time, min.	16.0	16.0
Number of Filters	1.0	1.0
No. of Cycles per 20-hr day	75	75
Metric tons/day/filter	1192.0	1192.0
Metric tons/cycle required	15.9	15.90
M^3/cycle required	16.19	13.50
No. of recess plate chambers	89	74

Notes: #1 Sizing basis in m^3 of pressure filter volume per metric ton (MT) of dry solids, including 1.25 scale-up factor.
#2 Miscellaneous time of 10 minutes for cake discharge and cloth washing.

TABLE 13.40 AUTOMATIC PRESSURE FILTER—COMPOSITE B SAMPLE OVERHEAD BEAM GHT – 2000MM – P/8

Parameter	Test #1	Test #2
Feed solids, %	69.1	69.1
pH	7.2	7.2
Air blow	Yes	Yes
Air blow & Squeeze	None	Yes
Feed temperature, ^oC	20	20
Feed pressure, kPa	551.6 kPa (80 psi)	551.6 kPa (80 psi)
Cake thickness, mm	60 (Full Cake)	60 (Full Cake)
Dry bulk density, kg.m^3	1760.48	2047.22
Wet bulk density	2052.71	1662.40
Moisture content, %	**14.2**	**13.1**
Sizing basis, m^3/MT	0.710 (Note #1)	0.611 (Note #1)
Filtration rate based on Total cycle time, kg/m^2.hr	128.1	138.5
Filtration rate based on 30 mm Cake + Dry time, kg/m^2.hr	386.6	377.4
Cake form time, min	3.56	4.81
Wash time, min.	0	0
Air blow/Squeeze time, min.	3.0	3.0
Miscellaneous time, min	10.0 (Note#2)	10.0 (Note #2)
Total cycle time, min..	16.56	17.81
Number of Filters	1.0	1.0
No. of Cycles per 20-hr day	72.5	67.4
Metric tons/day/filter	1192.0	1192.0
Metric tons/cycle required	16.45	17.69
M^3/cycle required	11.68	10.80
No. of recess plate chambers	64	60

Notes:
#1 Sizing basis in m^3 of pressure filter volume per metric ton (MT) of dry solids, including 1.25 scale-up factor.
#2 Miscellaneous time of 10 minutes for cake discharge and cloth washing.

TABLE 13.41 AUTOMATIC PRESSURE FILTER—COMPOSITE D SAMPLE OVERHEAD BEAM GHT – 2000MM – P/8

Parameter	Test #1	Test #2
Feed solids, %	70.7	69.1
pH	7.2	7.2
Air blow	Yes	Yes
Air blow & Squeeze	None	Yes
Feed temperature, ^{o}C	20	20
Feed pressure, kPa	551.6 kPa (80 psi)	551.6 kPa (80 psi)
Cake thickness, mm	60 (Full Cake)	60 (Full Cake)
Dry bulk density, kg.m^3	1806.96	2072.81
Wet bulk density	2052.71	2359.25
Moisture content, %	**13.2**	**12.1**
Sizing basis, m^3/MT	0.692 (Note #1)	0.603 (Note #1)
Filtration rate based on Total cycle time, kg/m^2.hr	130.0	139.3
Filtration rate based on 30 mm Cake + Dry time, kg/m^2.hr	385.6	376.3
Cake form time, min	3.75	4.93
Wash time, min.	0	0
Air blow/Squeeze time, min.	3.0	3.0
Miscellaneous time, min	10.0 (Note#2)	10.0 (Note #2)
Total cycle time, min..	16.75	17.93
Number of Filters	1.0	1.0
No. of Cycles per 20-hr day	71.6	66.9
Metric tons/day/filter	1192.0	1192.0
Metric tons/cycle required	16.64	17.81
M^3/cycle required	11.51	10.74
No. of recess plate chambers	63	59

Notes:
#1 Sizing basis in m^3 of pressure filter volume per metric ton (MT) of dry solids, including 1.25 scale-up factor.
#2 Miscellaneous time of 10 minutes for cake discharge and cloth washing.

HRI performed in 2012 vacuum filtration and Kynch settling experiments on slurry obtained by water leaching calcines produced at 850^{o}C and 30min residence time.

The vacuum filtration experiments were conducted to test the dewatering aids and to estimate the best operating conditions for the filtration operations. Traditional laboratory testing, using a filter leaf or Buchner Funnel, permits the multi-step process of the horizontal vacuum filter to be demonstrated in the lab as well as in the selection of production scale equipment. Leaf testing indicates the filtration rate and is used to check the feasibility of a type of filter. Proven filter leaf scale-up factors allow the leaf test results to be used to "size" vacuum filters.

- Three feed slurry loadings at an approximate pulp density of 50% were examined with the vacuum filtration.
- The cake form rate averaged 808kg/m^2/hr. over the three runs.
- The residual moisture content of the filter cake was 30%.

The vacuum filtration test results are summarized in Table 13.42.

TABLE 13.42 EIMCO VACUUM FILTER LEAF TEST RESULTS*

Filter Leaf Area, m^2	0.0045	Test ID		
Slurry Temperature	Ambient	3136-136-1	3136-136-1	3136-136-1
Feed slurry solids, %		47	50	50
Feed slurry mass, g		183	234	275
Wet Filter cake mass, g		125	164	191
Dry Filter cake mass, g		86.2	118.1	136.8
Diatomaceous earth, g		5.6	6.6	7.6
Flocculant, mg		0	0	0
Filter cake solids, %		69	72	71
Vacuum, mm Hg		520	520	520
Cake form time, min.		1.35	1.92	2.42
Cake thickness, mm		10	15	20
Cake form rate, (kg/m^2)/hr		851	821	755

*Mellon, Robert J. May 21, 2012

Settling, or sedimentation of a suspension is generally evaluated by a “jar test” during which a suspension is allowed to settle and the height of the clear liquid (supernatant)-suspension interface is measured as a function of the settling time. Thickener design involves the application of one of a number of alternative models together with jar test data. The Kynch theory of sedimentation is based on the assumption that at any point in the suspension the settling velocity of a particle depends only on the local concentration of the suspension and the particles are of the same size and shape.

A plot of settling rate versus concentration can be constructed from a single settling curve. Kynch developed methods for determining concentrations and fluxes of solids in the transition zone between the constant rate period and the final underflow concentration from a single batch settling test. This would allow the prediction of the unit area (ft^2/ton solids-day) for specific thickening applications. HRI used in the Kynch experiments two doses of Hychem AF-303-HH flocculant and Table 13.43 summarizes the Settling test results.

TABLE 13.43 KYNCH SETTLING TEST RESULTS*

Parameter	Units	Experiment 1	Experiment 2
Flocculant Dosage	mg/kg	12	30
Initial Settling Rate	ft/hr	242	390
Settled solids	%	48.4	50.6
Unit Area	ft^2/(st/d)	0.16	0.17

*Mellon, Robert J. May 21, 2012.

The Kynch settling tests indicated:

- The flocculant cleared the supernatant and produced an underflow pulp density of 48 to 50wt.% solids.
- The settling area for solids was estimated at 0.16 to 0.17ft^2/(st/day).
- Of the two flocculant dosages, the lowest dosage of 12mg/kg of solids proved sufficient in clearing the supernatant of turbidity.

13.18 CRYSTALLIZATION TEST WORK

HRI observed in the water-leach tests (Mellon, Robert J. May 21, 2012) the concentration of potassium in the leach liquor was 12.8g/L at five minutes and 12.4g/L at the end of 60 minutes leach cycle, whereas the respective aluminum concentration declined from 0.5g/L to 0.1g/L as the leach progressed. This suggests:

- Minimal contact time with the water may extract the potassium from the calcines.
- Longer contact time with water may allow precipitation and removal of aluminum from the leachate. This would reduce the amount of aluminum in the feed to the down stream K_2SO_4 crystallizer, thus enhancing the quality of the SOP product.

HRI performed one crystallization experiment with the goal to generate crystals in several stages so that the co-precipitation of impurities can be evaluated.

- The crystallization experiment recovered 43% of the potassium in the feed solution by reducing the liquor volume 91% by evaporation and the crystals assayed 25 wt.%K with small amounts of other contaminants.
- The brine was generated by leaching at 90°C for 1 hour the calcine produced at 900°C and a residence time of 30 minutes and filtering the slurry using Whatman 541 filter paper.
- The filtrate used as feed to the crystallizer contained 11.7g/L K and 0.081g/L Al.
- The crystallizer was a "resin kettle" with an overhead agitator placed in a heating mantle and under vacuum for maintaining boiling at 70 to 75°C.
- Designated Cycle 1, about 20mg of crystals was formed after reducing the feed volume to the crystallizer from 691mL to 305mL or 55%.

- The crystals from Cycle 1 contained both potassium and aluminum at 8.0 and 13.5wt.%, respectively.
- About 160mg of crystals assaying 24wt.% K and less than 0.05wt.% Al and indicating K_2SO_4 crystallization were precipitated between the first and second crystallization cycles by cooling the primary filtrate to room temperature.
- The slurry from the previous step was filtered and the filtrate was further heated for one hour to produce an additional crystal crop.
- The Cycle 1 filtrate as feed for Cycle 2 experiments was evaporated from 327mL to 69mL, or an additional 79%. After washing, 200mg of crystals were collected, which assayed 25wt.% K, less than 0.05% Al, and less than 0.001wt.% Ti, 0.021wt.% Na, 0.025 wt.% Fe, and less than 0.0025wt.% Mg.

Leaching the calcines at higher solids content and/or recycling the leach solution to buildup the concentration of potassium should be investigated to reduce the large (91%) evaporation requirement and to improve the potassium recovery without co-crystallizing contaminants.

In the commercial-scale evaporation/crystallization operation, a limited amount of K_2SO_4 will be recovered until other salts begin to crystallize, contaminating the SOP product. A bleed from the crystallizer will be necessary to prevent contaminating the K_2SO_4 product.

- Depending on the size of this bleed, which may contain considerable potassium values, an option such as membrane separation (Reverse Osmosis) process for recovering potassium from the bleed stream should be evaluated.

PRC contracted the services of Consultants with specialized knowledge in the applied fields of thermodynamics and heat transfer and Evaporators and Crystallizers. Because crystallization is energy-intensive process the best design requires consideration of the process fluids, contaminants, maintenance and the relative cost of electricity and steam.

Pending pilot plant test results on the brine produced by leaching the calcine, tentative recommendations on the Evaporator/Crystallizer circuit are as follows:

- The quadruple effect Crystallizer design which uses evaporated water from the process stream to drive a second heat exchanger at a lower pressure is recommended.
- Mechanical Vapor Recompression (MVR) systems utilize mechanical fans and use less steam than the multi-effect design. However, they require large heat exchangers with cooling water. After start up of an MVR system, no additional steam is required for operation.
- A quadruple effect Crystallizer is the equipment of choice due to the cooling water requirements of the MVR.

- Impurity build-up should be controlled in the recirculated centrate by establishing a purge or "bleed" stream from the Centrifuge dewatering circuit. Bench-scale investigations will be required to determine the amount of bleed and to identify a method of treatment to recover the contained values. After treatment, the barren solution essentially containing sodium chloride can be routed to an evaporation pond.

13.19 Flotation of Alunite

The alunite ore evaluated by HRI in 2012 contained up to 45wt.% quartz. Physical beneficiation, such as flotation, to remove the quartz fraction was not performed during the 2012 laboratory investigations.

In the 1970s, HRI investigated flotation as a method of concentrating of alunite and targeted an 80% recovery at 80% grade of alunite. In February 2013, objectives of a flotation test campaign were outlined:

- Target of 80% recovery at 80% grade.
- Depress the silica to tails.
- Float the alunite mineral.
- Use sulfur as the determinant for alunite concentration.

Flotation was identified as a candidate technology for the recovery of alunite concentrate with low levels of quartz as an alternative to whole ore processing because if this approach is technically viable, flotation has the potential to reduce the capital and operating costs, result in energy savings in the calcining step, and reduce the equipment size in calcining and leaching operations. By mid-2013, HRI completed 71 scoping alunite flotation experiments to evaluate the process parameters.

- P_{80} of 80 µm was chosen as the lower limit for fluid bed calcining/roasting operations.
- The dosage of collector was varied from 0.1 to 4.0lb/t of solids.
- Soda ash (Na_2CO_3) and potassium hydroxide (KOH) were the pH modifiers to achieve the target value of pH 10.0. KOH was used to eliminate the build-up, if any, of sodium in the feed to the Calciner.
- The dosage of frother W22C was approximately 0.026lb/t.
- Sodium silicate (Na_2SiO_3) at a dosage of 1.8lb/t was investigated to depress silica.
- Flotation experiments were conducted at ambient temperature.
- The flotation experiments indicated approximately 78% recovery at 85% alunite grade was possible using oleic acid and vapor oil (a by-product of petroleum refining process) as collectors in open-circuit flotation.

- Though flotation proved to be technical feasibility, the dosage and cost of reagents proved prohibitive. A shift was then made to whole ore processing.

Processing whole ore is the chosen technology over alunite flotation concentrate for the following reasons:

- Coarser particle size in the 1000 μm range is well suited for energy-efficient Gas Suspension Calciner or Fluid Bed Roaster.
- Relatively fine-ground flotation concentrate is not suitable for fluidized bed reactors.
- Moisture content of flotation concentrate filter cake is relatively high compared with coarse-ground whole ore filter cake.
- Water and energy conservation are of paramount importance at the Project site.

Historical Flotation Test Results

In 1976, Alumet investigated the flotation of alunite from the ore and silica from a water leach residue (R. Myertons, May 27, 1976). The results of laboratory investigations from this report are summarized below. The objective of this investigation was to produce a concentrate containing less than 5wt.% silica at greater than 50wt.% recovery of alumina or alunite.

- The test work on alunite ore produced concentrates containing about 8wt.% silica, at 30 to 40% to alunite recovery.
- At 50% alunite recovery, under open-circuit conditions, silica content of the concentrate was about 13 wt.% SiO_2.
- Examination of leach residue indicated that the silica was present as porous cryptocrystalline aggregates with alunite between the quartz crystals.
- Preliminary tests were conducted with amine as the reagent for floating silica from water leach residue and alunite concentrate.
- Amine reagents were found to be more effective as collectors for alumina than silica.

The PZC for corundum or alumina (Al_2O_3) is in the pH 9.0-9.3 range and amines should not float alumina below that pH. Based on this principle, preliminary tests were conducted to develop a reverse flotation system to separate silica from alunite. These tests indicate that amines tend to float alunite in preference to quartz.

13.20 Sulfuric Acid Plant

An estimated 3,390tpd of sulfuric acid (H_2SO_4) is manufactured at the Project site from sulfur dioxide (SO_2) produced by the decomposition of alunite during the Drying/Calcining/Roasting of ROM ore. FLSmidth estimates the concentration of SO_2 and SO_3 at 9.44vol. % in the off gases as feed to the acid plant from the proposed four (4) Dryer/Calciner/Roaster systems, each with a throughput capacity of 330tph of Filter Cake at 90% solids content.

Based on information developed by FLSmidth by desktop simulations for the Drying, Calcining, and Roasting of whole ore, Du Pont-MECS, a supplier of the Sulfuric Acid plant, estimates the composition of off-gases as feedstock to acid plant as given in Table 13.44.

TABLE 13.44 PRELIMINARY ESTIMATE OF COMPOSITION OF FEEDSTOCK TO THE SULFURIC ACID PLANT*

Base Case		84,993			Dilution Air		
Component	**Vol.%**	**WET NM^3/hr**	**DRY NM^3/hr**	**Vol.%**	**DRY NM^3/hr**	**DRY NM^3/hr**	**vol.%**
SO_2	9.2	9,164	9,164	11.5	-	9,164	7.3
O_2	1.3	1,300	1,300	1.6	9,697.2	10,997	8.7
CO_2	0.0	0	0	0.0	-	0	0.0
N_2	69.0	68,979	68,979	86.8	36,572.8	105,552	84.0
H_2O	20.5	20,494	0	0.0		0	0.0
SO_3	0.0	0	0	0.0		0	0.0
Total	**100**	**99,937**	**79,443**	**100**		**125,713**	**100**

*DuPont MECS

- It is proposed that two (2) Sulfuric Acid plants, each with a throughput capacity of 2,000tpd of concentrated acid be constructed, each dedicated to process the off-gases from two (2) lines of Roasters and with provisions for treating off-gases bypassing any of the Roasters being serviced.
- To improve yields and to minimize atmospheric pollution, modern day plants incorporate what is known as "interpass absorption" design, also known as "double catalysis" in which a second absorbing tower removes SO_3 formed from SO_2 before the last stage of the converter is inserted. With this design, it is possible for the yield to be 99.7% to 99.9% and for the SO_2 in the exit gas to be 100 to 350ppm. Without the interpass, the exit gas may contain up to 2,000ppm sulfur dioxide.
- Tail gas scrubbing is required to further reduce the concentration of SO_2 to less than 50ppm in stack discharges to the atmosphere.

13.21 SULFUR RECOVERY UNIT

Recovery of elemental sulfur from the roaster off-gases was considered as an alternative to manufacture of sulfuric acid at the Project site. PRC contracted Fluor Corporation (Fluor) to evaluate the technical feasibility and economic viability of a Sulfur Recovery Unit (SRU).

Fluor identified the following two technologies, which allow achievement of 99.9 plus percent sulfur recovery:

- Option 1: The Fluor SO_x® Claus process for production of bright yellow sulfur
- Option 2: Use of commercially-proven technology.

Option 1 consists of the following processing steps:

- Cansolv® (licensed by Shell) SO_2 Scrubbing System.
- Reducing Gas Generation (RGG) by burning natural gas to generate H_2, CO, etc.
- Hydrogenation Reactor for converting SO_2 to H_2S.
- Purification Section (Amine Absorber/Amine regenerator).
- Fluor SO_x® Claus Conversion to produce elemental sulfur.
- Thermal Oxidation.

Option 2 consists of the following commercially-proven technology.

- Catalytic Oxygen Elimination.
- Reducing Gas generation.
- Hydrogenation.
- Purification (or Solvent Section).
- Fluor SO_x® Claus Conversion to produce elemental sulfur.
- Thermal Oxidation.

The SRU as an alternative to sulfuric acid production is not being considered because of such considerations as CAPEX, OPEX, and volatility of the sulfur market.

14 MINERAL RESOURCE ESTIMATES

Four potential mine development targets have been identified within the Blawn Mountain property. Only Area 1 and Area 2 have sufficient geologic and analytical data to support resource estimation at this time. Areas 3 and 4 are defined by a limited number of historical holes and surface mapping along with only two validation holes in Area 4. Both areas are recognized as future exploration targets.

Norwest has estimated resources from 3DGBM's constructed in MineSight®, a software package developed by Mintec Inc. The estimate was prepared in compliance with NI 43-101 requirements for the definition of mineral resources. The 3DGBM's are based on the assays and lithologies of the current drilling database and on a series of 30 interpreted geological cross sections constructed through Area 1 and 29 cross sections constructed through Area 2.

A total of 142 exploration drill holes have been completed by PRC on the property as of the effective date of November 6, 2013. These holes include 75 twin and infill holes in Area 1 and 67 infill holes in Area 2. As discussed in Section 12, there was poor correlation observed with the twin drilling program conducted by PRC. A decision was made by PRC in 2013 to no longer use the historical data and that more reliable estimates would be achieved using only the recent (2011-2013) PRC drilling data. Holes not included in the geologic model include all pre-2011 historical holes completed by ESI. There are insufficient records for these air track holes to be used in the geologic model. Remaining historic holes were excluded from the geologic model due to lack of sufficient documentation relating to assay testing standards.

A number of criteria were established for determination of resources:

1. A statistical review of analytical results through the construction of a series of correlograms determined that there was no appreciable preferred orientation of grades for K_2O and Al_2O_3. Down-hole variograms were also prepared and showed that there were no significant nugget effects or directionality to the data that would require more robust kriging approaches. Correlograms and down-hole variograms are of K_2O and Al_2O_3 for Area 1 and Area 2 are outlined in Figure 14.1 and Figure 14.2 respectively.
2. Analytical results were based on composites developed over 10ft intervals in each hole.
3. Four lithologic domains are represented in the geologic block model: Alunite, Clay, Dolomite, and Silica.
4. The geologic block model for Area 1 has the overall dimensions of 5,900ft west to east, 3,900ft north to south and 1,400ft elevation range. The geologic block model for Area 2 has the overall dimensions of 8,800ft west to east, 11,960ft north to south and 1,700ft elevation range. The block model minimum and maximum dimensions are outlined in Table 14.1 for Area 1 and Table 14.2 for Area 2. All units outlined in Table 14.1 and Table 14.2 are in Utah

State Plane – South coordinates, NAD27. Unless otherwise specified, units are reported in US Customary Units (feet/short tons).

TABLE 14.1 AREA 1 BLOCK MODEL DIMENSIONS

	Minimum	Maximum	Block Size
Easting	1,418,100	1,424,000	20
Northing	591,300	595,200	20
Elevation	6,300	7,700	20

TABLE 14.2 AREA 2 BLOCK MODEL DIMENSIONS

	Minimum	Maximum	Block Size
Easting	1,423,200	1,432,000	20
Northing	583,540	595,500	20
Elevation	6,000	7,700	20

5. A standard cubic block size of 20ft (X-dimension) by 20ft (Y-dimension) by 20ft (Z-dimension) was used in both the Area 1 and Area 2 block models.
6. First pass data search radii for K_2O estimation were 350ft and Al_2O_3 were 250ft for both models. Second pass data search radii for K_2O and Al_2O_3 were 2,000ft for both models. The larger search radii for the Area 2 model was used to account for the more widely spaced drilling.
7. Topographic data for the Area 1 block model is sourced from a USGS digital terrain model (DTM). DTM has a 10 m (32.8ft) resolution. Topographic data for the Area 2 block model is sourced from a Utah Automated Geographic Reference Center (AGRC) digital elevation model (DEM). The DEM has a 5m (16.4ft) resolution.
8. Resource classification is based on set distances from drill hole sample intervals in 3D space. These distances were based on semi-variogram analysis of K_2O sample data. Table 14.3 outlines the distance from drill hole samples used to classify resource blocks within the alunite zone.

TABLE 14.3 CLASSIFICATION CRITERIA

Measured	Indicated	Inferred
<150ft	<350ft	<2,000ft

The assumed density of alunite and waste was established at 153.8 pounds per cubic foot (lb/ft^3) or specific gravity of 2.46 grams per cubic centimeter (g/cc), as derived from estimates used previously by ESI (1974). Norwest believes that this bulk density factor is reasonable for this deposit type.

9. The boundaries of the deposit were defined by the applied radii of influence of drill holes or interpreted structural controls such as known bounding fault systems and alteration limits. These limits have been updated to reflect the relationship between SO_4 grade data and alunite mineralization. As discussed in Section 12, the recent drill hole data has indicated the presence of high K_2O grades in feldspar-rich rhyolitic country rock and that there is an association between SO_4 and alunite mineralization. To better define the boundary between country rock and alunite mineralized zones drill hole sample intervals with greater than 0.8% SO_4 were used to separate alunite mineralization from surrounding country rock.
10. Both visual and calculated validation of model block values to posted drill assay values show strong correlation. A series of swath plots of comparing average K_2O and Al_2O_3 grades from the 3DGBM with against drill hole grades are illustrated in Figure 14.3 and Figure 14.4 respectively.

Resource classification is based on the CIM Standards on Mineral Resources and Reserves, a set of definitions and guidelines established by the Canadian Institute of Mining and Metallurgy and Petroleum. Table 14.4 shows the estimated classified resource for the Area 1 Blawn Mountain Alunite deposit at increasing incremental K_2O cut-off grades. Figures 14.5, 14.6, and 14.7 show cross sections through the block model for Area 1. The cross sections exhibit typical zoned mineralization for hydrothermal alteration also referred to as "nested cone geometry" by Krahulec. Figure 14.8 identifies the classified resource areas for the Area 1 Blawn Mountain property.

The preferred scenario for resource presentation is a 1% K_2O cut-off grade. At a 1% cut-off grade, there is a combined measured plus indicated resource of 164.8Mt of material carrying an average grade of 3.35% K_2O and 15.41% Al_2O_3. The calculated potassium sulfate grade (K_2SO_4) at a 1% K_2O cut-off grade is 6.20%. This cut-off grade maximizes the in-place tons while providing a quantity of K_2SO_4 deemed suitable by current processing studies.

Increasing the cut-off grade to 3% K_2O reduces the combined tons of material to 106.3Mt. Average grade at a 3% K_2O cut-off is 3.94% K_2O and 16.50% Al_2O_3 with a calculated equivalent grade of 7.29% K_2SO_4. Approximately 43% of the identified resources are classified as measured, 56% as indicated resource and 1% as inferred resource.

TABLE 14.4 CLASSIFIED RESOURCE ESTIMATE FOR THE AREA 1 BLAWN MOUNTAIN ALUNITE DEPOSIT

			IN SITU GRADES						CONTAINED RESOURCES				
RESOURCE CLASSIFICATION	CUTOFF GRADE K_2O (%)	IN SITU (TONS)	K_2O (%)	K_2SO_4 (%)	AL_2O_3 (%)	SO_4 (%)	Alunite based on K_2O (%)	Alunite based on Al_2O_3 (%)	K_2O (TONS)	K_2SO_4 (TONS)	Al_2O_3 (TONS)	Alunite based on K_2O (TONS)	Alunite based on Al_2O_3 (TONS)
MEASURED	0.00	72,400,282	3.42	6.32	15.68	2.09	30.04	42.45	2,473,700	4,574,597	11,353,088	21,746,273	30,734,994
	1.00	71,529,372	3.45	6.39	15.71	2.06	30.36	42.52	2,469,909	4,567,586	11,235,691	21,712,945	30,417,176
	2.00	64,979,040	3.64	6.73	16.19	2.09	32.01	43.83	2,366,212	4,375,819	10,520,042	20,801,341	28,479,776
	2.50	56,872,179	3.84	7.10	16.33	2.13	33.75	44.21	2,183,323	4,037,604	9,286,829	19,193,568	25,141,232
	3.00	48,362,178	4.03	7.44	16.62	2.19	35.39	45.01	1,946,916	3,600,418	8,040,067	17,115,319	21,766,008
	3.50	34,526,334	4.33	8.00	17.31	2.31	38.03	46.87	1,493,713	2,762,313	5,976,957	13,131,213	16,180,773
	4.00	19,624,648	4.78	8.84	18.71	2.55	42.01	50.64	937,725	1,734,127	3,671,046	8,243,526	9,938,226
INDICATED	0.00	93,823,555	3.26	6.03	15.19	2.03	28.65	41.12	3,057,710	5,654,601	14,250,203	26,880,292	38,578,041
	1.00	93,313,743	3.27	6.05	15.19	2.02	28.78	41.12	3,054,532	5,648,725	14,172,865	26,852,358	38,368,671
	2.00	83,493,484	3.48	6.43	15.88	2.05	30.57	42.98	2,902,985	5,368,470	13,255,676	25,520,109	35,885,665
	2.50	74,184,688	3.63	6.71	16.06	2.09	31.91	43.47	2,692,385	4,979,008	11,911,168	23,668,726	32,245,822
	3.00	57,939,557	3.87	7.15	16.40	2.14	34.01	44.40	2,241,624	4,145,419	9,503,304	19,706,088	25,727,272
	3.50	36,959,714	4.21	7.78	17.30	2.29	36.97	46.84	1,554,489	2,874,705	6,395,250	13,665,492	17,313,172
	4.00	17,565,100	4.73	8.75	19.11	2.57	41.61	51.73	831,391	1,537,486	3,356,462	7,308,752	9,086,589
MEASURED AND INDICATED	0.00	166,223,837	3.33	6.15	15.40	2.06	29.25	41.70	5,531,410	10,229,198	25,603,291	48,626,565	69,313,034
	1.00	164,843,115	3.35	6.20	15.41	2.04	29.46	41.73	5,524,441	10,216,310	25,408,555	48,565,303	68,785,847
	2.00	148,472,524	3.55	6.56	16.01	2.07	31.20	43.35	5,269,197	9,744,288	23,775,718	46,321,450	64,365,441
	2.50	131,056,867	3.72	6.88	16.17	2.11	32.71	43.79	4,875,708	9,016,612	21,197,996	42,862,294	57,387,054
	3.00	106,301,735	3.94	7.29	16.50	2.17	34.64	44.68	4,188,540	7,745,837	17,543,371	36,821,407	47,493,280
	3.50	71,486,048	4.26	7.89	17.31	2.30	37.49	46.85	3,048,201	5,637,017	12,372,207	26,796,705	33,493,946
	4.00	37,189,748	4.76	8.80	18.90	2.56	41.82	51.16	1,769,116	3,271,614	7,027,508	15,552,278	19,024,815
INFERRED	0.00	2,255,374	3.18	5.87	14.62	2.04	27.92	39.57	71,626	132,458	329,697	629,665	892,554
	1.00	2,255,374	3.18	5.87	14.62	2.04	27.92	39.57	71,626	132,458	329,697	629,665	892,554
	2.00	1,919,126	3.48	6.44	15.87	2.10	30.60	42.95	66,797	123,527	304,485	587,213	824,299
	2.50	1,793,895	3.56	6.59	15.98	2.11	31.33	43.26	63,938	118,240	286,657	562,078	776,036
	3.00	1,429,416	3.77	6.97	16.36	2.11	33.15	44.29	53,899	99,675	233,868	473,825	633,126
	3.50	665,917	4.37	8.08	18.68	2.44	38.41	50.56	29,097	53,809	124,375	255,794	336,706
	4.00	407,414	4.78	8.84	20.45	2.68	42.05	55.35	19,486	36,036	83,298	171,303	225,504

Effective Date: November 6, 2013

Table 14.5 shows the estimated classified resource for the Area 2 Blawn Mountain Alunite deposit at increasing incremental K_2O cut-off grades. Figures 14.9, 14.10, and 14.11 show cross sections through the block model for Area 2. Figure 14.12 identifies the classified resource areas for the Area 2 Blawn Mountain Property.

At a 1% cut-off grade, there is a combined measured plus indicated resource of 398.4Mt of material carrying an average grade of 2.97% K_2O and 12.99% Al_2O_3. The calculated potassium sulfate grade (K_2SO_4) at a 1% K_2O cut-off grade is 5.50%. This cut-off grade maximizes the in-place tons while providing a quantity of K_2SO_4 deemed suitable by current processing studies.

Increasing the cut-off grade to 3% K_2O reduces the combined tons of material to 180.5Mt. Average grade at a 3% K_2O cut-off is 3.65% K_2O and 14.25% Al_2O_3 with a calculated equivalent grade of 6.76% K_2SO_4. Approximately 20% of the identified resources are classified as measured, 55% as indicated resource and 25% as inferred resource.

TABLE 14.5 CLASSIFIED RESOURCE ESTIMATE FOR THE AREA 2 BLAWN MOUNTAIN ALUNITE DEPOSIT

RESOURCE CLASSIFICATION	CUTOFF GRADE K_2O (%)	IN SITU (TONS)	IN SITU GRADES						CONTAINED RESOURCES				
			K_2O (%)	K_2SO_4 (%)	AL_2O_3 (%)	SO_4 (%)	Alunite based on K_2O	Alunite based on Al_2O_3	K_2O (TONS)	K_2SO_4 (TONS)	Al_2O_3 (TONS)	Alunite based on K_2O (TONS)	Alunite based on Al_2O_3 (TONS)
MEASURED	0.00	110,497,331	2.87	5.31	12.42	1.94	25.23	33.62	3,170,721	5,863,592	13,720,785	27,873,773	37,144,805
	1.00	104,377,825	3.02	5.58	13.05	2.04	26.54	35.33	3,150,958	5,827,044	13,622,767	27,700,035	36,879,452
	2.00	93,679,360	3.18	5.87	13.38	2.11	27.91	36.22	2,974,320	5,500,388	12,532,237	26,147,212	33,927,177
	2.50	79,064,980	3.34	6.17	13.60	2.20	29.33	36.81	2,638,240	4,878,879	10,750,703	23,192,741	29,104,220
	3.00	50,041,863	3.68	6.80	14.26	2.43	32.34	38.60	1,841,190	3,404,900	7,134,769	16,185,883	19,315,191
	3.50	28,969,753	3.98	7.36	14.75	2.62	34.97	39.93	1,152,272	2,130,888	4,272,459	10,129,610	11,566,369
	4.00	9,150,291	4.47	8.26	15.62	2.72	39.27	42.29	408,771	755,937	1,429,550	3,593,501	3,870,067
INDICATED	0.00	307,822,418	2.84	5.25	12.47	1.90	24.96	33.77	8,739,386	16,161,686	38,398,692	76,827,849	103,952,645
	1.00	293,961,004	2.96	5.47	12.97	1.97	25.98	35.12	8,688,605	16,067,777	38,135,561	76,381,434	103,240,300
	2.00	263,614,932	3.10	5.73	13.34	2.05	27.25	36.12	8,171,536	15,111,563	35,174,931	71,835,880	95,225,305
	2.50	212,810,329	3.30	6.10	13.64	2.15	29.00	36.93	7,020,826	12,983,563	29,031,159	61,720,000	78,592,933
	3.00	130,484,506	3.64	6.74	14.25	2.37	32.04	38.57	4,755,377	8,794,086	18,592,476	41,804,470	50,333,410
	3.50	71,126,489	3.96	7.32	14.70	2.57	34.81	39.79	2,816,324	5,208,209	10,454,527	24,758,277	28,302,416
	4.00	20,689,481	4.48	8.28	15.52	2.62	39.37	42.03	926,537	1,713,438	3,211,794	8,145,177	8,694,943
MEASURED AND INDICATED	0.00	418,319,749	2.85	5.27	12.46	1.91	25.03	33.73	11,910,107	22,025,277	52,119,477	104,701,621	141,097,450
	1.00	398,338,829	2.97	5.50	12.99	1.99	26.13	35.18	11,839,563	21,894,821	51,758,329	104,081,470	140,119,752
	2.00	357,294,292	3.12	5.77	13.35	2.07	27.42	36.15	11,145,855	20,611,952	47,707,169	97,983,092	129,152,483
	2.50	291,875,309	3.31	6.12	13.63	2.17	29.09	36.90	9,659,066	17,862,442	39,781,862	84,912,741	107,697,153
	3.00	180,526,369	3.65	6.76	14.25	2.39	32.12	38.58	6,596,568	12,198,986	25,727,245	57,990,353	69,648,601
	3.50	100,096,242	3.96	7.33	14.71	2.59	34.85	39.83	3,968,596	7,339,097	14,726,986	34,887,887	39,868,784
	4.00	29,839,772	4.47	8.28	15.55	2.65	39.34	42.11	1,335,308	2,469,376	4,641,344	11,738,678	12,565,010
INFERRED	0.00	150,481,703	2.64	4.88	12.08	1.67	23.18	32.70	3,967,300	7,336,699	18,175,180	34,876,487	49,203,709
	1.00	134,770,366	2.90	5.37	13.25	1.84	25.51	35.88	3,911,306	7,233,150	17,862,464	34,384,244	48,357,127
	2.00	124,717,186	2.99	5.54	13.46	1.88	26.31	36.44	3,733,035	6,903,475	16,787,557	32,817,068	45,447,146
	2.50	94,690,184	3.23	5.97	13.70	1.97	28.37	37.10	3,055,842	5,651,146	12,974,828	26,863,870	35,125,355
	3.00	55,899,862	3.56	6.59	14.25	2.15	31.31	38.58	1,990,874	3,681,709	7,967,240	17,501,748	21,568,850
	3.50	21,879,368	4.02	7.44	14.72	2.51	35.38	39.85	880,623	1,628,529	3,220,577	7,741,544	8,718,722
	4.00	9,143,043	4.45	8.23	15.46	2.63	39.13	41.86	406,939	752,549	1,413,798	3,577,392	3,827,423

Effective Date: November 6, 2013

The resources outlined in Table 14.4 and Table 14.5 reflects a material change in Area 1 and Area 2 from the resources estimated in the 2012 PEA. The measured plus indicated resources at 1% K_2O cut-off grade have increased by 8.5Mt for Area 1 and decreased by 66.0Mt for Area 2 when compared to the 2012 PEA estimates. The inferred resources at 1% K_2O cut-off grade have increased by 1.9Mt for Area 1 and decreased by 116.0Mt for Area 2 when compared to the historic 2012 PEA estimates. The material change is attributed to the inclusion of additional infill drill hole data, the decision to use only PRC drilling data in the geologic models to maintain a common and verifiable assay reporting standard and improvements in separating alunite mineralization from surrounding country rock using sulfate grade data.

The accuracy of resource and reserve estimates is, in part, a function of the quality and quantity of available data and of engineering and geological interpretation and judgment. Given the data available at the time this report was prepared, the estimates presented herein are considered reasonable. However, they should be accepted with the understanding that additional data and analysis available subsequent to the date of the estimates may necessitate revision. These revisions may be material. There is no guarantee that all or any part of the estimated resources or reserves will be recoverable.

15 MINERAL RESERVE ESTIMATES

15.1 APPROACH

Norwest has used the instrument document, the Canadian Institute of Mining, Metallurgy and Petroleum's CIM Standards on Mineral Resources and Reserves prepared by the CIM Standing Committee on Reserve Definitions, August 2000, as the basis for the classification, estimation and reporting of potash resources and reserves for the Blawn Mountain property.

Based on the geological model produced by Norwest, resource areas were developed for the Alunite deposit. Various surface mine plans were then developed in order to meet certain criteria related to project economics, grade, target production rates, etc. Of the various mine plans created, a base mining case/mine plan was selected as the basis of the reserve estimate, as reported in this document. The mine plan was developed by first applying various criteria in selecting the method and approach to mining, including:

- Annual production rate (ROM ore) to be constrained by processing capacities.
- Ore cut-off grades of approximately 3.5% K_2O (Area 1) and 3.25% K_2O (Area 2) were utilized during the mining phase (Years 2017 through 2041) of the project, and a declining grade ranging from approximately 3.5% K_2O to 2.5% K_2O during the stockpile reclaiming phase (Years 2041 through 2057) of the project.
- Maximize economic use of the resource.

Taking into consideration the above, mine plans were developed (see Section 16) that use standard surface mining 'truck-shovel' techniques to mine the deposit. During the mining phase, the ROM ore production rate will be approximately 10.6Mtpy meeting the ore grade cut-off criteria. During mining, ore encountered that falls below the cut-off criteria listed above (down to roughly 2.5%K_2O) will be stockpiled and processed later during the stockpile reclaiming phase of the project. Under these criteria the reserve base will provide for an approximate LOM of 40 years.

Mine development was scheduled using MineSight® software to generate a LOM schedule of waste and ore volumes. Applying equipment productivities to these volumes, equipment hours and fleet sizes were estimated, which in turn formed the basis of workforce demands and schedules leading ultimately to estimates of capital and operating costs (see Section 21). Taking into account commodity pricing and market conditions, ore processing capital and operating costs and mining capital and operating costs a cash flow of revenues and direct and indirect costs was developed. This ultimately led to an estimate of project economics and value (see Section 22). The mine plan, at a prefeasibility level of assurance, was found to be of positive economic value and forms the basis of mineral reserves reported here. Mineral reserves, by category, are summarized in Table 15.1.

TABLE 15.1 MINERAL RESERVES, BY CATEGORY

	Reserve Category		
	Proven ('000 tons)	**Probable ('000 tons)**	**Total**
Alunite Ore (ROM tons)	**136,254**	**289,540**	**425,794**
Ore (average K_2O (%) grade)	3.56	3.49	3.51
Ore (average K_2SO_4 (%) grade)	6.59	6.46	6.49
SOP (tons)	8,457	17,970	26,427
Sulfuric Acid (tons) @ 98% Purity	18,888	40,136	59,024

This estimate of resources and reserves was generated using the best information available concerning issues related to environmental, permitting, legal, title, taxation, socio-economics, marketing and political factors that could have a material influence on Norwest's findings. Norwest is not aware of any additional factors which may affect our reserves estimate.

The accuracy of resource and reserve estimates is, in part, a function of the quality and quantity of available data and of engineering and geological interpretation and judgment. Given the data available at the time this report was prepared, the estimates presented herein are considered reasonable. However, they should be accepted with the understanding that additional data and analysis available subsequent to the date of the estimates may necessitate revision. These revisions may be material. There is no guarantee that all or any part of the estimated resources or reserves will be recoverable.

16 MINING METHODS

Mining operations at Blawn Mountain will utilize conventional truck/shovel techniques to remove ore and waste material from the mining areas. Two mining areas (Areas 1 and 2) have been identified based on geologic exploration and modeling. Mine plans were previously developed for the Blawn Mountain Project during a PEA, completed by Norwest in November of 2012. These plans have been developed further during the PFS to account for updated geologic exploration and modeling, additional metallurgical testing and various trade-off studies performed during the economic analysis.

During development of the PFS, Norwest examined nine separate mining cases, referred to as Cases A through I. These cases examined various ROM production requirements, several ore grade cut-offs and different ore processing methods and equipment. Ultimately, a case emerged (Case I) displaying the best economics for the project and was chosen as the base mining case. The base mining case is described in further detail in the discussion below.

16.1 GENERAL MINING METHOD

Mining operations at the Blawn Mountain Project will begin in Area 1 and once the targeted ore has been removed from this area, operations will begin mining in Area 2. Ore and waste material will be removed using area and bench mining techniques. In general, operations will begin at the top of the ridges and move downward utilizing multiple 20 to 40ft vertical lifts. Several different mining faces will be utilized throughout the mine life to assist with mine scheduling efforts. Conventional truck/shovel mining techniques will be employed using a mid-sized hydraulic excavator and one front-end-loader (FEL) to load end-dump mining trucks. Prior to ore and waste removal, the material must be drilled and blasted.

Before mining operations commence, all salvageable growth-media material will be removed and placed in temporary storage areas. In general, it's anticipated that a limited amount of growth-media material will be encountered within the mining area footprints of Areas 1 and 2 as the terrain present within the mining areas is steep, sparsely vegetated and rocky.

Mine pre-development work involves an initial year of stockpile area and haul-road construction as well as two years of mine production ramp-up. The ore deposit at the Blawn Mountain Project lends itself well to surface bench mining techniques. Minimal waste material is encountered during operations. The average strip ratio (yd^3/ton ore) for the base mining case is 0.20:1.

16.2 MINING CASE I (BASE MINING CASE)

As mentioned above, Case I is considered the base mining case for the PFS and uses a 3.5% K_2O ore grade cut-off for Area 1 and a 3.25% K_2O cut-off for Area 2. These cut-off grades were utilized in the development of the pit shells for Area 1 and 2. The mining schedule is driven by the capacities of the processing equipment chosen (four calcining units) which established the ROM ore schedule at approximately 10.6Mtpy. Additionally, the base mining case recognized a leach recovery of 90%, 2% moisture and the SOP product was adjusted to account for 92.5% K_2SO_4. This case also assumes that a significant portion of low grade ore would be stockpiled during active mining operations. These stockpiles will later be reclaimed and processed. Four low grade ore stockpiles were developed, and are segregated by quarter grade increments ranging from approximately 3.50% K_2O to 2.50% K_2O. No mining recovery or dilution was assumed for this mine plan. This decision was made based on the fact that low grade stockpiles are being utilized and a minimal amount of waste exists in the deposit by virtue of its natural formation.

The mining sequence initiates in 2016 and extends through 2057. Construction of the stockpile pad and the Area 1 haul road begins in 2016. 2017 and 2018 are production ramp-up periods and full production is realized in 2019. Mining operations cease during 2041, but SOP is still being produced by reclaiming the low grade ore stockpiles that were placed during active mining operations. Mining occurs in Area 1 for approximately 8 years (2017 through 2024) and mining in Area 2 continues for an additional 17 years (2025 through 2041). Low grade ore stockpile reclamation begins near the end of 2041 and continues through 2057. The Case I schedule is presented in further detail below as Table 16.1.

TABLE 16.1 CASE I MINING SCHEDULE

Year	2016	2017	2018	2019	2020	2021-2025	2026-2030	2031-2035	2036-2040	2041-2050	2051-2057
Topsoil (Myd^3)	0.2	0.4	0.4	0.4	0.3	1.1	0.9	0.7	0.5	1.2	0.6
Waste (Myd^3)	2.6	0.6	0.9	1.8	5.1	21.7	15.3	21.9	16.5	2.0	0.0
Ore (Mt)	0.0	3.5	7.1	10.6	10.6	53.2	53.2	53.2	53.2	9.2	0.0
Al_2O_3 (%)	N/A	15.2	15.9	16.1	16.3	17.4	14.2	14.3	14.6	15.2	N/A
K_2O (%)	N/A	3.7	3.8	4.0	4.2	4.3	3.7	3.6	3.8	4.11	N/A
NA_2O (%)	N/A	0.3	0.4	0.4	0.3	0.4	0.3	0.2	0.2	0.2	N/A
*LGO (Mt) Stockpiling	0.0	3.6	5.2	9.3	10.8	26.7	40.7	46.0	28.4	1.0	0.0
*LGO (Mt) Reclaiming	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	97.2	74.5
Al_2O_3 (%)	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A	13.9	13.0
K_2O (%)	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A	3.2	2.7
NA_2O (%)	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A	0.3	0.3
SOP (Mt)	0.00	0.23	0.48	0.75	0.79	4.1	3.5	3.4	3.6	6.0	3.6

*Low Grade Ore

The mining progression maps for Case I are presented in Section 28 as Figures 16.1 through 16.10.

Growth-media material encountered during mining operations will be salvaged and stored in temporary stockpiles. It is expected that a limited amount of growth-media material will be encountered in the extents of the Area 1 and Area 2 mining areas. The terrain encountered in these areas is steep, rocky and sparsely vegetated. The areas where low grade ore stockpiles, out-of-pit waste piles, access and haul roads and the tailings holding area are assumed to contain salvageable growth-media. Norwest has assumed the area within the footprint of the tailings holding area will contain approximately four feet of salvageable material while the rest of the areas mentioned above will contain approximately one foot of material. These quantities have been accounted for in the mine plan and are shown above in Table 16.1. The mining and tailings staging maps in Section 28 display the topsoil pile locations.

Waste material generated during mining will be handled and stored in different ways. The waste in 2016, which is generated during construction of the stockpile pad, is used to construct the majority of the haul road needed to access Area 1. The majority of the waste material encountered from 2017 through 2024 is placed in an out-of-pit waste storage pile adjacent to Area 1. This pile is illustrated in Figures 16.1 through 16.10 in Section 28. Some waste material during this period is disposed in the Area 1 mining pit, but the area available for in-pit backfilling is limited. Once mining operations finish mining in Area 1, the mining transitions to Area 2 where all waste material encountered is disposed of in-pit. During the first several years of mining in Area 2, the waste material is placed in the Area 1 mining void. As soon as practical, Area 2 waste is placed in the Area 2 mining void. In-pit backfill progression is shown in Figures 16.1 through 16.10.

The production schedule and mining sequence uses an equipment fleet that adequately meets the needs of the mining operation. Two spreads or units of mining equipment are envisioned to remove ore and waste material from the mine. Table 16.2 presents the type, size and quantity of major mining equipment anticipated to be used. The quantity of equipment presented in Table 16.2 is at maximum levels. The quantity of end-dump trucks varies throughout the mine life as haul distances and mining areas change. Truck productivities were determined utilizing Talpac, a haulage and loading simulator software. The productivity assumptions used for the selected equipment is the excavator at 1,600yd^3/hr and the FEL at 1,000yd^3/hr. The density of the ore is 2.08t/yd^3.

The primary equipment will remove both ore and waste material from the mine. This equipment was selected as it provides flexibility to the support the various tasks encountered during mining operations. This equipment fleet is mobile, which will allow for easier relocation to support the mining schedule and will assist with the various site conditions encountered during mining.

TABLE 16.2 CASE I MAJOR MINING EQUIPMENT

Primary Equipment		
Excavator	$22yd^3$	1
FEL	$16yd^3$	1
End-Dump Truck	148t	19
Support Equipment		
Water Truck	16,000gal	2
Grader	297Hp	2
Dozer	580Hp	5
Drill	50,000lb	1

The major mining equipment will be supported by a fleet of smaller support equipment including pumps, light plants, lube and fuel trucks, mechanics trucks, pick-up trucks, etc.

The production requirements necessitate a 7 day per week schedule operating approximately 356 days per year. The remaining idle days account for holiday and other events such as training and inclement weather. The workforce requirements are detailed below in Table 16.3. Table 16.3 shows the average personnel required in increments throughout the LOM.

TABLE 16.3 CASE I AVERAGE WORKFORCE REQUIREMENTS

Category	2016-2020	2021-2025	2026-2030	2031-2040	2040-2057
Hourly Workers					
Excavator Operators	3	3	3	3	2
FEL Operators	2	3	3	3	1
Truck Drivers	23	33	51	52	18
Dozer Operators	11	13	13	14	5
Grader Operators	4	6	5	6	3
Drill Operators	3	3	3	3	0
Blasters	3	3	3	3	0
Mine Support	6	8	8	8	3
Maintenance Labor	18	23	26	26	12
Sub-Total Hourly	**73**	**95**	**115**	**118**	**44**
Management					
Exec., Staff, Tech	12	15	15	15	10
Maintenance	6	8	8	8	7
Operations	4	5	5	5	2
Total Employees	**95**	**123**	**128**	**146**	**63**

The aforementioned equipment and workforce are assumed to be provided by contract mining services. The operating and capital costs associated with the mining operations takes this assumption into account and is presented and discussed in more detail in Section 21 of this report.

16.3 GEOTECHNICAL CONSIDERATIONS

To date, limited geotechnical investigation and testing have been performed at the Blawn Mountain Project. An initial evaluation of slope stability for the proposed surface mine was performed in January 2013. This analysis was based on data collected from the site and core from two holes drilled on the property. This analysis was performed by Seegmiller International of Salt Lake City, Utah (Seegmiller). This analysis recommended mine slopes with overall angles of 45^{o}. Norwest utilized these constraints in the creation of the mining pit shells. Additional geotechnical testing will be conducted as part of the feasibility study to provide additional data for mine design.

The base mining case assumes that waste material will be disposed of in external waste piles and using in-pit backfilling techniques. The external and internal waste piles are constructed using angle of repose side slopes. The material that is placed in-pit will be deposited in areas where all ore has been extracted.

16.4 HYDROLOGICAL CONSIDERATIONS

As discussed previously in Section 5, the processing plant and mine areas are located near the head of two ephemeral drainages. Therefore, the runoff potential for these drainages is limited. Groundwater within the mine area is limited to the andesite flows within the volcanic rocks. Flows are limited to the range of 5 to 10gpm from groundwater wells completed in zones at the depth of the proposed mine pit.

16.5 WATER MANAGEMENT

Preliminary water management plans have been developed for surface water management, groundwater handling, and dust control for the pits and haul roads. These plans are discussed in more detail below.

Surface Water Drainage

The site is located at the head of two ephemeral drainages, so little drainage area exists above the mine area. It is envisioned that the majority of surface water at the site will be controlled with a series of adequately sized channels that will collect water from the areas within the mine disturbance. This water will be discharged to sumps and ponds for treatment and allowed time for sediment to settle. This captured water is expected to be limited in volume, but may be used for mine haul road watering. Consideration will also be given to utilizing water for ore processing.

Details of the surface water drainage plan are illustrated on Figure 16.11. The plan consists of seven sediment ponds with collection ditches around the outside of mining Areas 1 and 2. The drainage from the area between Areas 1 and 2 and the processing plant facilities will be collected by internal ditches and conveyed to the tailings basin located down gradient of the mining areas and process facilities.

Groundwater

To assess the potential for groundwater issues during mining, a series of groundwater monitoring wells were installed in the mine area. The zones monitored are the volcanic tuffs and andesite flows. These wells show that limited groundwater is found in these zones. The flows in the andesite flows of the volcanic rocks are in the range of 5 to 10gpm. The flows in the tuff deposits are in the range of 0 to 0.5gpm. These zones are at approximately the maximum depths of proposed pits. Therefore, pit intercepted groundwater is not considered to be significant and should not be a hindrance to mining. For the majority of mining activities, groundwater is not expected to be encountered. Only at the bottom of the anticipated mining pit is groundwater inflow expected. Any groundwater encountered during mining will be collected, as needed, in sumps in the pit floor. This water will then be pumped for in-pit use and dust control.

16.6 MINE RECLAMATION

Following completion of mining, mine facilities will be removed and reclaimed. All utility connections and foundations will either be buried or removed. The mine pits will be left in a stable configuration of no more than 45 degree slopes. Any external or internal waste piles will be regraded to a stable configuration and sloped to minimize safety hazards and erosion. The materials used to construct the roads and pads will be removed and the area graded to approximate the original contour. Salvaged soil will be spread over the disturbed areas per regulatory requirements and re-seeded. A seed mix, approved by UDOGM and SITLA will be applied as necessary. Fertilization or other soil amendments will be applied if necessary. Revegetation efforts will be monitored to assure that revegetation achieves 70% of the pre-mining vegetative ground cover. Surface water drainage control will be maintained until vegetation is reestablished to aid in vegetation success and prevent sediment loading.

Public safety and welfare will be considered and if necessary signage, berms, fences and/or barriers will be installed above highwalls or other areas. A final post mining topography was not developed as part of the PFS. Reclamation planning has been considered as part of ongoing mining operations (in-pit backfilling, low-grade ore stockpile processing, etc.). The anticipated costs for these efforts have been included in the economic model supporting the project.

17 RECOVERY METHODS

ROM ore produced by open-pit mining methods will be processed, as envisioned, by crushing, wet grinding, reduction roasting, extracting SOP by leaching the calcine with water, solid/liquid separation, evaporation of brine, crystallization, and drying and preparation of SOP product for markets. Provisions have been made in the process plant to conserve energy and water through treatment and reuse of effluents and disposal of residues in an environmentally sound manner.

The proposed combination of unit operations in processing alunite ore is based on test results of investigations completed in 2012 and 2013 in support of process optimization at HRI in Golden, CO, Pocock Industrial, Inc., in Salt Lake City, UT, and thermal processing systems modeling at FLSmidth in Bethlehem, PA.

Process design criteria for the major unit operations in the proposed integrated process plant complex are summarized below:

- ROM ore production rate is 10.4 x 10^6stpy at 2% moisture.
- Plant operation schedule is 330 days per year (dpy), 24 hours per day (hr/d).
- The nominal throughput capacity of the process plant is 1,313 short tons per hour (stph) and 1,500tph (maximum).
- Particle size of wet Grinding Circuit Cyclone overflow at P_{80} = 1000 µm to Filters.
- Moisture content of Filter Cake as feed to Dryer/Calciner/Roaster at 10%.
- Roasting temperature at 1022°F (550°C) and not to exceed 1112°F (600°C).
- Roaster off-gases are routed as feed to a 4,000tpd Sulfuric Acid Plant.
- Water Leaching of Calcine: 35% solids; 176°F; 60 minutes residence time; and 90% SOP extraction.
- Alumina/silicate leach residues pumped at 55% solids to impoundment.
- Water withdrawal from well field estimated at 6,500ac feet per year, or 4,000gpm.

ICPE scope of work at present time includes sections 100 through 1000.

17.1 AREA 100 - PRIMARY CRUSHING

Design Criteria

A.	Ore bulk density (lbs/ft^3)	95.0
B.	Solids (sp. gr.)	2.46
C.	Solids (%)	98.0
D.	Moisture (%)	2.0
E.	ROM top size (inches)	36
F.	Crusher design tonnage (tons/hr)	1,500

G. Crusher nominal tonnage (tons/hr) 1,313
H. Crushing Bond Work Index (kWh/ton) 7.06
I. Crushing Abrasion Index (grams) 0.40

General Requirements

A. Up to 1,500stph of ROM ore from the mine sized at minus 36in. will be delivered to the Gyratory Crusher Truck Dump Pocket. The access ramp to the Truck Dump Pocket will allow up to one 150t capacity ore hauling truck to unload the ore into the Truck Dump Pocket.
B. A Rock Breaker will be designed to provide complete coverage of the Truck Dump Pocket and provide adequate reach to reduce any oversized plus 24in. rocks that might otherwise plug the Truck Dump Pocket.
C. A Gyratory Crusher with a capacity of 1,500tph will reduce ROM ore size by approximately a 4 to 1 ratio; therefore, the crushed rock will be minus 6in. The Truck Dump Pocket will be sized at a minimum for two truckloads (150-ton each) which will provide for approximately 300t of live load. The ore from the Truck Dump Pocket feeds directly into the Gyratory Crusher. The Gyratory Crusher is sized to operate 7,920 hours per year. The crushed ore will be discharged into a Crusher Ore Pocket and will be reclaimed by means of two variable speed discharge Apron Feeders.
D. Two Apron Feeders (72in. wide x 2ft long) will empty the Crusher Ore Pocket, which will deliver the crushed ore to a Feeder Discharge Conveyor. Each Apron Feeder will be equipped with a variable speed drive (VSD), and the drive speeds will vary from approximately 40 to 50ft/minute. A Metal Detector installed on the Feeder Discharge Conveyor will shut down the conveyor and the systems upstream including the Gyratory Crusher if a significantly sized metal object is detected.
E. A Belt Magnet with a small belt drive will be mounted or suspended above the Feeder Discharge Conveyor head pulley, and magnetic metal scrap will be continuously removed and dumped into a Trash Bin. The Feeder Discharge Conveyor delivers the crushed ore to the Coarse Ore Stockpile Stacker Conveyor, which in turn discharges the ore onto the Coarse Ore Stockpile.
F. A Belt Scale located on the Stacker Conveyor provides tonnage information to operators and metallurgists for use in optimizing plant processes. A dust collection system will be designed to both minimize dust emissions into the atmosphere and recover fine product.
G. A 50-ton capacity overhead crane will be designed for maintenance of the Gyratory Crusher and auxiliary equipment at the primary crusher building.

17.2 AREA 200 – ORE STOCKPILE AND RECLAIM

Design Criteria

A.	Angle of Repose (degrees)	37
B.	Reclaim Drawdown (degrees)	47
C.	Conical Basement Slope (degrees)	15

General Requirements

A. A 48in. wide Coarse Ore Stockpile Stacker Conveyor, 1035ft long with an approximate rise of 106ft with a final slope of 6°, delivers the minus 6in. ore from the Gyratory Crusher circuit to the Coarse Ore Stockpile (COS). The COS will be conical in shape and will contain up to approximately 150,000t of ore for a four-day total ore supply to the grinding circuit. This COS will be approximately 292ft in diameter at the base and 110ft high and will be covered with a geodesic type dome to contain dust. The dome will be 150ft high and 350ft in diameter. The live ore storage available to the grinding circuit will be approximately 26 hours. It is assumed that a portion of the COS storage will reside at a 15º sloped depression below the base of the conical portion of the COS.

B. The COS will enter a reclaim system consisting of four 7ft wide x 13ft long Reclaim Chutes. The ore then discharges to each of the four tractor type 48in. wide x 16ft long Apron Feeders, which in turn will discharge to the 48in. wide x 368ft long Reclaim Conveyor delivering the ore to the Sag Mill Feed Conveyor. Typically, three of the four reclaim systems operate with one on standby. Each of the four reclaim systems is installed in a line at 65ft centers and is parallel to the Reclaim Conveyor located in a tunnel below the COS. The Sag Mill Feed Conveyor is 48in. wide x 222ft long with an approximate rise of 43ft and has a Belt Scale to record throughput, which is useful information for operators and metallurgists in optimizing plant processes.

17.3 AREA 300 - WET GRINDING AND CLASSIFICATION

Design Criteria

A.	SAG Milling Specific Energy (kWh/ton)	4.8
B.	SAG Mill Abrasion Index (grams)	0.43
C.	SAG Mill Feed Size (inches – P_{80})	6.0
D.	SAG Mill Product Size (mm – P_{80})	1.0
E.	SAG Mill Recirculating Load (%)	300

General Requirements

A. Preliminary grinding tests indicate that a SAG Mill will be the only stage of grinding. The SAG Mill Feed Conveyor delivers the stockpile ore (minus 6in.) to the Feed Chute of the

SAG Mill. The nominal ore feed rate is 1,313stph (dry) and the design feed rate is approximately 1.15 times nominal or 1,500stph (dry). In addition, sufficient process water is added at the feed chute to provide a 65% by weight solids density for the grind cycle. The SAG grinding ball charge is maintained at between 10% and 12% by volume in the SAG Mill. The SAG Mill discharge slurry flows to the SAG Mill Screen. The Screen undersize (minus 0.5in.) material flows to the Grinding Mill Sump while the Screen oversize material will discharge to the Pebble Screen Discharge Conveyor. It is currently anticipated that approximately 10% of the SAG Mill product or up to approximately 132stph (dry) will report to the pebble mill circuit.

B. The SAG Mill pebble circuit would consist of a SAG Mill discharge screen, Pebble Screen Conveyors, a Surge Bin and a Cone Crusher to reduce the pebble size and then recycle the crushed material to the SAG Mill feed chute. The steel ball chips would be deposited and disposed of in a Trash Bin.

C. The discharge slurry from the SAG Mill cascades into a Grinding Mill Sump. Process water is added to maintain the slurry pulp density at 55% solids. This sump with a sloped ($\approx 20^{o}$) bottom will be designed to provide a slurry retention time of 1.5 to 2 minutes.

D. Recycle grinding begins when one of two large Cyclone Feed Pumps, each with VSD delivers slurry to a Grinding Cyclone Cluster. The second Cyclone Feed Pump remains on standby. Cyclone Feed Pump discharge pressure is typically about 80ft to 100ft of total dynamic head (TDH) to deliver sufficient pressure maintained in the cyclone cluster header (i.e., about 10 to 12 psig) to begin the sizing separation process of the solid particles. A spiral and downward centrifugal force will cause the coarse fraction of the ore slurry called the underflow portion to accelerate in velocity as it descends toward the bottom at the cyclone cone periphery and ejects at the cone apex spout into a collection trough and launder discharging into the SAG Mill feed chute. The cyclone underflow has a density of approximately 70% solids and represents approximately 75 percent of the cyclone feed resulting in a SAG mill circulating load of 300%.

E. While the coarse fraction of the ore descends, the finer portion of the ore slurry ascends up near the center of the cone entering the cylindrical portion above the cone; this is considered the overflow portion. The cyclone overflow slurry will have a density of approximately 35 percent solids and will flow by gravity to the Grinding Circuit Thickener.

17.4 AREA 400 – SOLID/LIQUID SEPARATION

General Requirements

A. The overflow from the grinding circuit cyclones is concentrated in a Grinding Circuit Thickener and the Thickener Overflow is recycled to the Overflow Water Tank.

B. The underflow from the Grinding Circuit Thickener is dewatered in a battery of Belt Filters to produce Filter Cake as feed to the Dryer/Calciner/Roaster units.

C. Process design criteria for the Solid/Liquid Separation equipment for grinding circuit ROM ore product slurry to recover Filter Cake as feed to the Dryer/Calciner/Roaster circuit are as follows:
   1. Pulp density of slurry pumped to the cyclones = 56%.
   2. Pulp density of cyclone overflow = 35%.
   3. Particle size of cyclone overflow P_{80} = 1000 µm.
   4. Concentration of solids in thickener overflow = 500 ppm.
   5. Pulp density of thickener underflow = 45%.
   6. Solids content of Filter Cake = 90%.
   7. Moisture content of Filter Cake = 10%.

17.5 Area 500 – Drying And Calcination

General Requirements

A. Filter Cake containing alunite and inert solids at approximately 90% solids (10% moisture) is dried, dehydroxylated, and roasted to decompose the alunite. The calcine produced contains a mixture SOP (K_2SO_4) and alumina (Al_2O_3).
B. The energy required for the Flash Dryer circuit is provided by the exhaust gases from the Flash Calciner circuit.
C. A start-up air heater is used during the system start-up to bring the flash roaster up to the auto-ignition temperature of the fuel. A portion of the heat energy in the off-gases is used in steam generation.
D. The calcine particles are separated in a cyclone.
E. The calcine reports to the leaching circuit where it is leached with hot water to extract SOP.
F. Sulfur dioxide (SO_2) bearing off-gases from the Roaster are sent to the gas cleaning section of the Sulfuric Acid Plant and for further processing to recover 98% sulfuric acid (H_2SO_4). FLSmidth generally estimates the concentration of SO_2 and SO_3 in the flues gases from the thermal processing units at 9.44 vol. % as feed to the Acid Plant. This value is an estimate based on similar industry experience. No pilot plant testing has been completed by FLSmidth to-date.

Chemical Reactions

A. Filter Cake containing alunite and inert solids at approximately 90% solids (10% moisture) is dried, dehydroxylated, and roasted to decompose the alunite. The calcine produced contains a mixture SOP (K_2SO_4) and alumina (Al_2O_3).
B. When alunite is progressively heated to higher temperatures during drying and calcination, a number of temperature-dependent transitional phases occur. The stable assemblage at the termination of the calcination cycle comprises potassium sulfate and alumina with evolution of sulfur trioxide.

Drying of alunite (Equation 1):

$$K_2SO_4.Al_2(SO_4)_3{\cdot}2Al_2O_3.6H_2O(s) \rightarrow K_2SO_4.Al_2(SO_4)_3(s) + 2Al_2O_3(s) + 6H_2O(g)$$

Decomposition of aluminum sulfate (Equation 2):

$$K_2SO_4.Al_2(SO_4)_3(s) \rightarrow K_2SO_4(s) + Al_2O_3(s) + 3SO_3(g)$$

Overall Calcination Reaction (Equation 3):

$$K_2SO_4.Al_2(SO_4)_3{\cdot}2Al_2O_3.6H_2O(s) \rightarrow K_2SO_4(s) + 3Al_2O_3(s) + 6H_2O(g) + 3SO_3(g)$$

Thermal Processing System Components

A. Filter Cake containing alunite and inert solids at approximately 90% solids (10% moisture) is dried, dehydroxylated, and roasted to decompose the alunite. The calcine produced contains a mixture SOP (K_2SO_4) and alumina (Al_2O_3).

B. The Dryer/Calciner/Roaster system components offered by FLSmidth Minerals Pyrometallurgy group in Bethlehem, PA are as follows:

   1. Flash Dryer Circuit: The flash dryer portion of the system supports removal of the free moisture present in the alunite cake using the flash calciner off gases.
   2. Flash Calciner Circuit: The dried material from flash dryer is delivered to Flash Calciner for further heating to remove all the chemically bound water and to decompose the alunite to potassium and aluminum sulfates. The material produced is transferred to a Flash Roaster so that it serves to complete the aluminum sulfate decomposition.
   3. Flash Roaster Circuit: The pre-calcined material from the Flash Calciner is delivered to the Flash Roaster for further heating and to decompose all aluminum sulfate to aluminum oxide and SO_3. It also serves to convert most of the generated SO_3 to SO_2 at reduced conditions.
   4. Flash Roaster Off Gas Circuit: Air-to-gas heat exchanges cool the process gas leaving the Flash Roaster cyclone to a temperature level suitable for the downstream electrostatic precipitator (ESP). The ESP removes non-condensable particulate from the final system process off gas stream and an induced fan delivers excess gases to acid plant after some amount of gases are recycled back to the Flash Roaster.
   5. Cyclone Cooling Circuit: Three cyclones operating in series transfer heat from the Flash Roaster circuit product to the incoming ambient air stream.
   6. Excess Air from Cooling Circuit: The excess air from the cyclone cooling circuit is delivered back to the combustion circuit as pre-heated combustion/excess air.

C. Preliminary process design criteria for the Dryer/Calciner/Roaster Circuit, developed by FLSmidth as adapted from similar experience in the cement industry and based on desktop simulation and general modeling of available information, are as follows:
   7. Solids content of Filter Cake as feed to Dryer = 90%
   8. Moisture content of Filter Cake as feed to Dryer = 10%
   9. Number of thermal processing units = 4
   10. Name plate capacity of each thermal processing unit = 330tph
   11. Alunite decomposition reactions are carried out at 1022^{o}F (550^{o}F).
   12. The maximum temperature in the Roaster shall not to exceed 1112^{o}F (600^{o}C).
   13. The residence time in the Roaster is about 2 seconds.
   14. Longer retention times are allowed by the inclusion of fluidized bed holding vessels.

Temperature Dependence Of Forms Of Alumina

A. Filter Cake containing alunite and inert solids at approximately 90% solids (10% moisture) is dried, dehydroxylated, and roasted to decompose the alunite. The calcine produced contains a mixture SOP (K_2SO_4) and alumina (Al_2O_3).
B. The gamma-alumina phase occurs in a porous cubic structure, which can be leached with sodium hydroxide (NaOH). It reverts at high temperatures to the recalcitrant alpha form with hexagonal close-packed structures. The temperature limits on thermal processing are, therefore, required assure that the gamma-alumina crystals are the end product of alunite roasting operations.

17.6 Area 600 – Sulfuric Acid Plant

General Requirements

A. An estimated 3,390tpd of sulfuric acid (H_2SO_4) is manufactured at the Project site from sulfur dioxide (SO_2) produced by the decomposition of alunite during the thermal processing of ROM ore. FLSmidth estimates the concentration of SO_2 and SO_3 at 9.44 vol. % in the off gases as feed to the Acid Plant from the proposed four (4) Dryer/Calciner/Roaster systems, each with a throughput capacity of 330tph of Filter Cake at 90% solids content.
B. DuPont-MECS, a supplier of the Sulfuric Acid Plant, based on information developed by FLSmidth by modeling, estimates the composition of off-gases as feedstock to Acid Plant as given in Table 1.

TABLE 17.1 PRELIMINARY ESTIMATE OF COMPOSITION OF FEEDSTOCK TO THE SULFURIC ACID PLANT

Base Case		**84,993**			**Dilution Air**		
Component	Vol.%	WET NM^3/hr	DRY NM^3/hr	Vol.%	DRY NM^3/hr	DRY NM^3/hr	vol.%
SO_2	9.2	9,164	9,164	11.5	-	9,164	7.3
O_2	1.3	1,300	1,300	1.6	9,697.2	10,997	8.7
CO_2	0.0	0	0	0.0	-	0	0.0
N_2	69.0	68,979	68,979	86.8	36,572.8	105,552	84.0
H_2O	20.5	20,494	0	0.0		0	0.0
SO_3	0.0	0	0	0.0		0	0.0
Total	**100**	**99,937**	**79,443**	**100**		**125,713**	**100**

C. Two Sulfuric Acid Plants, each with a throughput capacity of 2,000tpd of concentrated acid are proposed to be constructed, each dedicated to process the off-gases from two lines of Roasters and with provisions for treating off-gases bypassing any of the Roasters being serviced.

Major **Unit Operations**

A. The proposed Sulfuric Acid Plant manufactures concentrated (98%) sulfuric acid by catalytic oxidation of SO_2 in the Roaster off-gases to SO_3, followed by absorption of SO_3 in 98% H_2SO_4. The block flow diagram illustrates the system components of a Sulfuric Acid Plant. The feedstock to the Acid Plant is de-dusted and cleaned Roaster off-gases at approximately 550°F (287°C) leaving the ESP in the gas cleaning circuit. The major unit operations are given below:

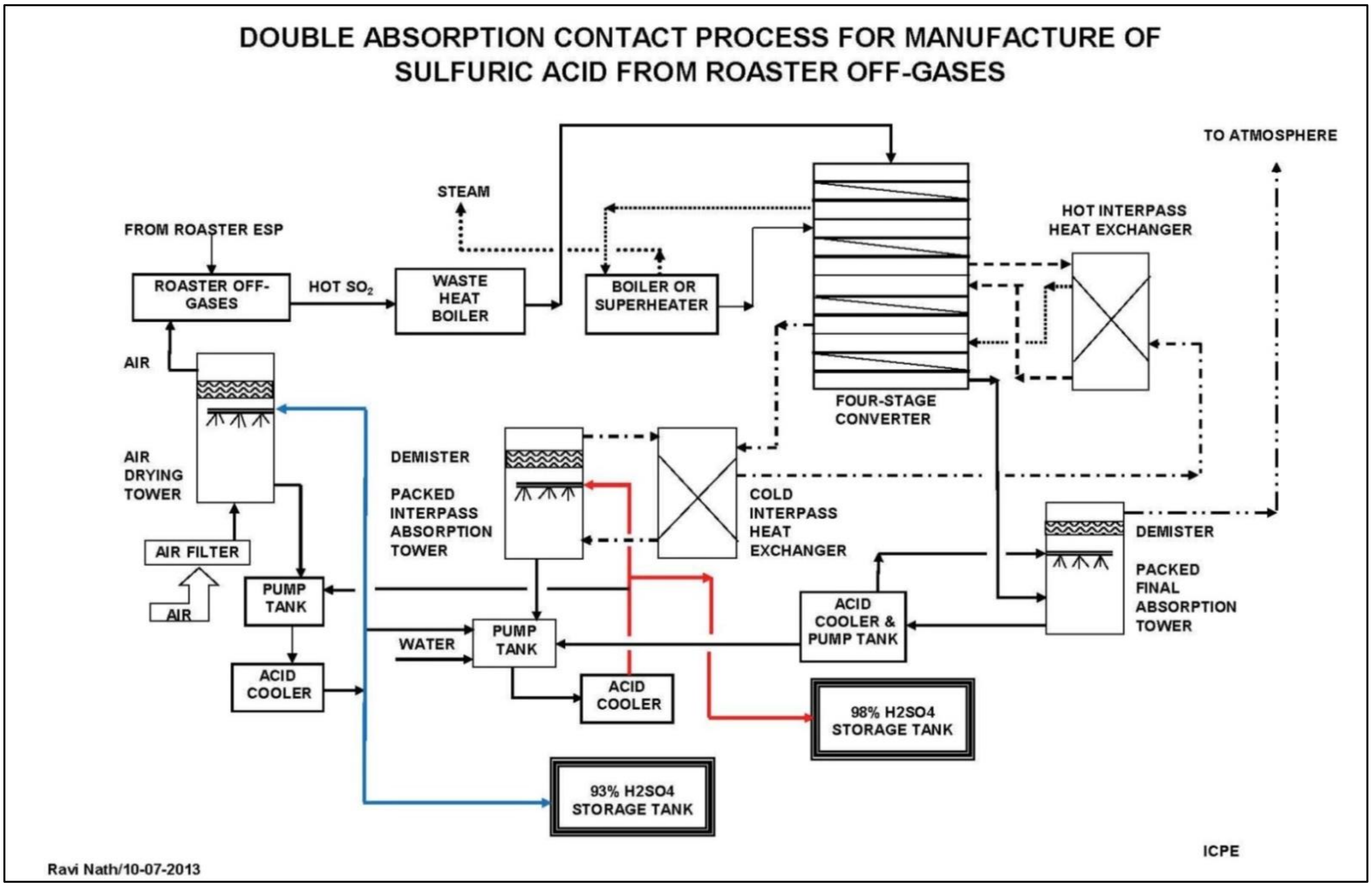
1. Step 1 – Cooling the SO_2-bearing and dust-laden Roaster off-gases in a mixing chamber, a bank of air to gas heat exchangers, tempering with air, and extracting the entrained dust in an ESP. The dust recovered is recycled to the water leach circuit. This step is completed in Area 500 at the discharge of the Roaster.
2. Step 2 – Conversion of SO_2 to SO_3 by reacting with oxygen is an exothermic reaction which is carried out in a four-stage reaction vessel with each stage consisting of a solid catalyst bed through which the gases are passed:

$$SO_2 + \tfrac{1}{2} O_2 \rightarrow SO_3 \quad \Delta H = -100\text{kJ/mol}$$

3. Step 3 – Absorption of SO_3 in a counter-current flow of concentrated sulfuric acid in a packed tower to produce more sulfuric acid according to the following reaction:

$$SO_3 + H_2O \rightarrow H_2SO_4 \quad \Delta H = -200\text{kJ/mol}$$

B. A block flow diagram of the Double Absorption Contact Process for manufacture of sulfuric acid from SO_2-bearing Roaster off-gases is shown below:



Emission Limits for Sulfuric Acid Plants

Emissions to the atmosphere from the Sulfuric Acid Plant at the Project site will be in compliance with the applicable Air Quality Permit requirements.

17.7 Area 700 – Calcine Leach And Solid/Liquid Separation

General Requirements

The calcine consists of inert siliceous materials in the ROM ore, and K_2SO_4 and Al_2O_3 produced by the decomposition of alunite during the roasting step. Extraction of K_2SO_4 by leaching the calcine with water as the solvent consists of dissolving potassium and sulfate values. The feed to the Leach Tanks consists of calcine discharged from the Roaster and SOP-bearing dust recovered in the ESPs in the Roaster circuit. The leachate or the potassium sulfate-rich solution is subsequently processed for the recovery of SOP in the Evaporator/Crystallizer system.

Calcine Leaching

A. The operating conditions of the water leach circuit are as described in US Patent 4,031,182 (K. W. Loost, July 21, 1977). Based on bench-scale test results at HRI, it is proposed that the calcine discharged from the Roaster will be leached with water at a pulp density of 35% solids. After dissolving approximately 90% of the K_2SO_4 in the calcine, the leach residue slurry at approximately 33% solids is filtered in a battery of Belt Filters and the Filtrate (brine) containing approximately 10.5 to 11.0% dissolved K_2SO_4 is pumped to the Evaporator/Crystallizer circuit. The Filter Cake at about 90% solids is washed with water and the washed Filter Cake consisting of inert solids, Al_2O_3 and undissolved K_2SO_4 is re-pulped and pumped at approximately 55% solids to the Tailings Impoundment.

B. Process design criteria for leaching the calcine with water, based on results of bench-scale investigations at HRI, are as follows:
   1. Water is the leaching medium (lixiviant)
   2. Average particle size of calcine = 1000 µm (1.2mm maximum)
   3. Specific gravity of calcine solids (assumed) = 2.7
   4. Solids content of slurry in the Leach Tanks = 35%
   5. Specific gravity of slurry at 35% solids = 1.283
   6. Average temperature of slurry during leaching = 176^{o}F (80^{o}C)
   7. Total residence time in Leach Circuit = 60 minutes
   8. K_2SO_4 recovery during leaching = 90%
   9. Recycling Filtrate for K_2SO_4 to Leach Tanks for increased concentration
   10. Number of Leach Tanks in series = 6
   11. Residence time in each Leach Tank = 10 minutes
   12. Leach Tanks fully baffled, covered, insulated, and steam-jacketed.

Solid/Liquid Separation

A. The calcine discharged from the Roaster is leached with water at a pulp density of 35% solids. After dissolving approximately 90% of the K_2SO_4 in the calcine, the percent solids content of the leach residue slurry is estimated at approximately 33%. As envisioned, the leached slurry is concentrated in a thickener, and thickener overflow is pumped to the Evaporator/Crystallizer. The underflow from the thickener is dewatered in a battery of Belt Filters and the Filtrate is pumped to the Evaporator/Crystallizer circuit. The Filter Cake at about 90% solids is washed with three-displacements.

B. Solid/Liquid separation is one of the most significant unit operations at the Project site because of its importance in energy conservation both in the Dryer/Calciner/Roaster and Evaporator/Crystallizer circuits, in water conservation through reuse of effluents, and its impact on the size and type of water and wastewater treatment facilities.

C. The leached slurry, at the end of the leach cycle, consists of inert solids originally present in the ROM ore, Al_2O_3 produced in the Roaster, an aqueous solution containing approximately

90% of the K_2SO_4 extracted during leaching, plus approximately 10% of unleached K_2SO_4 solids. Preliminary design criteria for the solid/liquid separation unit operations are assumed to be as follows:

1. Particle size of calciner feed solids P80 = 1000 µm (1.2mm maximum).
2. Solids content of Filter Cake as calciner feed = 90%.
3. Moisture content of Filter Cake as calciner feed = 10%.
4. Re-pulped leached solids Filter Cake = 55% solids.
5. Terminal density of settled solids in Tailings impoundment = 85% solids.

Water and Energy Conservation Measures

A. Measures being evaluated and to be validated in bench-scale tests for conserving water use and for reducing energy consumption in the operations are as follows:
   1. Achieving 90% solids Filter Cake with 10% moisture, on a sustained basis, as feed to the Dryer/Calciner/Roaster system for savings in natural gas consumption and for increasing the volume of water reclaimed for reuse.
   2. Increasing the pulp density of slurry from 35% to 40 or 50% solids during leaching to conserve water, reduce the size of Leach Tanks and downstream solid/liquid separation equipment.

17.8 AREA 800 - CRYSTALLIZATION

General Requirements

Once the dissolved K_2SO_4 content of the Filtrate from the Calcine Leach circuit has reached the desktop simulation "equilibrium concentration" estimated to be in the 10.5 to 11.0% range through recirculation of the filtrate, approximately 35% of the Filtrate will be pumped as feed to the Evaporator/Crystallizer circuit. The remaining 65% of the Filtrate and cake wash water, as required, will be recycled to quench the calcine solids discharged from the Roaster and to maintain the solids content of the slurry in the Leach Tanks at 35%.

Recovery of SOP crystalline product from the Filtrate containing K_2SO_4 in solution consists of an integrated Evaporator/Crystallizer system followed by separation of the SOP crystals from slurry, drying, sizing, and packaging of the product for shipment.

Quadruple Effect Evaporation and Crystallization

A. The quadruple effect evaporator removes the water (the solvent) in the feed (filtrate from the leach circuit) containing dissolved SOP by vaporizing a portion of the solvent in four stages ("effects") to produce a concentrated solution, thick liquor, or slurry. In each effect, heat is transferred to the solution or a suspension of the solid SOP crystals in the liquid.

B. In the quadruple effect evaporator system the vapor from one effect (vessel) is used as the heating medium for the next effect boiling at a lower pressure. The vessel in each effect provides most of the active volume in which the crystals that form within the vessel grow in size, and where the vapor is separated from the boiling liquid.
C. Each Crystallizer is a system complete with a heater, circulation piping, a tube-and-shell heat exchanger, and a slurry recirculation pump. The slurry is recirculated by the pump from the Crystallizer vessel, through the tube-side of the heater where it is heated, and returned to the Crystallizer vessel where boiling of the solution and the precipitation of SOP occurs.
D. The dewatering section of each fourth effect evaporator is a gravitational thickener (settling chamber). The thickened slurry discharged from the "thickener" is the feed to a centrifuge where the crystals are separated from the liquid in the slurry. The wet crystals are transported to the Fluid Bed Dryer/Cooler. The centrate from the centrifuge is collected in a tank and returned to the Crystallizers. The dried crystals are compacted and bagged for the markets.
E. Impurity build-up is controlled in the recirculated centrate by establishing a purge or "bleed" stream. Bench-scale investigations will be initiated to determine the amount of bleed required and to identify a method of treatment to recover the contained values. After treatment, the barren solution essentially containing sodium chloride will be routed to an evaporation pond.
F. Preliminary process design criteria for the Evaporator/Crystallizer circuit are summarized below:
   1. Design capacity of SOP product produced =750,000tpy
   2. Purity of SOP product produced = 92.5% K_2SO_4.
   3. K_2O content of product = 50%
   4. Simulation-based estimated flow rate of Filtrate to Evaporator/Crystallizer = 2,400gpm
   5. K_2SO_4 content of Filtrate evaporated = 85.7tph.
   6. Simulation-based estimated concentration of K_2SO_4 in Filtrate = 10.5 to 11.0%
   7. Estimated temperature of Filtrate (leaching at 176^{o}F) to Evaporator = 120^{o}F

17.9 AREA 900 – DRYING AND COMPACTION

General Requirements

A. SOP product in slurry form from the Evaporator/Crystallizer circuit is dewatered in a Centrifuge. The Centrifuge Cake, consisting of fine K_2SO_4 crystals, is dried in a dedicated, two-stage Fluid Bed Dryer/Cooler.
B. The dried product is fed to a Lump Breaker. The product from the Lump Breaker is transferred by the Lump Breaker Drag Conveyor, Bucket Elevator #1 and Drag Conveyor #1 to the Compaction Screens #1 and #2.
C. Dust generated during material transfer and/or during drying is collected using Wet Scrubbers and resulting slurry is recirculated back to the Wet Scrubber. Gas from the Wet Scrubber

discharges to the atmosphere through Stack #1. Slurry discharge from the Wet Scrubber is transferred to Mix Tanks 500-1200 A-D.

D. A Bucket Elevator is used to lift the dry, finely divided crystals of K_2SO_4 to Drag Conveyors 1 and 2 feeding the Compaction Screens.

E. Fine - 2mm, Compaction Screen undersize material flows by gravity to the Compactors. The Compactors are choke-fed so that any surplus of the feed by-passes the Compactor and reports to the Surge Bin. The Surge Bin maintains a constant flow through the Compactor and absorbs fluctuations during equipment start-up.

F. Compactors are used to compact the fine particles into flakes, which will then be crushed and screened to obtain the granular-grade product. Rejects are recycled back to the Compactors. The proposed Compactor is 1000mm x 1000mm (nominal) roll size unit fitted with twin-screw force feeders and equipped with one motor and planetary reducer driving each roll. The two large rolls compress the fines under pressure to the point they fuse into larger particles.

G. Flakes produced by the Compactor are broken in a gravity-fed double roll Flake Breaker located immediately beneath the Compactor. The maximum size of the particles delivered by the Flake Breaker is approximately 2mm.

H. Particles from the Flake Breaker are combined with crushed oversize material from the compaction screen, and then with product size, -4mm +2mm, material from the compaction screens, are sent to a conditioning drum to be treated with reagents to improve the quality of the granules.

I. Anti-caking additives are added to all products in the Conditioning Drum before transfer to a Polishing Screen. From the Polishing Screen, - 2mm powder is directed to the powder load out. Oversize, +4mm, material is sent to a Cage Paktor® and then combined with -4mm +2mm product to be conveyed to the standard granular product load out.

17.10 AREA 1000 – POWERHOUSE

General Requirements

The two Options for providing the electrical power and steam required for the processing facility are listed below. These Options are based on a Sulfuric Acid Plant as the preferred method of sulfur recovery. The final option will be selected based upon the completion of testing. In addition, both Options are based on the same sized natural gas pipeline.

A. **Option A Transmission Line:** Install a new 138kV transmission line from Rocky Mountain Power's Three Peaks Substation near Cedar City, Utah to the facility, and a 138/12.47 kV substation on the plant site. Install gas-fired boilers on the plant site to provide approximately 500,000lbs/hr of steam required for the processing facility in addition to the any potential steam produced by the Sulfuric Acid Plant. Currently no steam is available from the Sulfuric Acid Plant due to the low concentration of SO_2 in the feed system. Option A also includes an emergency gas-fired generator.

B. **Option B Powerhouse**: Install four gas turbine-generators (CT) on the plant site to generate the electrical power required. The fourth CT will be a spare. Three heat recovery steam generators (HRSG) will be installed on three of the four CTs to provide excess steam. The HRSGs will include gas duct burners and pollution control equipment to produce approximately 267,000lbs/hr steam each, and to control emissions within anticipated permit limits for NOx and CO.

Option A: Transmission Line

A. Transmission Line:
   1. The alternative to on-site power generation for the facility is to build a 138 kV power transmission line which ties into the existing electrical grid.
   2. Available Technology:
      a) The only option for routing an electrical power transmission line is above ground. Due to the remote location of the facility, the routing for an electrical power transmission line will be approximately 46 miles.

B. Boiler:
   1. The processing facility requires approximately 500,000lbs/hr of steam for use in the process. Currently no steam will be generated in the Sulfuric Acid Plant. The steam will be provided using packaged boilers fired with natural gas.
   2. Available Technology:
      a) Since high pressure, superheated steam is not required; the configuration of the boilers becomes simpler. To achieve the necessary steam production capacity from the boilers most economically the use of packaged boilers was evaluated.
      b) Three 200,000lbs/hr boilers are required for the processing facility and a fourth 200,000lbs/hr boiler will be used as a backup. Each of these 200,000 lbs/hr boilers will need to be shipped to the site in two or three modules, which will then be assembled on site.

C. Emergency Power:
   1. An emergency generating system is needed for the facility to mitigate the adverse effects of a sudden loss of power on the mineral processing system.
   2. Available Technology:
      a) Diesel and natural gas-fired units are available to drive an emergency generator. The diesel engines produce more air pollutants than the natural gas-fired units. The largest diesel engine generator sets are in the 4MW to 5MW range. GE offers an 8.5MW internal combustion engine generator set fueled by natural gas. Natural gas-fired turbine generators are available with much higher generating capacities. The emergency generating capacity needed for the facility is estimated to be in the 8MW to 10MW range.

Option B: Gas Turbine Generator

A. Gas Turbine:
   1. The processing facility requires approximately 90-105MW of electrical power. If an electrical power transmission line cannot be installed to supply the power, it will be necessary to generate the power on the site. The installation of three CTs at 42MW nameplate capacity de-rated to 31MW provides for 93MW of plant power. A fourth CT will provide back-up. The electrical demand is expected to be relatively constant throughout the year.
   2. Available Technology:
      a) The most commonly used energy sources for power generation are natural gas, oil, coal, and nuclear sources. The preferable energy sources are natural gas and oil and the availability of natural gas along with environmental permitting considerations favor the use of natural gas for the processing facility.

B. Heat Recovery Steam Generator:
   1. The processing facility requires approximately 500,000lbs/hr of steam for use in the process. This steam can be produced from recovering the heat from the exhaust of the gas turbines plus additional gas firing in a heat recovery steam generator (HRSG).
   2. Available Technology:
      a) There are several HRSG manufacturers. Since a HRSG is essentially a modified boiler, most large boiler manufacturers also produce HRSGs. The amount of steam produced only from the exhaust of the gas turbine is related to the temperature of the exhaust and the mass of combustion gases flowing through the exhaust. When more steam is desired than the turbine exhaust can produce by itself, additional gas-firing is required in the HRSG. With additional gas firing in the HRSG, each CT/HRSG combination will produce approximately 267,000lbs/hr of steam. With three CT/ HRSG units in service at once, the total steam produced will be approximately 801,000lbs/hr. This amount satisfies the established 500,000lbs/hr of steam required for the facility with a backup of 301,000lbs/hr. The remaining one CT will not be provided with an HRSG.

C. Emergency Power
   1. The redundancy supplied in the CT satisfies any emergency generation requirements.

17.11 AREA 1100 – TAILINGS AND RECLAIM

General Requirements

A. Discussed in more detail in Section 18.

17.12 AREA 1200 WATER DISTRIBUTION AND TREATMENT

General Requirements

Water distribution and treatment facilities at the Project site may be grouped into the following major types:

A. Water Distribution
   1. Header Tank – from wells
      – Fire Water
      – Day tank to feed water treatment and process water make-up
   2. Centrate recycle feed to water treatment (may be a closed loop in the crystallizer-leach circuit). (See 8.2.E.)
   3. Tailings repulp to tailings dam
   4. Crystallization condensate to water treatment make-up as required. The balance of the condensate continues to the leach circuit.

B. Water Treatment
   1. Potable Water Treatment
      – Treatment of groundwater pumped from wells to remove hardness (scale-forming constituents) and disinfection with chlorine for on-site consumptive uses.
   2. Boiler Water Treatment
      – Advanced methods of treatment of potable water, including removal of silica, addition of oxygen scavenging and corrosion control reagents, and/or demineralization for use as boiler feed water for steam generation and in the laboratory.
   3. Industrial Effluents Treatment
      – Treatment of a bleed stream from the Evaporator/Crystallizer circuit for build-up of sodium salts and/or impurity control and Thin Film Composite Membrane (TFCM)-type Reverse Osmosis (RO) system for reducing the total dissolved solids (TDS) in decant water from the Tailings Pond for reuse in the operations or reuse in the operations.

Preliminary Estimate of Water Demand

A. Preliminary design criteria for Water Distribution and Treatment processes are as follows:
   1. Potable water consumption = 100 gallons per capita day (gpcd).
   2. Filtered Gland Seal Water for pumps = 100gpm
   3. Treatment capacity of Sanitary Wastewater Treatment Plant = 60gpm, which consists of wastewater from the following sources:
   4. Steam Production: Steady state demand = 500,000lb/hr

5. Total blowdown for TDS control = 860gpm, which consists of:
   a. Boiler Plant blowdown = 70gpm
   b. Cooling Tower blowdown = 300gpm
   c. Glaserite build-up control blowdown = 250gpm
   d. Sulfuric Acid Plant blowdown = 150gpm
6. Total Treatment capacity of RO Plant for TDS control = 100gpm
7. The remaining (860gpm – 100gpm) = 760gpm blowdown containing elevated levels of TDS is available for use in the Grinding circuit and/or for repulping Tailings pumped to the Tailings Impoundment.
8. Treatment capacity of Ion Exchange (IX) Plant for demineralization = 165gpm

Water Quality Specifications

- The Blawn Mountain area has no perennial streams. The make-up water needed for mining and process plant will be sourced by extracting ground water from wells.
  A comprehensive well water sampling and analysis program is currently underway. The results from which will be used in the Feasibility Study and during detailed engineering of the Project facilities.
- American Society of Mechanical Engineers (ASME) guidelines for water quality in modern industrial water tube boilers are summarized in Table 17.2.

TABLE 17.2 ASME GUIDELINES FOR WATER QUALITY IN INDUSTRIAL WATER TUBE BOILERS

Boiler Feed Water*				Boiler Water		
Drum Pressure, psi	Iron, mg/L Fe	Copper, mg/L Cu	Total Hardness, mg/L $CaCO_3$	Silica, mg/L SiO_2	Total Alkalinity, mg/L $CaCO_3$	Specific Conductance, µS/cm (not neutralized)
0 – 300	0.100	0.050	0.300	150	700	7000
301 – 450	0.050	0.025	0.300	90	600	6000
451 – 600	0.030	0.020	0.200	40	500	5000
601 – 750	0.025	0.020	0.200	30	400	4000
751 – 900	0.020	0.015	0.100	20	300	3000
901 – 1000	0.020	0.015	0.050	8	200	2000
1001 – 1500	0.010	0.010	0.0	2	0	150
1501 – 2000	0.010	0.010	0.0	1	0	100

*Feed Water = Make-up water + Return condensate

- Table 17.3 summarizes water quality guidelines for Cooling Towers recommended by Marley SPX Cooling Technologies. A comprehensive water treatment program, including treatment with biocides to minimize the growth of bacteria (including Legionella Pneumophilia), is required in the operation and maintenance of evaporative-type cooling towers.

TABLE 17.3 COOLING TOWER WATER QUALITY GUIDELINES

Parameter	Guideline
pH	6.5 to 9.0
Maximum water temperature	120°F
Langelier saturation index	0 to 1.0
M-Alkalinity (Total alkalinity)	100 to 500 ppm as $CaCO_3$
Silica	150 ppm as SiO_2
Iron	3 ppm
Manganese	ppm
Oil and Grease	10 ppm for splash-filled towers None allowed for film-filled towers
Sulfides	1 ppm
Ammonia	50 ppm if copper alloys are present
Chlorine	1 ppm free residual intermittently (shock) Or 0.4 ppm continuously
Organic solvents	None allowed
Total Dissolved Solids (TDS)*	Over 5,000 ppm can affect thermal performance
Cations:	
Calcium	800 ppm as $CaCO_3$
Magnesium	Depends on pH and silica level
Sodium	No limit
Anions:	
Chlorides	750 ppm as NaCl, 455 ppm as Cl^-
Sulfates	800 ppm as $CaCO_3$
Nitrates	300 ppm (nutrient for bacteria)
Carbonates/Bicarbonates	300 ppm as $CaCO_3$ (maximum for wood)
Biological/Bacteria	
Film-Type Fills:	Aerobic bacteria Count:
MC75	Less than 10,000 CFU/mL
MCR 12/16	Less than 1,000,000 CFU/mL* when TSS<25 ppm and less than 100,000 CFU/mL when TSS>25 ppm
Total Suspended Solids (TSS)*	
Film-Type Fill	Less than 25 ppm
Splash-Type Fill	No specific limit
Other nutrients	For film fill, avoid fats, glycols, alcohols, sugars, and phosphates

***Notes:** CFU/mL = Colony-forming units per milliliter; TDS = Total Dissolved Solids; TSS = Total Suspended Solids

17.13 AREA 1300 WAREHOUSE

General Requirements

Discussed in more detail in Section 18.

17.14 AREA 1400 LOAD OUT

General Requirements

Discussed in more detail in Section 18.

Process flow diagrams for Areas 100-1200, mass balance information, as well as the General Arrangement drawings for the processing plant are presented as Figures 17.1 through 17.34 in Section 28.

18 PROJECT INFRASTRUCTURE

Project supporting infrastructure around the mine and processing plant site to support the operation was developed for the following major components:

- Project Access
- Water Supply
- Water Treatment
- Power Supply
- Natural Gas Supply
- Mine Roads and Pads
- Mine Support Buildings
- Surface Water Management System
- Tailings Management
- Tailings and Water Handling
- SOP Transportation
- Acid Transportation
- Miscellaneous Support.

Each of these is discussed in detail below.

18.1 PROJECT ACCESS

The Blawn Mountain Project will be accessed via existing county roads Revenue Basin and Willow Springs off of State Highway 21 (between milepost 53 and 54). These existing roads lie on land administered by both the BLM and SITLA. Beaver County has obtained a ROW grant from the BLM to upgrade these existing roads in order to accommodate traffic associated with the project. In addition to these road upgrades, an existing county road, referred to as a bypass road, will be relocated west of project area to provide a bypass for motorists and recreational users. The road upgrade project is divided into segments. The northern portion (10.5 miles) of the roadway travels through public lands administered by BLM while the southern portion (4.5 miles) travels through state owned lands administered by SITLA. The bypass road (3.3 miles) is within the SITLA lease area currently held by PRC.

The finished roadway surface will be chip-sealed gravel, but is approved for and can accommodate an asphalt pavement surface if needed in the future. The roadway will have two, 14ft lanes with 6ft shoulders (total width 40ft). Depending on the terrain, speeds of 30 to 50mph have been designed for. The bypass road will be a gravel road with a design speed of 25mph.

18.2 WATER SUPPLY

Water availability and sourcing is critical to the project. Consideration was given to developing the water supply from a series of sources. These sources may include the Wah Wah Valley, Pine Valley, and on-site wells to deep aquifers. For this study, it is assumed that the water supply will be provided from the Wah Wah Valley. The water system will consist of wells, surge pond/tanks, booster pumps, pipeline, storage tanks, and treatment plant.

18.3 DUST CONTROL

Managing impacts to air quality will be a concern for this large operation. The mine area is not expected to be a major dust producer. Within the processing area, the tailings area and the transportation components of the operation can impact air quality. Dust control will be handled by use of water sprays and contemporaneous reclamation.

18.4 POWER SUPPLY

Two options have been examined to supply electrical power to the site. The first option is on-site generation which will comprise four gas turbine-generators. The second option is to install a new 138kV transmission line from Rocky Mountain Power's Three Peaks Substation near Cedar City, Utah. Either of these options will satisfy the 90 - 105MW electric demand of the project. At present the economics supporting the PFS consider installing a new transmission line.

18.5 NATURAL GAS SUPPLY

A substantial amount of natural gas is required for the project. Gas will be required for several different areas of the processing facility, providing general heating needs to the facility and as previously mentioned may be used to generate on-site power. PRC has held discussions with Questar and Kern River to examine which supplier will meet their needs. The natural gas line will be owned and operated by the utility company.

18.6 MINE ROADS AND PADS

Access roads were developed from the intersection with the county bypass road to the mine site, continuing to the processing plant location. A guard shack will be placed at the intersection of the mine access road and the upgraded county road discussed above. The location and alignment of the site access road is illustrated on Figures 16.1 through 16.10 in Section 28. These figures also show the pad locations utilized by the mining facilities. Access roads are generally two lane roads with a shoulder on either side and a berm or ditch to support storm water flows.

18.7 MINE SUPPORT BUILDINGS

Mine support buildings were designed to support a year-round mining operation, and are specifically designed for intended purposes. The mine buildings include:

- Truck shop
- Warehouse
- Reagent warehouse
- Administration building
- Fuel depot
- Explosives storage
- Equipment ready-line
- Guard Shack.

The mine truck shop provides maintenance facilities for all mining equipment. The heavy equipment bays consist of four repair bays and one welding bay for maintenance of heavy equipment. An electrical shop, tool room and warehouse are included in the truck shop for electrical repairs, tool storage and small parts storage. On a mezzanine is office space for the workshop manager, shift foreman and planner. The mine truck shop is detailed in Figure 18.1.

The mine warehouse and reagent warehouse provide heated and covered storage for all material and supplies that must be protected from the elements. Additionally, the mine warehouse contains storage racks as well as offices for the superintendent and attendants. The location of these facilities makes them readily accessible for supply trucks hauling parts, supplies and other materials to the site. Figure 18.2 shows the layout of the mine warehouse.

The administration building is intended to provide office space for the bulk of the professional and engineering staff. An area within the administration building provides an area for mine employees (both male and female) to change prior to and after shifts, and to shower. It also provides lockers for storage of personal clothing and items, and restroom facilities. A medical clinic and ambulance bay will also reside in the administration building.

A mine fueling depot will be situated in a location convenient for mine operations. The fuel depot is used for providing fuel and top-up fluids for the haul trucks and rubber tired heavy equipment. Rapid fueling systems with high capacity pumps will be used to ensure minimum time and fuel loss at the site.

The equipment ready line will allow the mine trucks in waiting during cold temperatures to be ready for operation. Power outlets will be installed at the ready line for supply power for the engine block heaters. The hot line will also include area lighting and a compressed air line to assist in preparing the haul trucks for operation. Ten slots are provided at the ready line location, considering that many of the trucks will be in use around the clock.

18.8 SURFACE WATER MANAGEMENT SYSTEM

Storm water controls will be located downstream of all surface disturbances. These controls will consist of diversion ditches, sediment ponds, outlet control structures, and a combination tailings pile/runoff containment structure and a settlement pond. The sediment ponds and diversion ditches will collect and clarify water from the periphery of the site. The drainage from the plant and facilities area will drain to the area of the tailings pile/collection pond, where it will be collected and clarified. An illustration of the surface water management system layout is depicted in Figure 18.3 showing worst case conditions. Costs for these structures are included in the economic analysis.

18.9 TAILINGS MANAGEMENT

As ore is processed, tailings are produced requiring storage. Tailings associated with ore processing will be pumped from the processing plant to the tailings storage area. Based on the assumed gradation of this material, it is anticipated the tailings will be coarse grained sand that will be freely draining. Over the life of the project, approximately 185M yd^3 of tailings will be produced. Table 18.1 illustrates the amount of tailings released over the life of the project. Tailings will likely be deposited in a pile/beach that will drain to a collection pond which will include an earthen dam.

TABLE 18.1 TAILINGS STORAGE VOLUMES (CUMULATIVE)

Year	1	2	3	10	20	30	40
Tailings (Myd^3)	4.6	9.2	13.9	46.2	92.4	138.5	184.7

The collection pond will be constructed by a contractor prior to commencement of operations. The material used to construct the dam for the pond will come from a borrow area within the footprint of the tailings storage area. The tailings will be pumped from the processing plant facility to the tailings pile as slurry. The consistency of the tailings as it leaves the plant will be approximately 55% solids. The ultimate settlement percentage solids after decant of the tailings will be roughly 85%. The decant water will be collected via a dewatering tower and a gravity discharge line running under/through the collection pond dam to a settlement pond located below the toe of the collection dam. The clarified water will then be pumped to the process water storage tank. The tailings decant water collection system is described in later sections. Figures 18.4 through 18.8 show the collection pond/settlement ponds and tailings pile at various stages throughout the LOM. The temporary growth media storage pile is also illustrated on these drawings.

Tailings/Seepage/Collection Pond

A collection pond will be constructed to collect the drainage from the tailings and the runoff from the site and process facilities, mine haul roads and associated areas. The process facilities will produce approximately 500 to 1530tph of tailings (3.2 to 9.7Myd^3/yr). The estimated drainage rate will range from 710gpm at the beginning of operations to 2160gpm during the anticipated maximum tailings output.

This water will be collected by the collection pond dam and drained by a decant tower and conveyed under the collection pond embankment. To maximize the water re-use, this water will be collected in a settlement pond where it will be allowed to clarify and then pumped back up to the facilities area to the process water tank. The settlement pond is located downstream of the collection pond. A pump house will be located adjacent to the settlement pond and a pipeline will convey the water to the facilities area.

Tailings Characteristics

Based on the current understanding of the anticipated process, it is assumed, for the PFS that the tailings discharge will consist of non-hazardous, non-toxic materials. As such, no liner for the tailings facility has been proposed.

Tailings Reclamation

The majority of the tailings area is anticipated to be reclaimed concurrently as operations progress. As soon as practical, area behind the discharge line will be graded and growth media material spread over the surface which will then be seeded with an approved seed mix. This process will be repeated until the full development of the tailings pile is completed. The costs for these activities were considered part of the normal operation which is included in the economic analysis.

Final reclamation will include the removal of the collection dam, reestablishment of the drainage channel, grading of the outslope of the tailings to a stable slope (3H:1V or shallower), spreading of growth media material and seeding with an approved seed mix, covering the slope surface with erosion control blankets. Fertilization or other soil amendments will be applied if necessary. Outslopes of the rock fill tailings dam will be left as a rock slope.

18.10 Product Transportation

Processing ore from the Blawn Mountain Project results in two saleable products, SOP and sulfuric acid. Several different product transportation options were considered in the development of the PFS; including over the road trucking, a sulfuric acid pipeline and rail haulage. Several economic trade-off studies were performed comparing the various transportation options and ultimately, economics favored the on-site rail option. This option was utilized in the PFS economic analysis and details are described below.

A short line rail system will be used to transport SOP and sulfuric acid to market. Several rail line routes are currently being evaluated, and it is anticipated that 25 to 35 miles of rail line will be required. A future siting study will be completed to determine the exact route of the rail line. The proposed layout of the rail loop adjacent to the project area is presented on Figure 18.9.

SOP Rail Loop, Storage, and Loading near Processing Plant Site

SOP will be conveyed from the process plant to a nearby concrete storage dome. This dome will provide approximately 20,000t of storage. A storage reclaim system will directly feed the rail loadout equipment. Based on anticipated production rates, it is estimated that one unit train for SOP will be required every 4.5 days. The rail siding is sized to support a minimum 100 car unit train and three locomotives. Two separate loading sidings were developed to provide a dedicated loading area for sulfuric acid on one line (discussed below) and SOP loading on the second line. Figure 18.10 shows the flowsheet for the SOP storage and loading systems.

Acid Rail Loop, Storage, and Loading near Processing Plant Site

A short acid pipeline will be used to transfer acid from the processing plant to on-site acid storage and then to a rail loading area adjacent to the processing plant. Figure 18.11 illustrates the flowsheet for the acid storage system and loading system near the processing plant. This option includes 2.5Mgal of acid storage in four 625,000gal storage tanks. Dual pumping systems, schedule 60 stainless steel 316L pipe, secondary containment, and four rail loading systems are included in the design. Anticipated acid production will fill a unit train every 2.5 days.

18.11 MISCELLANEOUS SUPPORT INFRASTRUCTURE

Some support infrastructure has been addressed with allowances and alternative solutions common for this level of study. Norwest has made reasonable assumptions to provide for the required services listed below:

- Helipad
- Sanitary waste treatment
- Fire fighting
- Mancamp to be utilized during construction
- Site transportation.

Ambulance service is available to either of two small hospitals within driving distance of the operation in either Milford or Beaver, UT. Life flight for critical care or trauma care can occur either to Dixie Medical Center, in St. George or to Salt Lake City. For these reasons, Norwest has not included for a helipad on site. Norwest has preliminarily sized a waste water treatment facility septic tank and absorption leach field to support the long term project operation. The design flow is sufficient to support the full time mine staffing plan. Sludge will be removed and disposed of in a nearby municipal facility in Milford. For design purposes, a 50,000gpd design flow was selected, considering a 100gal per person per day effluent flow.

Fire suppression efforts on the mine and processing plant will be handled by a fire water network and handheld portable units. Fire water will consist of a ring network of fire hydrants around the mine and processing plant site. Based on the footprint of the planned process facilities, the anticipated fire flow requirements were estimated. Using these estimates, the total fire water storage was determined. This water storage volume was included in the header tank.

A 250 person mancamp is planned to support the mine and project development. Several locations were originally considered for the camp location, between on site, nearby private land, and close to town. Ultimately it was decided that the mancamp will be located on the western edge of Milford, on Highway 21 headed toward the mine to engage the local community and businesses with the mine development and allow the construction crew access to Milford. A typical all-inclusive mancamp, with catering options, lodging, activity center, as well as ablution and showering facilities was specified for the PFS. Transportation from site to town for the majority of the construction crews will be via high capacity crew busses. In this manner, the traffic between Milford and the mine site will be minimized and interruptions will be minimized.

19 MARKETS AND CONTRACTS

Norwest understands that PRC has conducted at least three market and price studies relative to this project and several other studies and forecasts are publicly available. Norwest has not independently reviewed this information. The information presented below was provided to Norwest from PRC.

19.1 MARKETS

Agricultural Fertilizer Marker

Fertilizers, a large component of the global chemicals industry, consist of essential plant nutrients that are applied to farmed crops in order to achieve favourable quality and yield. They replace the nutrients that crops remove from the soil, thereby sustaining the quality of crops, and are considered the most effective means for growers to increase yield. Growers adjust the types, quantities and proportions of fertilizer to apply depending on crop, soil quality, weather conditions, regional farming practices and fertilizer and crop prices. In 2006, over 45% of total global fertilizer applications were used in the production of corn, wheat and rice in almost equal proportions and 17% was used in the production of fruits and vegetables.

The key components of agricultural fertilizers are nitrogen (anhydrous ammonia and urea), phosphates (ammonium phosphates and superphosphates derived from phosphate rock), and potassium (potash). When blended, these three nutrients (nitrogen, phosphates and potassium) are known as "**NPK**". Most virgin soils contain adequate amounts of NPK to allow farmers to produce average crops. The agricultural cycle of growing and harvesting crops depletes soil of NPK, which needs to be reapplied in consistent ratios if the soil is to remain fertile. Global demand for all NPK nutrients is estimated to have been approximately 173 million tonnes (100% nutrient basis) in 2010/11 and is projected to reach 190 million tonnes by 2015/16.

Sulfur has gained increased attention in the fertilizer industry over the past several years due to the realization that crops were becoming sulfur deficient. Sulfur is necessary for the production of protein, fostering activity and the development of enzymes and vitamins. In addition to these benefits, sulfur also improves root growth and seed production, aids in the creation of chlorophyll and increases resistance to cold temperatures. The addition of sulfur to fertilizer (usually in the form of ammonium sulfate or SOP) creates a blend known as "**NPKS**".

Fertilizer Demand Drivers

Global fertilizer demand is expected to increase greatly in the coming years due to world population growth accompanied by decreasing arable land per capita, changes in diet and growth in alternative fuels which use crops as feedstock. By 2030, the average cost of key crops is projected to increase by 120-180% due to an increasing population's rising demand for meat and agricultural products. Much of this increase in demand is expected to originate from developing countries, where nearly all of the increase in world population is expected to occur. The increasing amount of food (both plant and protein based) needed to feed this growing population must be produced from a shrinking arable land base per capita as rural land is developed to accommodate the larger populace. Arable land per capita has decreased by an average of 1.5% per year from 1961-2009.

China and India, the world's most populous countries, are expected to reach population levels of 1.3 billion and 1.7 billion, respectively, by 2050 and both countries have a rapidly expanding middle class with improving nutrition and dietary preferences. By 2050, the global population is expected to be approximately 9.3 billion compared to 7.0 billion in 2011.

In addition to a projected increase in population, the global per capita income of developing countries is expected to increase from $4,800 per year in 2007 to $11,000 per year by 2030, according to the World Bank. Due to dietary changes, there is a strong correlation between income growth and increases in food consumption. As incomes grow, consumers move away from staple and/or traditional foods and begin consuming premium foodstuffs, such as fruits and vegetables, as well as adopting diversified protein rich diets. Increased protein consumption results in increased demand for grain and other animal feed.

Fertilizer demand is also being driven by the growing ethanol and bio-fuel industry. Many countries are attempting to supplement the consumption of fossil fuels with ethanol and bio-fuels. This is due to the high price of oil in recent years as well as increasing government support for alternative sources of energy. Global ethanol production is expected to grow from approximately 106 billion litres in 2011 to 155 billion litres in 2020.

Potash Market Overview

Potash is a generic term that refers to a group of potassium-bearing minerals, naturally occurring potassium salts and the products produced from those salts. The term potash arose through the traditional practice of producing potassium carbonate, needed for making soap, by leaching wood ashes in large iron pots. The ash-like crystalline residue left in the large iron pots was called "potash".

Potash is a plant's main source of potassium; one of the three primary nutrients essential for plant growth. Plants depend on potassium for water retention as well as the production, transport and accumulation of sugar. Potassium also supports plant hardiness and resistance to water-stress and disease. Plants deficient in potassium are less resistant to pests and disease, and have poor size, shape, color, taste, and reduced shelf life.

The amount of potassium contained in potash varies, thus, the industry has established a common standard of measurement in defining a product's potassium content in terms of equivalent percentages of potassium oxide ("**K_2O**"). Sylvite, potassium chloride ("**KCl**" or MOP), is the most commonly used global potash source given its high solubility characteristics and high potassium content of approximately 61% K_2O. Approximately 95% of all potash produced is used for agricultural fertilizer, mostly in the form of MOP. Potash has no commercially viable substitute as a potassium fertilizer source. Other bases of potash consumption are potassium bearing chemicals, detergents, ceramics, pharmaceuticals, water conditioner and de-icing salt.

In its processed state, potash appears as a granular mineral of varying sizes and ranges in colour from white to reddish, depending on the presence of trace elements, notably iron, which remains after processing. There are four principal potash grades: granular, coarse, standard and soluble. Granular, coarse, and standard grades are differentiated principally by the size of the particles, with granular being the largest and standard the smallest.

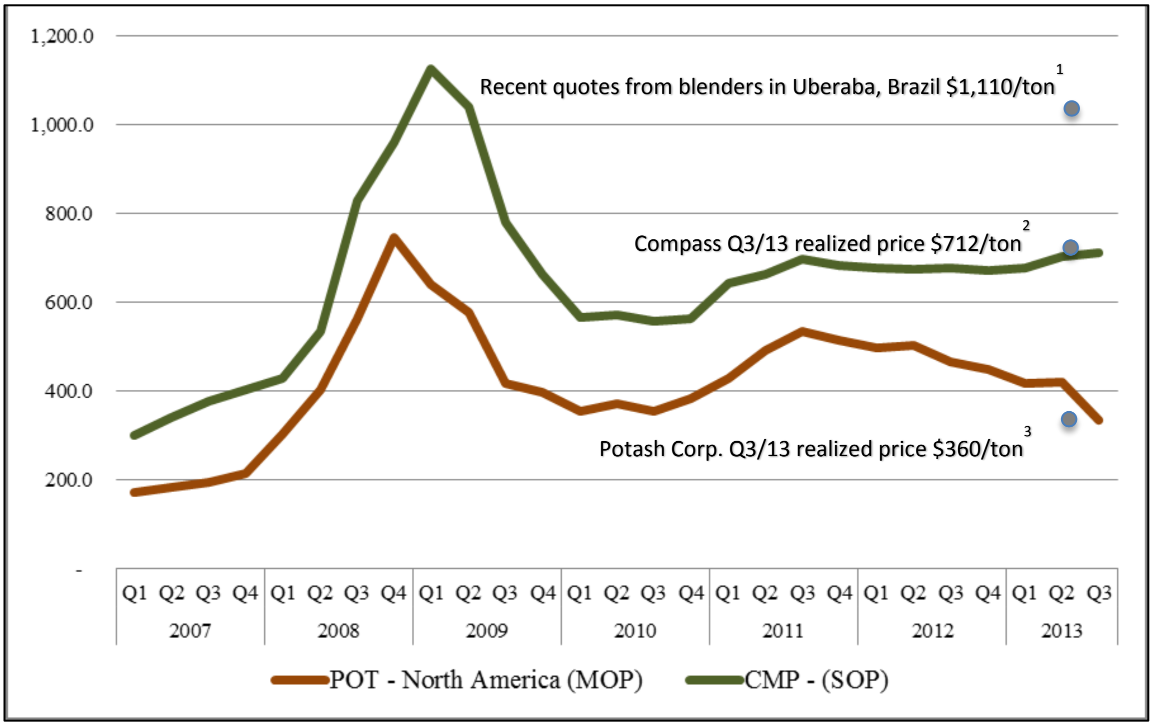
Potash Demand Drivers

At the macro level, Potash demand depends on the demand for fertilizer, which is expected to increase materially in the coming years due to world population growth, decreasing arable land per capita, changes in diet, and growth in alternative fuels that use crops as feedstock. At the micro level the demand is based on total planted crop acreage, crop mix, fertilizer application rates and farmer economics. Each of these factors is affected by current and projected stocks and prices of crops, governmental agricultural policies, improvements in efficiency, fertilizer application and weather.

As crop prices rise, farmers have a greater ability to invest in increasing the yield of their land, which in turn necessitates greater fertilizer and potash use. Global demand is expected to grow to 74.6 million tonnes in 2020 from approximately 55 million tonnes in 2011, an average annual increase of 4% per year.

CRU forecasts MOP prices averaging US $450 per tonne from 2013 through 2020. Historically, the price of MOP has been important to all potash producers, as prices of the alternate chloride-fee potash products have been based off the price of MOP. Over the past 18 months, however, the premium spread of SOP over MOP has expanded significantly in the United States, as recent drops in the price of MOP have not translated to similar declines in the price of SOP. This largely reflects the lack of supply of SOP into the United States. Chart 19.1 illustrates the premium SOP commands over MOP for the last several years.

CHART 19.1 AVERAGE REALIZED MOP AND SOP PRICE IN NORTH AMERICA



Source: [1]Verde Potash Presentation, November 2013, [2]Compass Minerals Q3 2013 Report, [3]Potash Corp.Q3 2013 Report.

Potash Supply Environment

Although potash is a commonly occurring mineral, the geological conditions needed for its economic extraction can only be found in a few regions of the world. Moreover, many of the currently unexploited potash reserves that have been identified are located in politically unstable and/or remote locations such as the Congo, Ethiopia, Laos, Russia, Thailand, Uzbekistan and the Rio Colorado region of Argentina, which generally require significant new infrastructure to be built to facilitate mine development. Greenfield construction in such locations can be prohibitively expensive. Canada accounts for almost 46% of the world's potash reserves. The majority of remaining reserves are found in Russia, Belarus and Brazil. There are currently more than 12 countries with active capacity for MOP production, with Canada, Russia and Belarus accounting for almost two thirds of current capacity. Together with Germany, Israel and China, these countries account for approximately 88% of the world's MOP production.

The scarcity of economically mineable potash deposits and increasing capital costs to develop and process new deposits have resulted in barriers to entry and a high degree of concentration among the leading producers. According to CRU, the world potash industry had capacity of 66.9 million tonnes in 2011, from more than 45 operations in 13 countries. However, 94% of global capacity is held by 10 companies and only a limited number of these potash producers are expected to increase annual production. Also, the current potash supply is being produced by aging mining infrastructure; worldwide, 85% of facilities are more than 25 years old. Consequently, additional capacity will be required to meet the forecast increases in demand.

Industry production capacity is stated in terms of nameplate capacity, which is higher than average sustainable production levels. KCl production nameplate capacity was estimated at 65.5 million tonnes in 2010 with operating rates of 78% versus 49% in 2009. Operating rates are expected to reach 86% by 2020, even taking into account an increase in capacity of a further 21.0 million tonnes. While operating rates of 80-85% are considered sustainable, higher rates are not considered sustainable due to equipment availability, utilization and reliability, and uncertainty in recovery for extracted mineral resources. The projected operating rates of 86% could result in a tight supply and demand condition, which would justify capacity expansion. The majority of production capacity expected to be brought online prior to 2015 is owned by established producers, with the largest component consisting of brownfield expansion by Potash Corp. Global capacity, including new projects and expansions, is projected at 86.5 million tonnes of KCl by 2020 with operating rates of 86%.

Chloride-Free Potassium Fertilizer – Potassium Sulfate (SOP) Market Overview

While MOP is widely used in all types of farming, the chloride ion within it can be detrimental to some plants, especially fruits and vegetables. Chloride-free potashes are priced at a premium to MOP, due to supply constraints, high production costs and because chloride-free fertilizers are typically used on chloride sensitive corps that sell at sufficiently high prices to absorb the incremental fertilizer cost. There are three chloride-free forms of potash: SOP (potassium sulfate); potassium magnesium sulfate; and potassium nitrate. Of these chloride-free alternatives, SOP has the highest potassium content, at 50% K_2O, and contains sulfur, another key nutrient.

SOP is the most commonly used alternative to MOP, comprising approximately 10% of total potash consumption. SOP can be used in every application that MOP is used and is preferred in many circumstances as it enhances yield and quality, extends shelf life of produce and improves taste. MOP rarely outperforms SOP in terms of crop quality. SOP performs particularly well with crops that have a low tolerance to the chloride in MOP (such as fruits, vegetables, beans, nuts, potatoes, tea, tobacco and turf grass) and in arid, saline and heavily cultivated soils. SOP can be sold as a standard powder or as a premium granular or soluble grade product.

SOP Specific Demand Drivers

As the most commonly used alternative to MOP when the presence of chloride ions is undesirable, SOP sells at a premium over MOP, largely due to the yield, quality, shelf life and taste benefits of using SOP, combined with the scarcity of primary SOP production and the high cost to produce SOP through secondary production methods. For the period 2001 – 2010, SOP has commanded an average premium of 47% over MOP, ranging from 38% to 61%.

The majority of SOP use is for "premium" agriculture, broadly defined as all crops other than cereals and oil seeds. It is commonly used for growing tobacco, tea, fruits, vegetables, nuts, and turf grass. The major consumers (and producers) of SOP are China, Europe and the United States.

According to the International Fertilizer Association, 47% of potash fertilizer consumption, equivalent to 17-18Mtpy of K_2O, is used for "premium" crops where SOP is preferred over MOP. In 2011, consumption of SOP was approximately 6.0Mtpy or 11% of total potash consumption. The low volume of SOP consumption relative to its potential market demand is primarily a result of the scarcity of SOP supply.

China (with a population of 1.35 billion and the world's largest producer of tobacco, fruits and vegetables) is the largest consumer of SOP, accounting for approximately 42% of global demand. Over the past 20 years, the demand for SOP in China has experienced significant growth, from approximately 0.5Mtpy in the early 1990s to 1.9 million tonnes in 2011. According to CRU, SOP demand growth in China is expected to increase by 6.3% per annum through to year 2020. According to CRU, SOP consumption in Europe is approximately 1Mtpy and staying relatively flat through year 2020.

In the United States, specialty crops best suited for using SOP account for approximately 40% of total crop revenues. SOP consumption in the United States is approximately 350,000 tonnes per annum, with over 50% of this demand coming from California. California is the number one state in cash farm receipts, growing 58% of US-grown non-citrus fruits, nuts and vegetables and 100% of US almond production (the second highest commodity in value after milk).

SOP consumption in the US is largely constrained by the availability of product. Compass Minerals is currently the only significant producer of SOP in the US with sales volume of approximately 300,000 tonnes of SOP in 2012. CRU forecasts annual growth in US SOP demand of 0.5% through to 2020, however this forecast is based on the assumption that SOP markets remain supply-constrained. Compass' realized SOP sale price for the third quarter of 2013 was US$712 per metric tonne, a 130% premium over the average realized MOP price for the same period as reported by Potash Corporation, highlighting the fact that the US SOP market is very much supply constrained.

India and Brazil represent markets of large SOP demand growth potential. SOP consumption by these countries has typically been constrained by a lack of product available for export by SOP producing regions.

Brazil is the world's fourth largest consumer of nutrients used in fertilizer production, consuming 28.3 million tonnes of fertilizer in 2011. SOP consumption in Brazil is, however very low, at only 32,000, SOP in 2011 (or 0.3% of total potash consumption), notwithstanding the fact that it is the largest grower of various premium crops, such as citrus fruits and tobacco in the world and has 20% of its planted land dedicated to premium crops.

India, with a population of 1.2 billion, consumed only 50,000 tonnes of SOP in 2011, representing less than 1% of its overall potash consumption, even though it is the second largest tea-growing and tobacco-growing nation in the world.

Based on an analysis by Serecon discussing sulfate of potash and the agronomic premium over muriate of potash ("Serecon's Agronomic Premium Study"), there are three distinct market segments for SOP. These markets are classified as: the "Premium SOP Market"; the "Value SOP Market"; and the "Commodity SOP Market".

Crops in the Premium SOP Market are highly chloride-sensitive and include tree nuts, tobacco, berries, and various fruits and vegetables. Due to the significant detrimental quality and yield effects of chloride, MOP is rarely, if ever, applied to these crops; accordingly, the MOP price is not a strong indicator of what growers may be willing to pay for SOP. For this segment, growers typically weigh incremental revenue improvements through better quality and yield by applying SOP versus simply not applying SOP. Serecon's Agronomic Premium Study concluded that "a 40% percent premium is more than justified by the difference in crop yield for a number of crops that exhibit high sensitivity to chloride".

Crops in the Value SOP Market are less sensitive to chloride than Premium SOP Market crops, however will experience lower yield and quality using MOP instead of SOP. Crops in the Value SOP Market comprise certain fruits and vegetables, such as apples, broccoli, and cucumbers. Producers of these crops typically compare the incremental cost of SOP over MOP against the revenue benefits from improved quality and yield by applying SOP. Serecon's Agronomic Premium Study concluded that growers in this segment would typically consider switching to SOP from MOP at "a premium of 10 to 30 percent over the price of MOP", depending on a number of factors, including specific crops, soil conditions, etc.

The Commodity SOP Market largely comprises row crops, such as corn, canola, cotton and rice. These crops are not chloride sensitive, however, they do require sulfur. Typically, MOP, ammonium sulfate and urea is used in fertilizer blends to meet the fertilizer needs of these crops. Serecon's Agronomic Premium Study concluded that "SOP should be expected to trade at up to a 12% premium to MOP as a substitute for MOP and ammonium sulfate in Commodity SOP Market crops.

SOP Specific Supply Environment

Total worldwide production capacity of SOP in 2011 was 6.7Mtpy, with approximately 48% located in China, 37% in continental Europe, 10% in the Americas, with the remaining in other countries. The largest SOP producer is SDIC Xinjiang Luobupo Potash Co., Ltd. ("**Luobupo**") in Western China. The international export market is supplied mainly by two European companies, K+S and Tessenderlo Chemie NV.

Production from China in 2011 was approximately 1.9 million tonnes. Luobupo represented approximately 1.2M tonnes of this production, with approximately 0.7 million tonnes coming from high-cost furnaces used in the Mannheim Process that convert MOP to SOP ("**Mannheim Furnaces**"). Most production from Europe comes from a conversion process similar to Mannheim Furnaces. In the United States, Compass Minerals produces approximately 280,000 tonnes through an evaporation process in the Great Salt Lake in Utah and approximately 36,000 tonnes in Saskatchewan Canada.

Of total worldwide SOP production, approximately 15% comes from evaporation of salt lakes, with the remaining 85% coming from Mannheim Furnaces or similar conversion processes. Opportunities for new production from evaporation are limited and the high cost of Mannheim Furnaces and similar conversion processes makes additional investment to increase capacity using these technologies unlikely.

In recent years, increases in Chinese demand has largely been absorbed through additional supply coming from the newly-commissioned Luobupo project. While the average premium of SOP over MOP has decreased slightly in China, many of the high-cost Mannheim Furnaces remain competitive due to their location and quality/consistency of product compared with Luobupo. Exports from China are negligible and are not expected to increase in the future, as domestic production is subject to export tariffs.

Outside of China, no new major sources of SOP have been developed since the 1990s. The profitability of producers using Mannheim Furnaces or similar conversion processes is largely dependent on the price of MOP, sulfur and energy. Several high cost operations using Mannheim Furnaces or similar conversion processes have curtailed operations due to their high operating cost and the limited market for hydrochloric acid produced as a by-product of the Mannheim Furnace production process.

There are a handful of new primary SOP development projects currently under evaluation, although production from these projects is not expected until 2017 at the earliest.

PRC SOP Marketing Strategy

PRC intends to target its SOP production primarily at the US market. Studies undertaken by PRC, based on utilization rates for certain key crops, indicates that there is potential incremental SOP demand of up to 485,000 tonnes per year in the United States, a certain portion of which can be supplied to the Premium SOP Market, thereby maintaining existing premiums within that market.

PRC intends to focus its marketing efforts on the economic value of SOP to growers of Premium Value Crops. California will be a key market given its large base agricultural base of premium crops. Florida will be another key target. Currently, 100,000 tonnes per annum of SOP is imported into Florida from Europe and Chile, which can be displaced.

Outside of the United States, China and Brazil are seen as primary targets.

Using CRU's 6.3% annual growth rate assumption, China's demand for SOP will grow by about approximately 120,000 tonnes every year or by a total of 840,000 tonnes by 2020. Given the low quality of its SOP, Luobupo's target market is largely the Value SOP Market, where it realizes a relatively low premium to MOP. Luobupo is not able to effectively compete in the Premium SOP Market, where higher premiums can be attained. Current Mannheim Furnace producers in China have plans to expand capacity, providing evidence that a market for higher quality, higher priced SOP for the Premium SOP Market exists in the country. These expansion plans will be challenging, however, given the need for Mannheim Furnace producers to find a market for the by-product hydrochloric acid.

Studies undertaken by PRC, based on utilization rates for certain key crops, indicates that there is potential incremental SOP demand of up to 7 million tonnes per year in China, a certain portion of which can be supplied to the Premium SOP Market. This suggests a large potential SOP market in China that is more than capable of absorbing PRC's production.

As noted earlier, Brazil is significantly underserved in SOP supply. Studies undertaken by PRC, based on utilization rates for certain key crops, indicates that potential incremental SOP demand in Brazil could be as high as 2 million tonnes per year, primarily in the Premium SOP Markets, including fruits, nuts and coffee. The Company is pursuing offtake, marketing and distribution arrangements for its SOP in each of these regions.

Sulfuric Acid Market Overview

Domestic production of sulfuric acid in the United States is currently about 32 million tonnes per year with domestic demand of approximately 34 million tonnes per year. The majority of this supply deficit is met through imports from Canada, Western Europe, Japan and South Korea, primarily through the Gulf of Mexico. The supply deficit in the US is expected to increase to approximately 2.4 million tonnes per year by 2015.

Today, a significant amount of fertilizer production in the US requires sulfuric acid. Overall, the greatest annual demand for sulfuric acid in the US is derived from the production of phosphoric acid at 23 million tonnes, followed by industrial uses at 6.7 million tonnes, ammonium sulphate production at 2.2 million tonnes and copper production at 1.6 million tonnes.

Currently, there are seven smelters producing sulfuric acid in the US. Of the three smelters in the Eastern US, one is expected to shut-down by 2015, reducing annual production of sulfuric acid by 245,000 tonnes. There are four smelters in the Western US, two of which are located in Arizona, which together supply approximately 1.1 million tonnes of sulfuric acid per year. The largest smelter, owned by Kennecott Utah Copper, is in Garfield, Utah and supplies about 975,000 tonnes of smelter acid per year to phosphoric acid production and copper oxide production facilities in the region and elsewhere.

PRC Sulfuric Acid Marketing Strategy

The US has a significant amount of fertilizer production that requires sulfuric acid consumption. The majority of US acid consumption in the eastern portion of the US, is largely supplied by dedicated sulfur-burning sulfuric acid plants. The western US sulfuric acid consumption is primarily in copper oxide leach and the fertilizer and industrial sectors. Current demand by phosphate producers is estimated to be approximately 3.0 million tonnes per year, while copper producers in the region consume approximately 2.1 million tonnes per year.

According to CRU, by 2015 Arizona is expected to consume up to 1.8 million tonnes per year of sulfuric acid, primarily for the production of copper. The majority of this acid is expected to support production at Freeport McMoRan's five operating copper mines – Morenci, Bagdad, Sierrita, Safford and Miamia. Freeport also has the Tyrone and Chino copper mines in New Mexico. Of their operating mines, the both the Chino and Morenci mines are undergoing expansion. Freeport utilises dedicated sulfur-burning sulfuric acid plants at its operations, however the Morenci mine does not have dedicated sulfur-burning sulfuric acid plants and it is expected to require an additional 1.0Mtpy of acid post 2015, after expansion.

In Utah, following the April 10, 2013 land slide at Kennecott's Bingham Canyon mine there is serious uncertainly among acid consumers about the long-term availability of acid from this smelter. The smelter initially cut sulfuric acid production levels by half, to approximately 40,000 tonnes per month from a typical production level of 80,000 tonnes per month. This uncertainly does not bode well for consumers, as it results in them having to increasingly rely on trade acid which is prohibitively more expensive than the contracted supply.

PRC will market its sulfuric acid to existing mines and mines under development providing a long-term safe supply of acid and much needed diversification of acid supply for existing consumers.

PRC is targeting phosphate processing facilities in Wyoming and Idaho and metals processing facilities, such as copper, gold, vanadium and uranium, in Utah, Nevada and Arizona.

PRC is also targeting mine expansions and mines under development, where the only source of acid is trade acid, and where the price is typically determined by the Gulf Coast import price plus a transportation cost to the mine site. The company believes that the potential exists to place the majority of its acid with these mines.

PRC offers two key benefits to potential acid consumers. First, a long-term security of supply of acid that could make mine development and expansions viable under long term fixed price contracts. Second, PRC can offer price certainty over the life of a long-term contract, with a fixed price linked to either the price of acid or to the price of the commodity the potential customer is processing; thereby reducing the input commodity risk to the consumer.

Based on information from CRU, PRC's analysis indicates that Utah trade acid is currently selling for between \$150 - \$175/tonne delivered.

PRC Alumina Marketing Strategy

The leaching process utilized to produce SOP from the Blawn Mountain Project leaves behind an alumina rich material which, with beneficiation may have the potential to be used as a substitute to bauxite as a feedstock into a Bayer alumina production facility as well as a raw material feed for low temperature refineries. Additionally, the alumina rich material may meet specifications for feed material in the production of ceramic proppants in North America.

While it is presently uncertain if any of these potential upsides can be realized, PRC is in the process of additional metallurgical testing programs to prove or disprove the economic viability of these options. Because of this uncertainty, the economics supporting the PFS summarized in this technical report do not included revenue from the sale of alumina rich material.

The Metallurgical testing PRC has completed to date has shown that the alumina contained in this material is soluble in high temperature caustic solutions. PRC has had preliminary discussions with various parties on off-take arrangements for its alumina rich material, pending quality analysis

The primary market for the metallurgical use of bauxite material is China. The potential alumina from the Blawn Mountain Project would not only enter a supply constrained market in China but also provide an environmentally superior product compared to typical bauxite. Unlike Indonesia, and other politically challenging sources of bauxite, alumina refiners in China are facing pressures from Indonesian and Chinese policy guidelines leading to an upward trend in bauxite prices. Increased tariffs from Indonesian and Indian bauxite exports, rising capital costs of new bauxite projects with higher unit costs of production, higher import prices from regions outside of the Pacific, and increasing underlying infrastructure costs as mines are developed inland further from the coast all contribute to higher delivered cost of bauxite. According to China customs statistics, the delivered cost of bauxite in China from new sources like Guyana and Guinea was over $80/tonne in first quarter of 2013.

Sea transportation is one of the major components of the landed cost of bauxite in China. Blawn Mountain enjoys a transportation advantage compared to other projects located in Guinea and Brazil. The Blawn Mountain project is approximately 5,700 nautical miles from Shandong province via the Port of Los Angeles whereas distance from Boke Guinea and Trombetas Brazil are approximately 11,000 nautical miles.

The other major advantage of the material, that would be particularly attractive to Chinese consumers, is the lack of red mud typically associated with the production of alumina from traditional bauxite. Red mud is a solid waste product of the Bayer process in the production of alumina and a typical plant produces one to two times as much red mud as alumina

19.2 PRICING

PRC utilized pricing information generated from CRU to forecast future revenues for the project. Table 19.1 details this pricing information. Table 19.1 presents the pricing information used for the initial nine years of the project, forecasted prices through the remainder of the project after this time period have been held constant at 2020 prices.

TABLE 19.1 FORECASTED PRICES

Product		2016	2017	2018	2019	2020
Potassium Chloride (MOP)	US$/t	419.0	428.8	442.0	491.0	534.0
Potassium Sulfate (SOP)	US$/t	566.0	578.0	597.0	663.0	721.0
Sulfuric Acid CFR US Gulf	US$/t	79.0	82.0	82.0	82.0	82.0

Potash Price Environment

Potash prices refer to the delivered cost of potash and are usually negotiated as delivery contracts (typically free-on-board, or "**FOB**") between suppliers and their customers. By their nature, such contracts contain terms which vary depending on the suppliers' and consumers' geographic locations. The contracts are typically structured as either large, fixed-price sales contracts, monthly contracts with annual minimums or "spot" purchases. Premium potash grades include coarse and granular material with larger particle sizes (1-4mm) and soluble industrial products generally purer than 98% KCl. Granular and coarse potash is generally priced at a premium.

Potash prices were relatively stable prior to 2007. In 2007, escalating prices helped by fertilizer demand from Brazil, China, India and the United States and pushed producer "operating rates" over 90%. The resulting supply/demand imbalance caused MOP prices to continue rising in 2008, reaching a peak of approximately US$900 per tonne (more than three times the highest price realized between 2001 and 2007). High commodity prices declined sharply due to the economic crisis in 2008 and falling grain prices, in conjunction with high fertilizer costs, squeezed growers' margins. This margin squeeze, and a tightening of global credit, negatively affected growers' ability-to-pay for fertilizer, which led to a decline in potash demand.

By early 2010, MOP spot prices were back to the January 2008 level of US$340 per tonne – down from an average of US$630 per tonne in 2009. MOP prices started to increase along with overall commodity prices in 2010, reaching approximately US$350 per tonne.

The recent announcement by Uralkali to pull out of their marketing joint venture with Belaruskali caused a fair amount of near-term uncertainty over prices of MOP. The joint marketing company BPC (Belarusian Potash Company) accounted for approximately 40% of the global potash market and given the Uralkali announcement, buyers are expecting lower prices as a result of this decision and are holding off on committing to long term contracts as reported by major North American potash producers. In the most recent financial report, Potash Corporation reported an average realized price of $307 per tonne of MOP. The average realized price for North America was higher at $333 per tonne.

SOP Specific Price Environment

Non-chloride potash prices have generally followed the path of MOP prices in recent years, although a wider price gap is starting to develop between these two products as demonstrated by the average realized prices by Compass Minerals and Potash Corporation in their second quarter, 2013 financial results. Although the recent decision by Uralkali has negatively affected the global MOP prices, the supply constrained US SOP market served primarily by Compass Minerals has not observed any price sensitivity for SOP. SOP has historically attracted a premium to the MOP price due to its limited availability, better quality and the fact that 85% of SOP production comes from manufacturing processes that use MOP as a primary feedstock. Virtually all producers that use Mannheim Furnaces for production of SOP acquire MOP from third parties at market prices. SOP has commanded an average premium of 47% from 2001 to 2010 and from 2006 to 2011 has ranged between 30% and 61%.

According to CRU, based on the current capacity of potash producers and the announced additional production planned by existing mines and new entrants, potash prices should remain at relatively robust levels for the next several years. For the period from 2016 to 2020, CRU forecasts MOP prices in the range of US$419 and US$534 per tonne (FOB Vancouver/Portland) and SOP prices between US$566 and US$721 per tonne (CIF northwest Europe). Inflationary pressures in the energy and materials industries could also provide support for continued high potash prices. Compass Minerals reported an average realized price of $712 per tonne of SOP in their third quarter 2013 financial results, a 98% premium over the realized price ofr MOP for the same period

US Mountain West Sulfur Pricing

Based on PRC discussions with potential customers for sulfuric acid, the price for acid is based on Gulf Coast benchmarks adjusted for transportation costs to the local customer. Based on a logistics study by Savage Industries, the freight cost to transport from these markets into the Mountain West ranges between $60/t and $80/t. From this basis discounts are then given to larger volume customers. The CRU forecast prices for Sulfuric acid are noted in Table 19.1.

PRC has a memorandum of understanding (MOU) in place with an existing Utah mine that would result in a $150/t price for sulfuric acid based on the CRU number noted above and the Savage transportation numbers. The MOU does include a discount from the current market price and would result in the placement of 20% of the acid produced by the Blawn Mountain Project. The offtake customer indicated that they would be willing to accept the equivalent amount of elemental sulfur from PRC should the decision be made to produce sulfur rather than sulfuric acid. The company has also agreed to provide a sink for the acid, whereby any unsold acid could be used at their facility, eliminating the possibility of a shutdown due to lack of storage facility for the acid at the plant.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 REGULATORY ENVIRONMENT

State Regulations

Mining and processing operations in the US must comply with all applicable federal and state regulations. Utah has primacy over major environmental laws applicable to the project including mining, air and water permitting. Based on Norwest's review of the current planned operations the following state regulations may apply to the Blawn Mountain Project:

- Title R647 Natural Resources; Oil, Gas and Mining; Non-Coal
- Title R317 Environmental Quality, Water Quality
- Title R307 Environmental Quality, Air Quality
- Title R655 Natural Resources, Water Rights
- Title R657 Natural Resources, Wildlife Resources
- Title R850 School and Institutional Trust Lands Administration.

Mining operations must obtain proper permits and approvals and submit proper reclamation surety prior to mine start-up per the R647 state regulations. The Blawn Mountain Project will require permits and approvals from Utah Division of Oil, Gas and Mining (UDOGM).

Utah's Water Quality Act and associated regulations prohibits discharging water or depositing wastes or other substances without prior approval or authorizations. Based on current mining plans, the Blawn Mountain Project may require a permit from the Utah Division of Water Quality (UDWQ) to manage their storm water and other process water.

The Blawn Mountain Project will have a septic tank and an absorption leach field to handle sanitary waste. Disposal of sanitary waste will require a wastewater treatment facility permit by the UDWQ. Water will be treated onsite to provide potable drinking water to staff. Engineering plans and specifications for all public drinking water projects must be approved by the Division of Drinking Water prior to construction. Plans and specifications must be prepared by a Utah-licensed professional engineer.

Based on current mining plans, the Blawn Mountain Project will require a groundwater discharge permit from the UDWQ to address potential groundwater impacts. In addition, a construction permit may be required for the certain structures that represent a potential source to groundwater impacts.

All sources that emit a regulated pollutant are required to obtain an Approval Order from the Utah Division of Air Quality (UDAQ) prior to construction. Based on specific thresholds the Blawn Mountain Project will require a Prevention of Significant Deterioration permit from the UDAQ. This permit must be obtained prior to the start of construction of the source. One year after operations begin an operating permit, referred to as Title V permit, will need to be obtained by the UDAQ.

The Division of Water Rights is the state agency that regulates the appropriation and distribution of water in Utah. A “water right” is a right to divert (remove from its natural source) and beneficially use water. PRC must obtain the necessary water rights to support the project.

State wildlife sensitive species are managed by the Utah Division of Wildlife Resources (UDWR). Field surveys have determined there are no state sensitive wildlife species within the Blawn Mountain Project area.

Federal Regulations

When BLM lands (minerals or surface) are impacted, BLM approvals are required per the Federal Land Policy and Management Act. Federal actions requiring permits or approvals trigger compliance with the National Environmental Policy Act (NEPA). The level of scrutiny a project receives is based upon the BLM’s discretion, the significance of impacts to the environment, and/or the public’s interest or involvement. The mine and processing plant are located on SITLA controlled mineral and surface land and is not expected to require a federal action.

Wetlands and waters of the US (WoUS), defined under the Clean Water Act, are regulated by the US Army Corps of Engineers (ACOE) regardless of land ownership. Based on current mine design and baseline surveys (delineation study), it appears that there are no wetlands, WoUS, or other ACOE jurisdictional waters which will be impacted by the Blawn Mountain Project so an ACOE permit should not be required.

The Endangered Species Act of 1973 was passed by Congress in order to protect and recover endangered species and their habitat. Site specific surveys completed for the Blawn Mountain Project area did not identify any threatened, endangered, or candidate species or potential habitat.

County Regulations

Beaver County’s ordinances require mining operations to obtain a Conditional Use Permit (CUP) prior to construction. PRC has been in close coordination with Beaver County and county officials have responded with strong support for the project. In addition to the CUP, PRC will be required to obtain other ancillary permits and approvals from the county in accordance with the county’s ordinances.

20.2 HISTORICAL ENVIRONMENTAL STUDIES

In the 1970s ESI proposed to develop the Blawn Mountain Project resource. At that time the land and minerals were managed by the BLM. In 1977, the BLM completed an environmental review, an EIS level study, on the proposed project in compliance with NEPA. Subsequently, the BLM, through a land exchange process, granted the Blawn Mountain Project land and minerals, and other surrounding areas to SITLA.

20.3 ENVIRONMENTAL SETTING

An environmental study area was delineated, covering all areas proposed for disturbance plus adequate room to adjust areas as necessary. Surveys and studies have been completed for the following disciplines:

- Air Quality
- Archeological Resources
- Wildlife Habitat including Threatened, Endangered, and Sensitive Species
- Vegetation including Threatened, Endangered, and Sensitive Species
- Soils
- Surface and Groundwater
- Wetlands and Waters of the US.

Each of these disciplines is discussed below.

Air Quality

In September 2012, a meteorological monitoring station was installed near the Blawn Mountain Project and a particulate monitoring station and meteorological station was installed closer to Milford, Utah. Both monitoring stations started recording data in October 2012 and completed the one-year monitoring requirement on September 30, 2013. The meteorological and particulate matter data is being reviewed. This data will be used to support air quality permits.

Archeological Resources

The entire environmental study area, the Revenue Basin/Willow Springs Road, and the well field and water pipeline alignment have been surveyed for archeological resources. Archeological sites were identified. The significance of these sites is currently being evaluated. Significant sites will be avoided, mitigated, or managed as appropriate.

Wildlife Habitat Including Threatened, Endangered, and Sensitive Species

The entire environmental study area was surveyed in May 2013 to evaluate general wildlife habitat and determine the presence of any threatened and endangered species or their habitat protected under the Endangered Species Act, or other state designated sensitive species or their habitat. This information is necessary to support permit applications. No wildlife species listed under the Endangered Species Act or Utah State Sensitive Species were found within the project area.

Vegetation Including Threatened, Endangered, and Sensitive Species

The Blawn Mountain Project is located in the pinyon-juniper community as defined by the BLM (1977). The valleys of the area have been extensively chained to remove Juniper and Pinyon to improve grass growth to support livestock grazing. Site specific surveys completed for the environmental study area did not find any vegetation species listed under the Endangered Species Act or any potential habitat.

Soils

An analysis of soils was conducted in order to determine soil suitability and salvage depths for use during revegetation. This survey will aid in preparing permit applications and reclamation planning. The soil survey for the environmental study area was completed in August 2013.

Surface and Groundwater

Surface and groundwater resources are detailed in Section 5 of this report.

Wetlands and Waters of the US

A WoUS inventory which includes jurisdictional wetlands administered under Section 404 of the Clean Water Act was completed for the entire lease area, the water pipeline route, and access roads. Several springs were identified during the survey. None of these springs would be impacted or disturbed as a result of the project.

20.4 ACCESS AND UTILITIES

Access Road

Existing Beaver County maintained roads (Revenue Basin and Willow Springs roads) provide access to the Blawn Mountain Project area. In its current condition, the road is not adequate or wide enough to accommodate the type and amount of vehicles needed to support the project. The land adjacent to the road is managed by the BLM and impacts to this land required for expansion require a ROW grant from the BLM.

Beaver County submitted a SF-299 application to apply for a ROW across BLM lands in July 2012 on the basis that improvement of the road will enhance economic development for future uses in their county as well as adjacent counties. A draft EA was prepared documenting the baseline environmental conditions and the impacts to the environment from the proposed upgrading. The ROW grant was issued to the county in June 2013.

Preliminary construction activities started on the road later in the summer of 2013 with major construction slated to start in spring 2014. PRC will work with Beaver County to develop a road use agreement to allow use of the road to support the Blawn Mountain Project.

Natural Gas Line

Major natural gas suppliers in the State have been contacted regarding natural gas supply to the Blawn Mountain Project. Permits required to construct and maintain the line will be the responsibility of the utility provider. Negotiations are currently underway to allocate both permitting and construction costs associated with the supply of natural gas to the project.

Power Transmission Line

Rocky Mountain Power has been contracted to assess the engineering, cost, and available capacity associated with a power transmission line to the project. Permits required to construct the line will be part of any construction agreement with Rocky Mountain Power, and are expected to be obtained and managed by the utility.

Rail Line

PRC has contacted UP to assess a rail line to the project. UP or other partners as determined will assess engineering and cost associated with the rail line. Permits required to construct the rail line will be part of any construction agreement with UP, or other partner, and managed by that entity.

Water Line

As discussed in Section 18, water necessary to operate the project will be provided mainly from a well field approximately seven miles north-east of the project. Water from this well field will be piped to the site. The pipeline and associated wells and infrastructure will be on SITLA lands and will require a ROW grant from SITLA prior to construction.

20.5 Major Operating Permit and Authorizations

The following discussion and Table 20.1 identifies the major permits and approvals that need to be obtained prior to the construction and start-up of the mine and processing plant.

TABLE 20.1 MAJOR REQUIRED PERMITS

Major Permits or Approvals	Issuing Agency
Exploration Permit	Utah Division of Oil, Gas and Mining
Large Mine Operation Approval	Utah Division of Oil, Gas and Mining
Water Appropriations	Utah Office of State Engineer
Groundwater Permits	Utah Division of Water Quality
Air Quality Permit	Utah Division of Air Quality
General Multi-Sector Industrial Storm Water Permit	Utah Division of Water Quality
Army Corps of Engineers Jurisdictional Waters Concurrence	US Army Corps of Engineers
County Conditional Use Permit and Other Permits	Beaver County
Water Treatment Plant	Utah Division of Drinking Water
Waste Water Treatment Plant	Utah Division of Water Quality

The permits listed in Table 20.1 are not meant to be all-inclusive and cover only the major permits required for the mine and processing plant. In addition various ROWs across state lands will need to be obtained from SITLA in order to construct the water pipeline, and to upgrade existing roads. PRC has been actively working with SITLA and obtaining these ROWs is expected to be very straightforward.

Exploration Permits

Exploration activities of minerals require an approval from UDOGM. Exploration activities within the lease area are being completed under exploration permits E/001/0171 and E/001/0182. The holder of these permits is UAC.

Approval for Large Mine Operation

The Notice of Intent to Commence Large Mining Operations must contain a complete description of the existing environmental resources and impacts. Environmental baseline studies necessary to support the application are complete. The Notice of Intent will include a description of mining methods, a comprehensive reclamation plan, and will identify the financial security acceptable to UDOGM to cover the costs of reclamation to be completed by an independent third-party as required under R647 administrative rules. Execution of the acceptable financial security instrument will be required in advance of commencing mine activities.

Approval of a Notice of Intent to commence Large Mine Operations in Utah can occur within 6-9 months of an application submittal. It is expected that the Notice of Intent will be submitted to UDOGM in late 2013 with an anticipated approval in June 2014.

Water Appropriations

Waters are available adjacent to the project area for which the state is willing to issue rights, or appropriate, water for activities that will put the water to a beneficial use. Based on the criteria the state uses to issue water rights, a defensible appropriations application for water within the Wah Wah Valley was filed with the State Engineer's Office. The State Engineer is currently reviewing the application. A site visit with State Engineer representatives and other affected parties was completed in summer 2013 to discuss issues associated with the pending application. A hearing has been scheduled for 4Q 2013 with an anticipated permit decision within two months following the hearing.

Groundwater Discharge Permit

A groundwater discharge permit application will require the completion of sufficient groundwater investigations in order to evaluate potential impacts to nearby waters, and if necessary provide sufficient mitigation. Ten groundwater monitoring wells were drilled to help characterize the hydrogeologic conditions of the lease area; eight encountered water. These eight wells were completed and equipped for routine monitoring. The hydrogeologic interpretation of data from these wells will be included in the groundwater permit application for the project.

Subsequent to approval of the groundwater discharge application by the UDWQ, it may also be necessary to file a construction permit with UDWQ to validate the engineering and designs for the source of any potential impacts. This would include engineering designs for the tailings facility or other operations that may potentially impact groundwater.

Groundwater discharge permit applications typically are processed in approximately 6-9 months. It is expected that the groundwater discharge permit application will be submitted to UDWQ early in the first quarter of 2014 with an approval in June 2014. Approvals for construction applications typically take a much shorter time and can be approved in as few as three months.

Air Quality Permit

In September 2012, a meteorological monitoring station was installed near the project area and a particulate monitoring station and meteorological station was installed closer to Milford, Utah. Both monitoring stations started recording data in October 2012. The one year data collection requirement was completed September 30, 2013.

Preliminary modeling will be completed to assess the impact of the project to ambient air quality. Once modeling is completed to demonstrate that the project can meet the applicable air quality standards, the application can be prepared and submitted for agency review. The review process can take between 9-12 months. The planned submission of the application to UDAQ is Q2 2014 with an anticipated approval early in Q2 2015.

One year after the start of operations PRC will apply for an Operating Permit, also referred to as a Title V Permit. This permit grants the source permission to continue to operate while self-reporting on performance.

General Multi-Sector Industrial Storm Water Permit

A storm water pollution prevention plan (SWPPP) must be prepared as outlined in the general industrial permit prior to receiving permit coverage. The drainage control plan developed as part of the mining and reclamation plan will be used to develop the SWPPP. The SWPPP must be fully developed and permit coverage granted prior to breaking ground at the site. A SWPPP will be developed so that an application can be submitted for permit coverage Q2 2014 with an approval in Q3 2015.

Army Corps of Engineer's Jurisdictional Waters

Site surveys have been completed for the entire lease area, the water pipeline route and access roads. Current mining operations will avoid all currently identified potential jurisdictional waters. Therefore, no permits or approvals from the ACOE are expected to be required. The delineation survey report will be submitted to the ACOE for their review and concurrence in fall 2013. A site visit of the area is scheduled to occur in November 2013 with ACOE concurrence of the delineation expected in May 2014.

County Conditional Use Permit and Other Permits

PRC has been proactive in maintaining good communication with the local community. To date, county officials as well as local ranchers have expressed strong support for the project, and have expressed high interest in seeing the project succeed. With this level of support for the project, the CUP should be issued without significant challenges. Anticipated time for approval would be 2-4 months once all the supporting studies have been completed. The application is anticipated to be submitted to the county in Q1 2014 with an anticipated approval the end of Q2 2014.

20.5.9 Water Treatment Plant Permit

Water will be treated onsite to provide potable drinking water to staff. Engineering plans and specifications for all public drinking water projects must be approved by the Division of Drinking Water prior to construction. Plans and specifications will be prepared by a Utah-licensed professional engineer. PRC will prepare and submit the application in mid-2015 with an anticipated permit approval in 2Q 2016.

20.5.10 Wastewater Treatment Plant Permit

The Blawn Mountain Project will have a septic tank and an absorption leach field to handle sanitary waste. Disposal of sanitary waste will require a wastewater treatment facility permit by the UDWQ. PRC will prepare and submit the permit application to UDWQ in mid-2015 with an anticipated permit approval at the end of 1Q 2016.

20.6 Social or Community Impact

Representatives of Beaver County have expressed strong support for the project. Beaver County has pursued the ROW grant from the BLM in order to complete road improvements to Revenue Basin and Willow Springs roads to support economic development. The proposed project is consistent with the Beaver County General Plan. The Blawn Mountain Project will assist SITLA in meeting objectives outlined in the "School and Institutional Trust Lands Management Act" by optimizing trust land revenues.

SITLA has teamed with PRC in the development and submittal of a water appropriations application to support the project. SITLA's participation substantiates their interest in seeing the project succeed to further their mandate of promoting sound economic development.

20.7 Summary

A strong permitting and environmental strategy has been developed and implemented to support the permitting timeline. PRC has been very diligent in evaluating the project area's environmental conditions in order to satisfy permitting and regulatory requirements. All of the required environmental baseline field studies and surveys are completed and final reports are being prepared this fall. These studies will assist in preparing major operating permit applications.

The likelihood of a federal action requiring a NEPA analysis for the mine and processing facility is minimal. Based on completed wetland and WoUS delineations avoidance of any jurisdictional waters is achievable.

The mining operation plans have been developed to meet all the regulatory requirements to operate a mine in the State of Utah. All major operating permits required will be obtained prior to mine start-up.

21 CAPITAL AND OPERATING COSTS

21.1 OPERATING SCENARIOS

Capital and operating costs were prepared for a number of different operating scenarios based on the following criteria:

- Recovery methods
  - Flotation
  - Whole ore processing (calcining, roasting and leaching)
- Annual production rates
  - Average of 387,000t of SOP
  - Average of 741,000t of SOP
- Cut-off grade
  - From 3.0% to 3.75% K_2O
- Power supply
  - Line power
  - Self-generated power
- Mill feed material grinding method
  - Wet grinding
  - Dry grinding.

Capital and operating costs for each of the above scenarios were compared to determine which resulted in the most favorable economics. The resulting basis of operations is as follows:

- Whole ore processing
- 40 year plan (500,000 to 860,000t of SOP annually)
- Mining ore grade cut-offs of 3.5% K_2O for Area 1 and 3.25% K_2O for Area 2. Ultimately, all ore of approximately 2.5% K_2O or above is processed resulting in an average of 3.5% K_2O over the project life
- Line power
- 1000 micron mill feed
- Wet grinding.

Recovery Methods

Early operating cost and capital estimates for the flotation case indicated that this method of recovery would not be cost effective. For flotation to work successfully, the material must be crushed and ground to 80 microns, which results in higher capital and operating costs. Additionally, the volume and cost of reagents was significant. Whole ore processing requires slightly more capital, but the operating costs are considerably lower than the flotation scenario resulting in favorable economics for the whole ore processing scenario.

Annual Production Rates

To evaluate the sensitivity of project economics to changes in production levels, multiple scenarios of SOP production were developed. These scenarios considered not only annual SOP production but also annual ROM ore production at various cut-off grades. In the lower production scenarios, a higher grade of ore was used, which reduced the volume of ore processed as a percentage of SOP produced. However, the fixed capital for items such as infrastructure, facilities, and some of the processing plant common to both scenarios made the economics less attractive.

In addition, there were several scenarios developed regarding annual ore volumes and plant size. Because the calciners are the most costly item in the SOP production process, the production capacity of each unit and the number of units became the determining factor in annual ore production volume which, in turn, drove the need to evaluate various cut-off grade scenarios to meet the highest possible SOP production with the optimum level of ore production.

Cut-off Grade

As noted in Section 16, numerous mine plans were developed using various ore cut-off grades ranging from 3.0% to 3.75% K_2O with the intent of optimizing the resource at various production levels and timeframes. The objective of these scenarios was to determine the most economic average grade over various mine lives and to determine whether mining from higher to lower grades over the mine life provided an economic benefit.

As would be expected, the higher the ore cut-off grade, the shorter the mine life. Due to the configuration of the pit shell; in the higher ore cut-off scenarios, significant volumes of lower grade ore (below the selected cut-off grades) had to be mined and stockpiled. The higher ore cut-off grades produced more SOP in the early years than the lower ore cut-off grades, but this scenario required re-handling of lower grade ore so it could be processed after the pit was mined out. Several iterations were run through the economic model to find the optimum level of ore cut-off grade, annual SOP production and re-handling of stockpiled ore.

The result is a mine plan that utilizes an ore cut-off grade of 3.5% K_2O in Area 1 and 3.25% K_2O in Area 2. The lower grade ore (approximately 2.5% K_2O to 3.5% K_2O) is stockpiled for later processing. Mining in Area 1 and 2 is completed by late 2041 at which time the stockpiled ore is then delivered to the processing plant over the remaining 16 years. Ultimately, all ore roughly 2.5% K_2O or higher is processed resulting in an average of 3.5% K_2O over the 40 year project life.

Power Supply

Two options were considered for power supply to the project, line power provided by the local utility or self-generated power using gas fired turbines. In both cases, additional consideration was given to the steam requirements for the processing plant.

In the line power scenario, power is assumed to be provided at tariff rates by the local utility. A power line extension would need to be constructed to bring power to the project site as well as the main substation. Also in this scenario are three 200,000lb/hr gas fired boilers to provide the necessary steam for the processing plant.

In the self-generated power scenario, power is assumed to be provided by three gas turbine generators. Also in this scenario, heat recovery steam generators (HRSG's) are used to capture heat from the turbines to generate steam for the processing plant which eliminates the need for separate steam generators. The power line is not required in this scenario.

The resulting economics for the two scenarios were very close, with the line power case being slightly better. The analysis is however quite sensitive to natural gas prices, the volume of steam recovered from the acid plant and changes in the installed load of the equipment. In addition to slightly better economics, line power reduces the project's sensitivity to natural gas prices, as self-generation requires purchasing natural gas at market prices, whereas electricity in Utah is rate regulated. While further analysis is recommended during the course of the next level of study, the ultimate decision may be made on non-economic factors.

Mill feed material grinding method

Because dewatering and drying are such a large part of the cost of processing when wet grinding is utilized, a comparison to dry grinding was developed. The result was a slight increase in capital and slightly lower natural gas consumption for the dry grinding case. This produced a slight economic benefit for dry grinding but not enough to warrant the additional dust control, material handling and operational problems associated with dry grinding.

21.2 CAPITAL COSTS

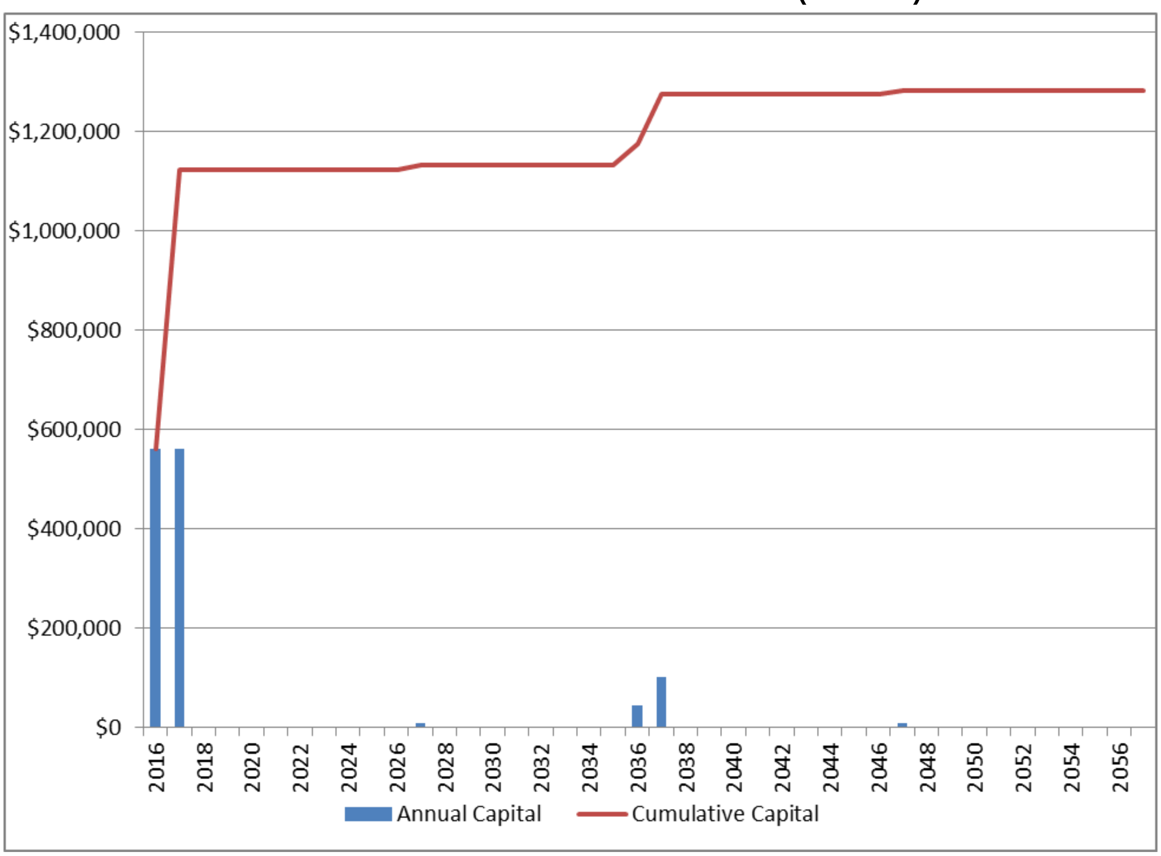
Capital costs for the processing plant are summarized in Table 21.1 below. Chart 21.1 illustrates the annual and cumulative capital over the life of the project. As previously discussed in Section 16, mining operations will be conducted by a contract mine operator. It is assumed that the contract miner will provide the necessary infrastructure and equipment which is reflected in the operating costs shown below.

TABLE 21.1 TOTAL PROJECT CAPITAL ESTIMATE (USD M'S)

	2016	2017	Total Construction and Development Capital	Sustaining Capital	Total Life of Project Capital
Project Infrastructure	$45	$45	$90	$3	$93
Processing Plant	$477	$477	$954	$153	$1,107
Product Storage and Handling	$15	$15	$30	$4	$34
Contingency	$25	$25	$50	$0	$50
Total	**$562**	**$562**	**$1,124**	**$160**	**$1,284**

Contingencies of 15% to 18% were added to the direct costs for various areas of the process plant depending on the design and basis for the cost estimates resulting in an average of 15.4%. It was not applied to turnkey quotes from vendors for the acid plant and calcining process as these included separate contingencies in the estimates.

CHART 21.1 ANNUAL AND CUMULATIVE CAPITAL (USD M'S)



The table and chart above do not include capital costs for the access road, the rail spur and loop, the gas pipeline, the acid plant, the water supply and treatment facility and the mining operations. These items are assumed to be either provided by a third party or financed through government programs. In either case, a provision has been made in the operating costs to account for these items. PRC has received indicative estimates from various parties with respect to the majority of these support assets. These capital costs will not be incurred by PRC. Table 21.2 below shows the capital cost of each item used to formulate the basis for the operating costs.

TABLE 21.2 THIRD PARTY PROJECT CAPITAL (USD M'S)

	Third Party Equipment and Infrastructure
Access Road	$53
Rail Spur and Loop	$76
Natural Gas Transmission Line	$83
Acid Plant	$280
Water Supply and Treatment System	$60
Initial Mine Capital	$89
Total	**$641**

Further discussion of these costs and the amounts included in operating costs are discussed in the Operating Costs Section below.

21.3 BASIS OF ESTIMATE FOR PROCESSING PLANT

Methodology

- The capital cost estimate is based on the industry standard front end loading one (FEL2) conceptual engineering and design of a plant capacity of 750,000tpy of SOP.
- The estimate development methodology is based on major equipment supply costs factored to installed equipment cost.
- Indirect costs are factored on the direct costs and have magnitudes selected to account for the characteristics of the project.
- The major equipment items have been identified from the engineering portion of the study and the developed equipment list.
- Where possible, budget pricing for major cost items has been obtained from vendors based on preliminary duty specifications developed during engineering.
- Alternatively, where recent and relevant project data enables an item to be estimated it may be based on that information.
- Where neither is possible, such as in the case of equipment that will require design and fabrication, preliminary estimates of unit dimensions, materials of construction and material

quantities have been used, concurrently taking into account the nature and complexity of the equipment.

- When none of the above was available, allowances were assigned based on experience and judgment of the engineers and estimators involved in the estimate.

Accuracy

The majority of the direct costs are based on budget pricing with the remaining costs based approximately equally on quantities, determined from engineering developed during the study and allowances. Some budget prices received from vendors have been factored up or down to match plant capacity requirements and mass balances.

A budget pricing means that a budget equipment price is factored to an estimated total cost. It does not mean a budget price was provided for the installed equipment cost.

This study has completed the necessary conceptual engineering to FEL1 and contains a high proportion of base prices from engineering quantity determination and budget pricing of the conceptual duty specifications. As a result, the accuracy of the estimate provided is approximately -30% / +30%.

Qualifications

Budget prices are obtained from vendors based only on general duty specifications of the scope of supply, therefore inaccuracies may occur. Additionally, prices may change significantly in the time between the estimate development and the implementation of the project, developed from process equipment specification and layouts.

Currency

Unless otherwise specified, the default currency is United States dollars (USD).

21.4 Estimate Structure of Direct Cost

A multiple factored estimate of direct cost involves the method of combining factors to the common base cost of the major equipment. The major equipment is referred to as Main Plant Items (MPI). The total cost of the MPI is used as the base cost. The MPI includes all equipment within the battery limits that have significant costs. For example, storage tanks, pumps, heat exchangers are classed as MPI. Multiple factor cost estimating includes the cost contributions for each given activity which can be added together to give an overall factor. This factor can be used to multiply the total cost of delivered equipment to produce an estimate of the total direct costs.

The factored costs have been divided into the following categories:

- Freight – included in indirect costs
- Flow sheet adjustment
- Installation
- Piping
- Structural steel foundations, reinforced concrete
- Architectural buildings
- Electrical
- Instrumentation
- Battery limits, building and service
- Excavation and site preparation
- Painting and installation.

The summation of these categories provides a total factor to multiply against the total MPI which results in the total direct costs.

Notes on factor categories include:

- Early in the development of the process-flow diagrams it is advisable to adjust the estimated MPI cost by 5-20% to allow for later additions or adjustments to the process.
- For order of magnitude estimates the cost of equipment delivered (FOB Plant) varies between 2%-6% of the equipment cost.
- Building Services within the battery limits include electric lighting, sprinklers, plumbing, heating, ventilation and general service compressed air.

21.5 INDIRECT COSTS

Project Indirect Costs

- Basic Engineering- nominally assigned at 2-5% of direct cost.
- Detail Engineering- assigned at 12-15% of direct costs reflecting significant engineering complexity.
- Procurement and construction management services- normally assigned at 4-8% of direct costs.
- Engineering Procurement and Construction Management services (EPCM) including basic and detailed engineering, procurement, and construction management services are assigned at the total of 20% of direct cost.
- Site Facilities and running cost for construction- temporary facilities for construction are included in the indirect costs.
- Operating and Commissioning Spares- spares for start-up and expected breakages during commissioning estimated are included in the indirect costs.

- Outside consultants and vendors support during construction and commissioning- special industry consultants and additional vendor support are included in the indirect costs.
- Operation Manuals- development of operating manuals are included in the indirect costs.
- Major construction cranes services and special construction equipment are included in the indirect costs.

Client Indirect Costs- Excluded

- Owner's Project team- which includes the salaries and costs for Owner's Project team.
- Owner's operator training- There will be a substantial and important activity required in this area if successful commissioning, start-up and operations are to be achieved.
- Commissioning assistance and start-up - Outside the scope of Engineering, Procurement and Construction Management services and requires commissioning engineers and vendor support.
- Permits- Costs associated with developing permit approval applications including baseline testing expenses and Environmental Assessment study.
- Insurance - Insurance against project failure for a range of potential causes.
- Cost of Land
- Cost Escalation
- Hazardous material handling/disposal
- Mining
- Stockpiles, ponds and roads
- Changes in scope of work.

21.6 CONTINGENCY AND TAXES

Contingencies of 15% to 18% were added to the direct costs for various areas of the process plant depending on the design and basis for the cost estimates resulting in an average of 15.4%. It was not applied to turnkey quotes from vendors for the acid plant and calcining process as these included separate contingencies in the estimates. Contingency is an integral part of the estimate and is used as an allowance for the undetermined cost of items that will be incurred within the defined project scope. The contingency covers the cost of these unforeseen items due to the lack of detailed information. It must be assumed that the contingency will be spent. Utah state sales tax at 5.95% is applied to 30% of the total direct cost.

21.7 ESTIMATE SUMMARY

Processing Plant Cost Breakdown by Area

Table 21.3 below summarizes the capital cost estimate for the plant by area.

TABLE 21.3 PROCESSING PLANT CAPITAL DETAIL (US$M)

Description	Total Cost (M)
Direct Cost Summary	
Area 100. Primary Crushing	$ 30.3
Area 200. Alunite Stockpile and Reclaim	$ 24.7
Area 300. Wet Grinding and Classification	$ 55.7
Area 400. Solid/Liquid Separation	$ 29.9
Area 500. Concentrate Drying and Calcination	$ 475.8
Area 600. Acid Plant (Third Party Build, Own, Operate)	$ 0
Area 700. Calcine Leaching and Solid/Liquid Separation	$ 44.3
Area 800. Crystallization and Sop Product Solid/Liquid Separation	$ 86.8
Area 900. Product Drying And Compaction	$ 27.0
Auxiliary Services – Electrical And Steam Distribution	$ 17.4
Total Direct Cost	**$ 791.9**
Indirect Cost Summary	
EPCM Cost*	$ 65.1
Construction Related Cost	$ 71.7
Owner's Related Cost	$ 0
Freight and Sales Tax	$ 25.6
Total Indirect Cost	**$162.4**
Total Installed Project Cost (Excluding Contingency)	**$ 954.3**

*Calculated based on direct costs excluding Area 500, as lump-sum fixed price quote was provided for this area.

21.8 THIRD PARTY PROVISION OF PROJECT INFRASTRUCTURE, MINING AND UTILITIES

PRC has pursued build-own-operate (BOO) arrangements for much of the Project's utility requirements as well as identified funding opportunities through established government programs for certain infrastructure requirements.

Under BOO arrangements, third party service providers will finance the construction, own, operate and maintain certain utility support assets. These include the natural gas pipeline; water production, the sulfuric acid plant and associated facilities; and potentially the electrical transmission line and associated facilities.

Current state and federal funding program are available to assist in the development of the County access roads and the rail line. In addition to mitigating PRC's capital requirements, these BOO arrangements dedicate responsibility for operating these utility and infrastructure assets to experienced operators that will manage the assets in a reliable and cost-efficient manner, allowing PRC to focus on the production of its key products.

Table 21.4 summarizes the utility and infrastructure assets that will be built under BOO arrangements or through indirect government funding. PRC has received indicative offers from various parties with respect to the majority of these support assets. These capital costs will not be incurred by PRC.

TABLE 21.4 THIRD PARTY CAPITAL COSTS

	Third Party Equipment and Infrastructure
Access Road	$53
Rail Spur and Loop	$76
Natural Gas Transmission Line	$83
Acid Plant	$280
Water Supply and Treatment System	$60
Initial Mine Capital	$89
Total	**$641**

The following section provides information pertaining to aforementioned infrastructure, utilities and mining operations items:

Mining Operation Cost Breakdown

Capital required for the mining operation is derived from the equipment fleet and facilities necessary to meet the annual material volume requirements described in Section 16. Cost estimates for major equipment are based on the 2012 Western Mine and Mill Cost Estimating guide. Cost estimates for mine facilities are based on Norwest's experience with similar sized operations. Table 21.5 below summarizes the capital cost estimate for the major mining equipment. A contingency of 10% was assumed due to the fact that mining equipment cost is subject to closer estimation than processing plant costs. The mining equipment contingency is included in the contingency line in Table 21.1 and is not included below.

As noted in Section 16, it is assumed that mining operations will be conducted by a contract mine operator. The terms include a return on capital and a markup on operating expenses. An annual cost of this arrangement has been included in operating expenses.

TABLE 21.5 CAPITAL COST ESTIMATE FOR MAJOR MINING EQUIPMENT (USD M)

Primary Equipment			Unit Cost	Total Initial Capital
Backhoe	22yd^3	1	$4.4	$4.4
Front-End-Loader	16yd^3	1	$3.4	$3.4
End Dump Trucks	148t	14	$3.1	$44.0
Support Equipment				
Water Truck	16,000gal	1	$1.6	$1.6
Grader	297Hp	2	$0.9	$1.8
Track Dozer	580Hp	3	$1.2	$3.7
Drill	50,000lb	1	$1.2	$1.2
Other Support Equipment				$4.6
Capitalized Mine Development Costs				
Mine Development				$15.6
Mine Facilities				$9.4
Total Initial Mine Capital				**$89.7**

Access Roads

The Revenue Basin/Willow Creek road is a County Class D road that traverses about 10.5 miles of U.S. federally managed lands and another 9 miles across SITLA owned property. The U.S. Bureau of Land Management has recently approved upgrading the road for improved access, including truck traffic.

Beaver County holds bonding authority that can be used to fund the road and it is expected that this approach can be utilized. Should any considerations need to be addressed, PRC believes that arrangements with the County can be made such that the County can exercise its bonding authority in support of the road improvement. The County has bonded for other road improvements in the area in support of new projects and economic development. Included in operating expenses is an amount required to amortize the County bond payment over 30 years at 3% interest.

Rail Line

The Railroad Rehabilitation & Improvement Financing (RRIF) program was established by the Transportation Equity Act for the 21st Century (TEA-21) and amended by the Safe Accountable, Flexible and Efficient Transportation Equity Act: a Legacy for Users (SAFETEA-LU). Under this program the US Department of Transportation Federal Railroad Administration (FRA) Administrator is authorized to provide funding, direct loans and loan guarantees up to $35.0

billion to finance development of railroad infrastructure. Up to $7.0 billion is reserved for projects benefiting freight railroads (other than Class I carriers).The funding may be used to:

- Develop or establish new intermodal or railroad facilities
- Acquire, improve, or rehabilitate intermodal or rail equipment or facilities, including track, components of track, bridges, yards, buildings and shops; and
- Refinance outstanding debt incurred for the purposes listed above.

Potash Ridge intends to work with a qualified partner in order to access this program to directly fund the development of the rail spur for the Blawn Mountain Project. Direct loans can fund up to 100% of a railroad project with repayment periods of up to 35 years and interest rates equal to the cost of borrowing to the government. Eligible borrowers include railroads, state and local governments, government-sponsored authorities and corporations, joint ventures that include at least one railroad, and limited option freight shippers who intend to construct a new rail connection. Included in operating expenses is an amount required to amortize the loan over 35 years at 3% interest.

Natural Gas Line

As discussed in Sections 17 and 18, major natural gas suppliers in the State have been contacted regarding natural gas supply to the Blawn Mountain Project. Permits required to construct and maintain the line will be the responsibility of the utility provider. The natural gas line capital cost is $83.0M and PRC has received indicative terms to allocate both permitting and construction costs associated with the supply of natural gas to the project. The annual cost of this natural gas facility charge is included in operating expenses.

Water Line

As discussed in Section 18, water necessary to operate the project will be provided mainly from a well field approximately 7 miles north-east of the project. Water from this well field will be piped to the site. The pipeline and associated wells and infrastructure will be on SITLA lands and will require a ROW grant from SITLA prior to construction. PRC has received an indicative offer to own, operate and maintain the well field, water pipeline, surge pond, storage tanks and storage and reclaim water. The terms include a return on capital and a markup on operating expenses. An annual cost of this arrangement has been included in operating expenses.

Sulfuric Acid Plant

As discussed in Section 17, an estimated 3,390tpd of sulfuric acid (H_2SO_4) is manufactured at the Project site from sulfur dioxide (SO_2) produced by the decomposition of alunite during the thermal processing of ROM ore. PRC has received indicative terms for the ownership and operation of the Sulfuric Acid Plant as well as the marketing of a portion of the Project's sulfuric acid. The terms include a return on capital and a markup on operating expenses. An annual cost of this arrangement has been included in operating expenses.

21.9 Operating Costs

Average annual operating costs for the processing plant and mining operation are shown in Table 21.6.

Table 21.6 Average Annual Plant and Mine Direct Operating Costs (USD$M)

Direct Plant and Mine Cash Production Cost	Annual Average Cost($)/Ton SOP	Life of Plant Annual Average (000)
SOP Tons Sold		645
Sulfuric Acid Tons Sold		1,440
Mining (Contract Mine Operator Cost)	$66	$42,381
Processing		
Crushing & Grinding	$35	$22,322
Concentrate	$7	$4,317
Roasting	$199	$128,418
Acid Plant	$60	$38,474
Leaching & Crystallization	$14	$8,834
Drying and Compaction	$4	$2,553
Steam Plant	$23	$14,951
Tailings and Reclaim	$3	$2,108
Water and Tailings Thickeners	$0	$0
Product Storage and Loading	$4	$2,420
Total Processing	$348	$224,396
Credit for Value of Acid	($302)	($194,348)
Total Direct Operating Cost (Mining and Processing)	$112*	$72,430

*Rounded

The material balance for the project which forms the basis for the production volumes, plant and mine sizing and consumables was developed by PRC and ICPE. The average annual volume of saleable products from the project includes 645,000t of SOP, ranging from 861,000 to 496,000t, and 1.4Mt of sulfuric acid ranging from 1.9Mt to 1.3Mt.

Labor and benefits costs were developed by Norwest using regional data and our experience in the mining industry. PRC provided manpower requirements for the plant while Norwest developed manpower requirements for the mine as shown in Section 16.

Headcount for the processing plant is projected to be 362 employees. Headcount for the contract mining operation will range from 60 to 180 employees over the life of the project. Average mining headcount is 108 employees. Total project employment level is projected to be 470.

Plant equipment operating costs, except for the acid plant and water treatment facility, are based on the equipment list provided by ICPE. Costs per hour were estimated using the 2012 Western Mine and Mill Cost Estimating Guide and 7,920 estimated operating hours per year. An additional cost for miscellaneous operating expenses is included and is based on 15% of maintenance costs. Costs for the operation of the acid plant and water treatment facility were provided by PRC and are based on third party contract operation of those facilities.

Mine equipment operating costs are based on the equipment list shown in Section 16, the operating hours estimate included in the mine plan and the cost per hour taken from the 2013 Western Mine and Mill Cost Estimating Guide with an additional cost for miscellaneous operating costs equivalent to 15% of maintenance costs. Explosives costs are based on quantities derived from the mine plan and an estimated cost per pound based on Norwest's experience. Other supplies and mine costs are based on Norwest's experience with similar sized operations.

Quantities of consumables such as power, water and natural gas were either developed from the equipment schedule developed by ICPE or provided by PRC. Unit costs for consumables were developed by PRC and include pricing for natural gas at $3.90 per mmbtu delivered to the site, electricity based on tariff rates provided by the local utility and costs for the water and acid plants based on third party indicative offers to own and operate those facilities.

Power, natural gas and wear materials are the most significant operating costs associated with process facilities and plant infrastructure. The main power consumers include the crushing and grinding circuit, the drying, calcination and roasting plant and the acid plant. Power consumption for the grinding circuit was determined using the results of metallurgical testing and subsequent grinding mill design by Outotec. Power consumption for the acid plant was provided as part of the DuPont MECS, Inc. preliminary design for the acid plant. Power consumption for the calcining and roasting plant was provided as part of the preliminary system design by FL Smidth. The major power consumers in the calcination and acid plants are the fans required for gas transfer.

The primary natural gas consumers in the process facilities are the drying, calcining and roasting system, the calcine leaching circuit and the SOP crystallization circuit. Natural gas is the main fuel used in calcining and roasting of the ore and the products of natural gas combustion are the main reactants used in ore reduction. FL Smidth provided the preliminary design for the calcining circuit, which included the quantities of natural gas required.

Natural gas is used to produce steam for heating the solutions in the calcine leaching circuit and for evaporation of the leach solutions in the evaporation and crystallization circuits. The steam requirements for crystallization were determined as part of the preliminary crystallization system design by Swenson Technology, Inc. Typical sources of process steam are excess heat recovered from the sulfuric acid plant and from natural gas fired package boilers; the steam plant. In the Blawn Mountain case, the SO_2 concentrations from roasting the alunite are too low for the acid plant to provide excess steam and so the entire load must be provided by Cleaver Brooks package boilers. The natural gas consumptions and steam production data were determined from the boiler performance tables for the boilers selected by Cleaver Brooks.

Wear materials for the crushing and grinding circuits were calculated from the results of JK Tech comminution testing by HRI and power consumption of the crushing and grinding equipment.

Included in operating expenses are annual costs associated with the arrangements described in more detail above in Section 21.8. A summary of the annual costs are shown in Table 21.7.

TABLE 21.7 THIRD PARTY UTILITY, INFRASTRUCTURE AND MINE OPERATING COSTS

Utilities, Infrastructure and Mining	Third Party Annual Costs (USD M's)
Access Road and Rail Spur	$5
Water, Acid Plant and Natural Gas Line BOO Arrangements	$67
Average annual contract mining cost	$42
Total	**$114**

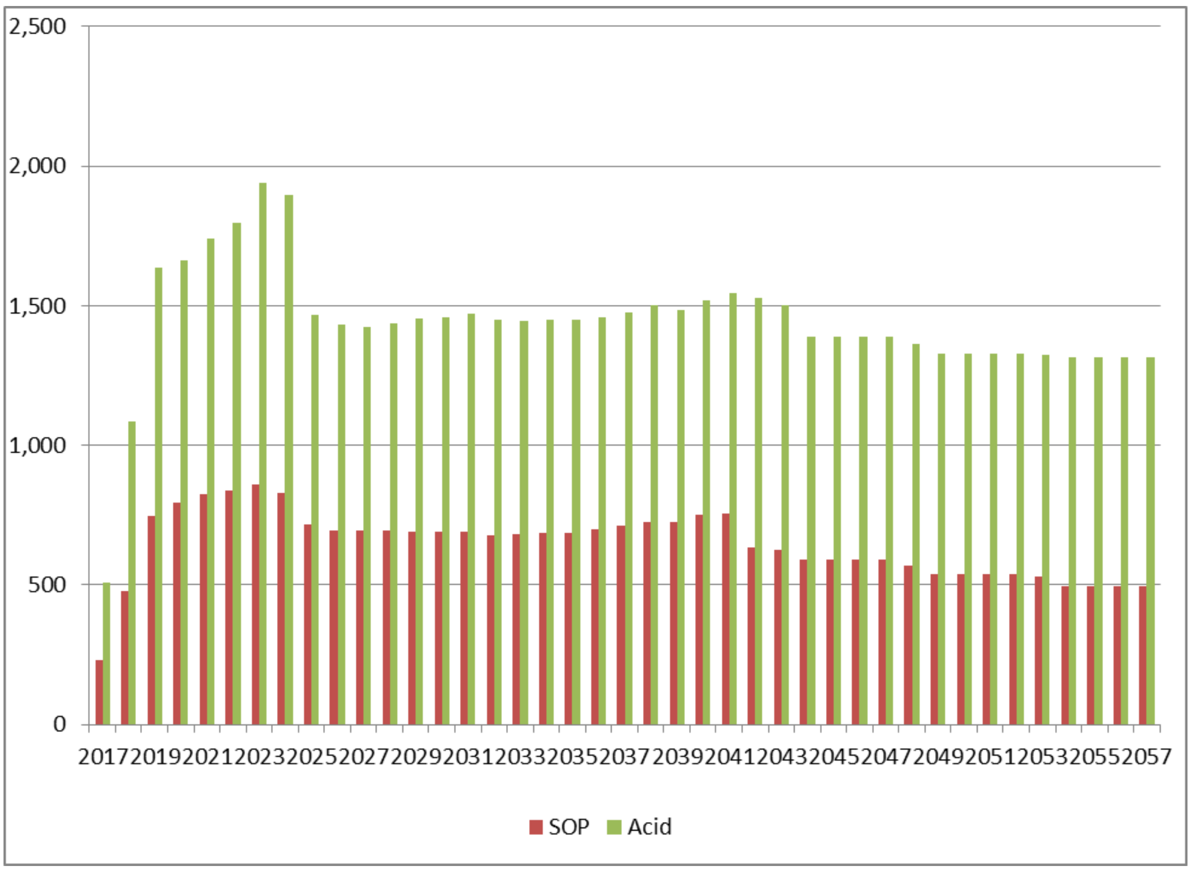
22 ECONOMIC ANALYSIS

22.1 PRINCIPAL ASSUMPTIONS

Production Volume and Schedule

Production volume is planned at an average of 645,000t of SOP per year for the 40 year life of the project, ranging from 861,000t to 496,000t. As a result of the SOP production process, an average of 1.4Mt of sulfuric acid is also produced annually. This requires an annual average of 10.4Mt of ore which is constant at 10.6Mt after a short ramp up period. Over the 40 year period, there are 26.4Mt of SOP and 59.0Mt of sulfuric acid produced. Production volumes are shown below.

CHART 22.1 PRODUCTION VOLUMES (SOP AND ACID TONS)



Product Pricing and Transportation

Product sales prices are discussed in Section 19. Those prices were used in developing the cash flows for the project. Table 22.1 below summarizes average selling prices and annual revenues. As noted in Section 19, the selling prices below are at the plant gate. All prices are stated in constant 2013 dollars, there is no provision for inflation.

TABLE 22.1 PRICING SUMMARY USD

Pricing and Transportation Costs	Unit	Life of Plant Annual Average $M
SOP Tons Sold	645,000	
Sulfuric Acid Tons Sold	1,440,000	
Average SOP Selling Price – FOB Rail at Plant	$649/t	$419
Average Sulfuric Acid Selling Price – FOB Rail at Plant	$135/t	$194
Average annual revenue		**$613**

Cash Production Costs

Direct cash production costs were summarized in Section 21. Additional cash production costs include site G&A expenses, property taxes, third party infrastructure and utility costs, corporate overhead and royalties. Site G&A expenses were developed by Norwest based on our experience with similar operations. Royalties are based on the lease agreement which provides for a royalty of 5% and 4% of selling price for SOP and sulfuric acid respectively. Property taxes are based on current regulations from Beaver County, Utah. Corporate overhead cost was provided by PRC. The basis for third party utility and operating costs is discussed in Section 21. Total cash production costs are shown in Table 22.2 below. All costs are stated in constant 2013 dollars, there is no provision for inflation.

TABLE 22.2 TOTAL CASH PRODUCTION SUMMARY

Total Cash Production Costs	Annual Average Cost($)/Ton SOP	Life of Plant Annual Average
SOP Tons Sold		645,000
Sulfuric Acid Tons Sold		1,440,000
Direct Plant and Mine Cash Production Cost	$414	$266,777
Credit for Value of Acid	($302)	($194,348)
Subtotal of Direct Plant and Mine Cash Production Cost	$112	$72,430
Royalties	$45	$28,704
Site G&A	$12	$7,932
Property Taxes	$11	$7,308
3rd Party Facility Charges (Road, Rail, Water & Gas)	$34	$21,632
Corporate Overhead	$4	$2,500
Total Cash Production Cost	$218	$140,506*

*Rounded

Income Taxes

Income taxes include both US Federal and State of Utah corporate taxes at a blended rate of 38.25% applied to estimated taxable income from the project including the effect of the Alternative Minimum Tax provisions. Tax depreciation is calculated based on US Federal tax regulations. Percentage depletion is also taken as a deduction in computing taxable income. Because the regulations regarding percentage depletion where significant processing of the mineral takes place are complex, PRC provided Norwest with an opinion from a tax expert on the application of the percentage depletion to this project. The opinion was provided by Wisan, Smith, Racker, & Prescott LLP of Salt Lake City, Utah. The income tax calculations in the project economics reflect this opinion. The opinion of the tax expert is qualified based on the completeness and the accuracy of the description, provided by PRC, of the processes involved and the relationship of those processes to the marketable products.

22.2 CASH FLOW

Cash flow from the project is summarized in Table 22.3.

TABLE 22.3 PROJECT CASH FLOW SUMMARY

Project Cash Flow Summary	Life of Plant Annual Average $M
SOP Tons Sold	645
Sulfuric Acid Tons Sold	1440
Net SOP revenue FOB - Plant	**$419**
Net acid revenue FOB - Plant	**$194**
Total revenue FOB - Plant	**$613**
Direct Plant and Mine Cash Production Costs	$267
Royalties	$29
Site G&A	$8
Property Taxes	$7
Third Party Facility Charges	$22
Corporate Overhead	$3
Total Cash Production Costs	$336
Operating Margin	$277
Income Taxes	$53
Cash Flow from Operations	**$224**

Pre-production cash outflows total $1.1 billion over the two year construction period. Cash flow is positive beginning in 2018. Payback occurs mid-way through 2022 which is approximately 7 years after the initial investment. Cash flow after payback averages $221M per year for a total net cash flow of $8.0 billion over the life of the project. Annual and cumulative cash flows are shown in Chart 22.2. The summary of cash flow for the project is presented in Table 22.4.

CHART 22.2 ANNUAL AND CUMULATIVE CASH FLOW

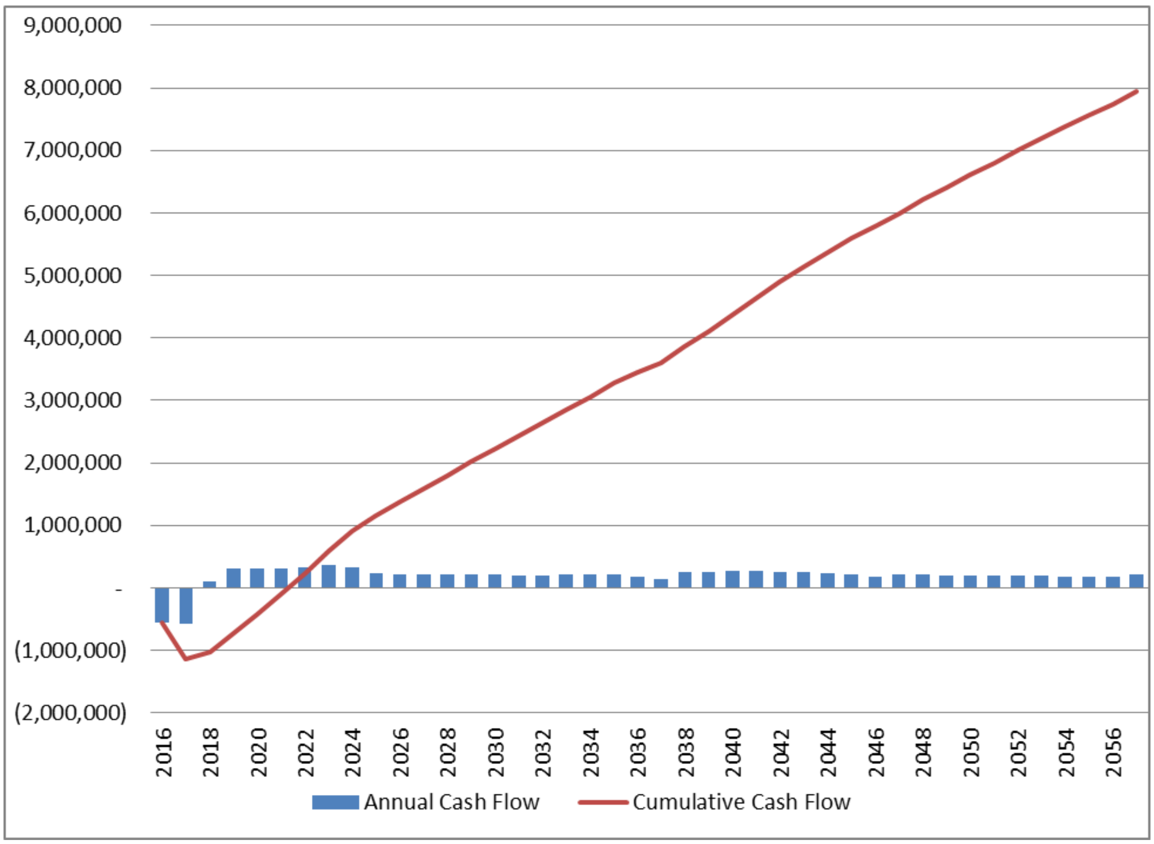


TABLE 22.4 PROJECT CASH FLOW

	2016	2017	2018	2019	2020	2021	2022-2026	2027-2031	2032-2036	2037-2041	2042-2046	2047-2051	2052-2057	Life of Project
	Total	**Total**	**Total**	**Total**	**Total**	**Total**	**Average**	**Average**	**Average**	**Average**	**Average**	**Average**	**Average**	
SOP Tons Sold		229	479	745	792	824	789	692	686	734	605	555	508	26,427
Sulfuric Acid Tons Sold		509	1,086	1,638	1,663	1,740	1,707	1,448	1,450	1,506	1,438	1,347	1,318	59,024
Net SOP revenue FOB Plant		$120	$259	$448	$518	$539	$516	$453	$449	$480	$396	$363	$333	$17,162
Net acid revenue FOB Plant		$69	$147	$221	$224	$235	$230	$195	$196	$203	$194	$182	$178	$7,968
Total Revenue		$189	$406	$669	$743	$774	$747	$648	$645	$683	$590	$545	$510	$25,131
Direct Plant and Mine Cash Production Cost		147	229	286	298	308	293	309	310	295	228	227	226	10,938
Royalties		9	19	31	35	36	35	30	30	32	28	25	24	1,177
Site G&A		3	8	8	8	8	8	8	8	8	8	8	8	325
Property Taxes		0	0	5	8	9	10	7	7	7	9	8	7	300
3rd Party Facility Charges		21	29	29	29	29	29	29	22	21	25	13	9	887
Corporate Overhead		3	3	3	3	3	3	3	3	3	3	3	3	103
Total Cash Production Cost		$182	$288	$362	$380	$394	$378	$386	$380	$366	$300	$284	$276	$13,729
Operating Margin		$8	$118	$308	$362	$380	$369	$262	$265	$317	$290	$261	$234	$11,402
Income Taxes		0	0	7	51	68	73	49	52	61	61	55	48	2,165
Cash Flow from Operations		$8	$118	$300	$312	$312	$296	$213	$214	$256	$228	$206	$187	$9,236
Capital Expenditures	($562)	($577)	($15)	$0	$0	$0	$0	($2)	($9)	($20)	$0	($2)	$0	($1,284)
Net Cash Flow	($562)	($569)	$103	$300	$312	$312	$296	$212	$205	$236	$228	$204	$187	$7,953

22.3 Financial Analysis

The after tax, 2013 constant dollar, internal rate of return for the project is 20.5%. After tax net present values at 8%, 10%, and 12% are shown in Table 22.5.

Table 22.5 Net Present Value Results

Discount Rate	8%	10%	12%
After Tax Net Present Values	$1.5 billion	$1.0 billion	$0.7 billion

The accuracy of resource and reserve estimates is, in part, a function of the quality and quantity of available data and of engineering and geological interpretation and judgement. Given the data available at the time this report was prepared, the estimates presented herein are considered reasonable. However, they should be accepted with the understanding that additional data and analysis available subsequent to the date of the estimates may necessitate revision. These revisions may be material. There is no guarantee that all or any part of the estimated resources or reserves will be recoverable.

22.4 Sensitivity Analysis

Table 22.6 below shows the sensitivity of the project economics to changes in selling price, direct operating costs, and capital costs. Also included are changes in SOP price, acid price and natural gas price. Acid revenues represent 32% of total revenue and natural gas represents 35% of total operating costs and 50% of processing costs.

Table 22.6 Sensitivities

Discount Rate	8%	10%	12%
Base Case	$1.5 billion	$1.0 billion	$0.7 billion
10% Increase in Revenue	$1.9 billion	$1.4 billion	$1.0 billion
10% Decrease in Revenue	$1.0 billion	$0.7 billion	$0.4 billion
10% Increase in SOP Selling Price	$1.8 billion	$1.3 billion	$0.9 billion
10% Decrease in SOP Selling Price	$1.2 billion	$0.8 billion	$0.5 billion
10% Increase in Acid Price	$1.6 billion	$1.1 billion	$0.8 billion
10% Decrease in Acid Price	$1.4 billion	$0.9 billion	$0.6 billion
10% Increase in Operating Costs	$1.3 billion	$0.8 billion	$0.5 billion
10% Decrease in Operating Costs	$1.7 billion	$1.2 billion	$0.8 billion
10% Increase in Natural Gas Price	$1.4 billion	$1.0 billion	$0.7 billion
10% Decrease in Natural Gas Price	$1.6 billion	$1.1 billon	$0.7 billion
10% Increase in Capital Costs	$1.4 billion	$0.9 billion	$0.6 billion
10% Decrease in Capital Costs	$1.6 billion	$1.1 billion	$0.8 billion

The project economics are more sensitive to selling price than changes in capital or operating costs.

23 ADJACENT PROPERTIES

There is no data or information available for adjacent properties that are pertinent to this report.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 PROJECT SCHEDULE

The Blawn Mountain Project schedule has been prepared in order to meet commissioning and production target dates in 2018. A summary of this schedule is included below and is represented on Table 24.1.

- Exploration drilling of Areas 1 and 2 in support of the prefeasibility study was completed in August 2013. Additional drilling may be completed over the course of the project for expansion of the existing areas and development of new areas on the project site.
- Project financing includes timing to cover five major categories for financing feasibility through production. Long lead items identified will be procured starting in 2014, during the feasibility study.
- Environmental permitting will span into second quarter 2016. Major operating permits and those required to start civil construction will be obtained prior to November 2015 when civil work begins.
- Engineering studies and procurement started in January 2013 and will conclude in March 2017. Basic and detailed engineering will last for 24 months. Civil construction is scheduled to begin following completion of basic engineering at the end of 2015.
- Third party design/build utilities include power, gas and rail. Engineering and environmental studies will start first quarter of 2014. Construction of these utilities will be in the final stages for the start of the plant commissioning in third quarter of 2017. These activities will be dependent on early financing being structured to ensure that property alignments, environmental studies and plans of development can be executed to meet the plant commissioning portion of the schedule.
- Construction of the access road to the project area will be completed first quarter 2016. Acid and calcine plant construction will begin after primary foundation and civil works are completed at the end of 2015. Prior to the start of mining, haul roads and stockpile pads will be constructed.
- Mining will start in December 2016. Ore will be stockpiled until the processing plant is online.
- Plant commissioning quality control and construction support will end November of 2017. Commissioning will occur from third quarter 2017 through second quarter of 2018. Once complete, stockpiled ore (from mining activities over the previous 18 months) will be processed until full production. Full production will start fourth quarter of 2018.

TABLE 24.1 SCHEDULE SUMMARY

	2013				2014				2015				2016				2017				2018			
Major Tasks	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4
Confirmation Drilling																								
Financing																								
Permitting																								
Environmental Permits																								
Construction Permits																								
Engineering Studies and Procurement																								
Third-Party Design Build Packages																								
Railroad																								
Power Line																								
Gas Line																								
Construction																								
Access Road																								
Civil																								
Processing Plant (includes acid and calcine plant)																								
Production																								
Mining																								
Plant Commissioning																								
Production																								

24.2 PRE-FEASIBILITY STUDY

The information in this report is a summary of a comprehensive and detailed pre-feasibility study prepared by Norwest and ICPE. This study includes numerous process design, mine plan and economic trade-off studies that led to the basis of the overall project design criteria.

25 INTERPRETATION AND CONCLUSIONS

25.1 MINING AND RESOURCE

The mineral tracts are controlled by PRC by the Exploration/Option Agreement through SITLA. The property has undergone exploration which can be considered sufficient for the delineation of mineral resources in Area 1 and Area 2. Other areas (3 and 4) under PRC control are considered exploration targets. The drilling and surface mapping within Area 1and Area 2 has led to a geologic interpretation of the deposit as rhyolite porphyries and ignimbrites that have experienced hydrothermal alteration and consequent enrichment in potassium and aluminum compounds, and termed alunite.

Mineral resources have been estimated and reported at a 1% K_2O cut-off grade for Area 1 and Area 2. Measured plus Indicated in situ resources for Area 1 are 164.8Mt and 398.3Mt for Area 2.

Utilizing the resource estimate as defined in this report; Norwest developed and examined several mine plan options based on various parameters. Economic trade-off studies were performed, comparing different mine plan and processing options. Ultimately a mine plan emerged that displayed the best economics and served as the base case for the PFS. Norwest used the base case mine plan to develop capital and operating costs for the project. Ultimately, a cash flow was generated that formed the basis of a NPV and IRR calculation that confirmed the economic viability of the mine plan and hence allowed Norwest to define reserves.

The accuracy of resource and reserve estimates is, in part, a function of the quality and quantity of available data and of engineering and geological interpretation and judgment. Given the data available at the time this report was prepared, the estimates presented herein are considered reasonable. However, they should be accepted with the understanding that additional data and analysis available subsequent to the date of the estimates may necessitate revision. These revisions may be material. There is no guarantee that all or any part of the estimated resources or reserves will be recoverable.

25.2 MINERAL PROCESSING AND METALLURGICAL TESTING

ICPE reviewed a number of technical reports and results of a comprehensive metallurgical testing program commissioned by PRC and performed at HRI during the 2011-2013 period and input from desktop simulation of unit operations by equipment manufacturers. The extensive metallurgical tests included all aspects of comminution, beneficiation, flotation, calcination, leaching, crystallization, material handling, and solid/liquid separation aspects of the envisioned Project facilities.

The Blawn Mountain Project's exploration drilling program supplied the bulk samples and composites of drill core and rotary drill cuttings for the experiments. Additionally, historical test results from bench-scale and pilot plant experiments commissioned by ESI during the 1970s and performed at HRI were also critically reviewed. The conclusions are as follows:

The production to the process plant will be approximately 10.4Mtpy ROM ore (dry solids). The plant is designed with a throughput of 1,313tph. It is anticipated that the quality of the ore will consist of:

- K_2SO_4 grade in feed ≈ 5.9+%
- Specific gravity of ore ≈ 2.46
- Moisture content of ore ≈ 2%

The process plant will include:

- Whole ore feed, with ore blending, as required
- Grinding circuit will generate a plant feed of 1000 µm (1.2mm maximum), P_{80}
- Plant feed to Gas Suspension Calciner
- Processed ore to water leach for SOP recovery
- Filtrate recycling to leach for SOP concentration build-up
- Quadruple Effect Evaporator/Crystallizer will process brine to extract SOP
- Compacted and sized SOP product for markets
- Roaster off-gases (SO_2) as feedstock to Sulfuric Acid Plant
- Washed and repulped tails will be stored as Al_2O_3 resource. It is anticipated that the terminal density of the tails will be approximately 85%.
- Source water will be from groundwater wells at a rate between 1,200 to 2,000gpm
- Water will be conserved through extensive re-use of effluents.

It is anticipated that the quality specifications for the Standard SOP product are:

- Purity: 50 wt.% K_2O (92.5 wt.% K_2SO_4)
- Particle size: 70 to 10 Tyler mesh
- Chloride content: < 1.0%.

The byproduct of the process includes sulfuric acid and alumina. The sulfuric acid will be produced at a production rate of approximately 4,000tpd. The product purity will be about 98% H_2SO_4. The alumina (Al_2O_3) will exist in the tails residue. This material will be stored in the tailings pond and may be able to be processed as part of a gamma phase alumina amenable to caustic leach.

25.3 PROJECT RISKS

Other than those noted elsewhere in this report regarding resource and reserve estimates, there are no further significant risks or uncertainties that affect the reliability or confidence in the exploration information, resource estimates or reserve estimates.

As noted in Section 22.4, the project economics are subject to variations in SOP product price, acid price, and natural gas pricing as well as changes in capital and operating cost estimates. Acid revenues represent 32% of total revenue and natural gas represents 35% of total operating costs and 50% of processing costs. The potential impact of changes in these items is shown in Table 22.6.

25.4 CONCLUSION

Based on the results of the PFS, Norwest and ICPE have reached the following conclusions:

- There are sufficient mineable tons of ore at an average grade of 3.51% K_20 to produce approximately 645,000 tons of SOP over a 40 year project life.
- No fatal flaws have been identified at this stage of project development.
- Pre-production capital costs estimated at $1.1 billion along with several third party build, own, operate arrangements will be required to bring this project into production.
- Cash costs of production per ton of SOP, after sulfuric acid credits and before royalties, is estimated at $173
- Based on the assumptions defined in this report, the project will generate positive cash flows and achieve an after tax IRR of 20.5%.

The overall conclusion is that the results of this study indicate positive economic results and the project should be continued to the next phase of development.

26 RECOMMENDATIONS

26.1 MINERAL PROCESSING AND METALLURGICAL TESTING

Recommendations on additional metallurgical test work and trade-off studies required for optimized flow sheet development and process plant design are as follows:

- Perform mineralogical studies using such technology as QEMSCAN, an automated mineralogy and petrography system to identify and delineate texture, grain sizes, and mineralogical associations in the ore from different parts of the mine, calcine from the roasters, and leach residue, which have a direct bearing on product grade and recovery.
- Conduct tests on dry ROM ore samples to identify the choice of crushing and grinding equipment for producing feed at P_{80} 1000 µm for the Dryer/Calciner/Roaster circuit and develop quantitative data on liberation of alunite as a function of grain size screen fractions from different areas of the mine.
- Estimate grinding media consumption and the amount and composition of fines generated during crushing and grinding which can be sent directly to leaching.
- Evaluate the results of slurry rheology, sedimentation, and filtration tests to establish type and dosage of flocculant, if required, and to select thickeners and filtration equipment to reduce the moisture content of the feed to the pyroprocessing steps.
- Perform Drying/Calcining/Roasting tests to determine the operating parameters and trade-off studies to assist in equipment selection, determination of energy requirements and composition of SO_2-bearing off-gases for recovery of sulfuric acid as by-product.
- Identify the phases (potassium sulfate, crystalline alumina, and residual alunite) in the respective calcines produced at **a range of temperatures** by XRD and microscopic examination.
- Perform agitated tank water-leach studies to determine operating parameters such as pulp density, residence time, temperature, intensity of agitation, as well as to identify the phases (potassium sulfate, crystalline alumina, and residual alunite) in the water-leach residues of calcines produced at **a range of temperatures** by XRD and microscopic examination.
- Conduct pilot plant tests on evaporation and crystallization of SOP product from the brine to determine the operating parameters and trade-off studies to assist in equipment selection, product quality, bleed requirements for impurity control, size of crystals formed, and compaction of product and handling requirements.

26.2 RECOVERY METHODS

The following is a list of comments and recommendations for the PFS of Area 100 through 1300, excluding Area 1100 and 1400. It should not be considered a complete list, but as a partial list of comments and recommendations to be considered at the beginning of the Feasibility Study:

Area 100

- Review gyratory crusher capacity criteria and perform Bruno Modeling (by Metso) to validate final selection.
- At the completion of testing, verify ore characteristics of -24in. ROM ore, percentage of fines, provide crusher plugging analysis, crushing size distribution.
- Consider a grizzly at the Truck Dump Pocket.
- Depending upon final ROM ore throughput, consider (1) Apron Feeder.

Area 200

- Provide 15-17° maximum slope on the Pebble Circuit Conveyor.
- The reclaim area was conceptually designed for 100% redundancy.
- Size dust collector based on ore characteristics and percentage of fines.
- Consider smaller concrete dome (i.e. DOMTEC) with automated dome reclaim systems (i.e. Cambelt).

Area 300

- Validate assumptions of minus 24in. ROM ore to Primary Crusher and size gradation.
- Upon completion of testing, validate the need for a Pebble Crusher Circuit.
- Upon finalization of all test work, execute proposal from Orway Mineral consultants (OMC) for SAG Mill circuit simulations to validate Vendor Data and Mass Balance.
- Consider grinding Mill Sump Distributor.
- Provide adequate height at SAG Mill discharge to feed pebble circuit screen.
- Due to high wear and maintenance, consider replacing pebble screen with SAG Mill trommel screen or oversized pebble screen.
- Consider Ball Storage Bin at COS with charging of balls onto the SAG Mill Feed Conveyor.
- Ensure adequate space on platforms for Mill Feed Liners.
- Consider elevating SAG Mill if pebble screen is utilized.
- Consider additional building height depending on final cyclone selection.
- Optimize crane access to pumps.
- Optimize sump drainage and front end loader access to sump.
- Optimize mill floor slopes for proper drainage.
- Optimize pump to cyclone pumping loops by locating pumps as close as possible directly below the cyclone.
- Consider Resistoflex piping in place in place of rubber lined pipes.
- Consider utilizing Caterpillar tracked options for transfer cart.
- Consider future expansion options and adequate lay down area inside and outside of mill building.

- Optimize building crane by using one crane for SAG Mill, one crane for Ball Mills and one crane for pumps and cyclones.
- Adjust mill building width as required to optimize support equipment, i.e. lube units, lay down, mill liner handling, mill charging, mill access platforms, etc.

Area 400

- Complete testing to validate the Grinding Circuit Thickener underflow percent solids and the Belt Filter capacity and performance.

Area 500

- Care should be taken to minimizing gas volume and limit O_2<2%.
- Engage vendors to provide pilot plant-scale tests of Drying/Calcining/Roasting crushed and ground ROM ore with P_{80} = 1000 µm (1.0 mm) and not > 1.2 in size range should be completed to validate the thermal processing concept and to validate the estimated concentration of SO_2 in the off-gases as feedstock to the Sulfuric Acid Plant.

Area 600

- Engage vendors to provide pilot plant testing to validate the feed stock composition including percent volumes of SO_2, O_2, CO_2, N_2, H_2O and SO_3.
- Consider consolidating plants into one large plant.

Area 700

- Complete testing to validate Tailings Thickener and Belt Filter capacity and performance.
- Optimize water and energy conservation measures.

Area 800

- Complete testing of brine to validate Crystallization equipment capacity and performance.

Area 900

- At the completion of testing, verify SOP characteristics, percentage of fines, and adjust equipment and flow streams accordingly.

Area 1000

- A final gas consumption and pressure analysis must be completed for the sizing of the gas line. A gas load of 100 Million Standard cf/day was assumed at the beginning of the Prefeasibility Study to establish a base line for comparison between Kern River and Questar Gas.

Area 1100

- Verify the tailings material characteristics for any potential contaminants. Evaluate at a minimum pH, gradation, and specific gravity. May consider oil &grease and other potential constituents of concern.
- Determine tailings discharge handling methods.
- Evaluate water reclaim methods – decant versus barge pumping.
- Determine water quality of the water draining from the tails – pH, TDS, cation-anion balance, etc.
- Determine rate of water draining from the tailings.
- Geotechnical evaluation of foundation characteristics of the tailings area for embankment and tailings stability.
- Determine geology and permeability of materials underlying the tailings to assess potential for seepage from the site.

Area 1200

- Complete testing on the various flow streams as indicated on the following process plant water balance block flow diagram.
- Provide detailed chemical analysis of water harvested from the well field, including a mass balance of cations and anions.
- Establish limits on the total dissolved solids (TDS) content and/or the concentration of residual reagents in water recycled to the processing plant.
- Provide recommendations on a method of recovering SOP values from the bleed stream established for controlling formation of glaserite.
- Determine infiltration rate and soil type of leach field area. Work with State regulators for approval as a sewage treatment option.

Area 1300

- Review anticipated storage needs for both the Mine and Plant to ensure adequate capacity
- Geotechnical evaluation for slab design.

Area 1400

- Geotechnical evaluation for the foundation of the dome storage and the conveyor bases.
- Evaluation for acid handling and storage containment.

26.3 CONTINUED EXPLORATION AND RESOURCE DELINEATION

Exploration has identified significant resources and reserves on Area 1 and Area 2. There are other target areas within the Blawn Mountain Project that have potential of hosting additional alunite mineralization. As the project advances towards development, exploration should continue to evaluate resource potential of other targets within the project area including the following four targets:

Target 1

Extending resource limits southeastward from the central portion of Area 2 with additional drilling. The geologic model and field observations suggest mineralization may extend farther to the southeast than currently defined.

Target 2

Two rhyolitic eruptive centers occur west-southwest of Area 2 that are coincident with a lineament that projects into the main zone of mineralization in Area 1. Though historic mapping does not identify this lineament as a fault, the geometry of the two deposits suggests the lineament may be a source conduit for sulfide fluids and hydrothermal alteration. Outcrops on the two volcanic domes exhibit alunite veining. This area should be tested with drilling.

Target 3

ESI exploration efforts identified alunite mineralization on two areas referred to as Area 3 and 4. Limited drilling was carried out by ESI on both areas and Potash Ridge completed two holes on Area 4. Additional mapping, sampling, and drilling should be carried out on both areas.

Target 4

A prominent fault projects northeastward from the north side of Area 1 out across the alluvial valley. The fault separates a small ridge from Area 1 proper, and three small hills protrude through the alluvial fill along the fault. The fault marks the contact between Devonian carbonate rocks and Miocene volcanic rocks that host the alunite mineralization. Jasperoid development (silicification) has also been observed along the contact. Outcrop sampling and alteration mapping leading to drill testing along the fault should be planned to identify potential new alunite resources.

Table 26.1 presents the budget for carrying out exploration on the four targets. Figure 26.1 identifies the proposed exploration targets for the Blawn Mountain Project.

TABLE 26.1 EXPLORATION BUDGET FOR 2014

Description	Unit Cost ($)	January - May		June-July		August-September		Total ($)
		Units	Cost ($)	Units	Cost ($)	Units	Cost ($)	
Target 1								
Drill Planning	7,500	1	7,500					7,500
Permitting	5,000	1	5,000					5,000
Site Preparation	1,500	7	10,500					10,500
RC Drilling	28	2,800	78,400					78,400
Geologic Personnel	1,850	8	14,800					14,800
Analyses	39	280	10,864					10,864
Abandonment/ Reclamation	3,700	7	25,900					25,900
SubTotal			**152,964**					**152,964**
Target 2								
Drill Planning	7,500	1	7,500					7,500
Permitting	5,000	1	5,000					5,000
Site Preparation	1,500			6	9,000			9,000
RC Drilling	28			1,200	33,600			33,600
HQ Core Drilling	50			600	30,000			30,000
Geologic Personnel	1,850			12	22,200			22,200
Analyses	39			180	6,984			6,984
Abandonment/ Reclamation	3,700			6	22,200			22,200
SubTotal			**12,500**		**123,984**			**136,484**
Target 3								
Mapping/Sampling								
Geologic Personnel	1,850	25	46,250					46,250
Analyses	53	150	7,965					7,965
Drilling								
Drill Planning	7,500			1	7,500			7,500
Permitting	5,000			1	5,000			5,000
Site Preparation	1,500			12	18,000			18,000
RC Drilling	28			2,700	75,600			75,600
HQ Core Drilling	50			1,000	50,000			50,000
Geologic Personnel	1,850			17	31,450			31,450
Analyses	39			370	14,356			14,356
Abandonment/ Reclamation	3,700			12	44,400			44,400
SubTotal			**54,215**		**246,306**			**300,521**
Target 4								
Mapping/Sampling								
Geologic Personnel	1,850			15	27,750			27,750
Analyses	53			100	5,310			5,310
Drilling								
Drill Planning	7,500					1	7,500	7,500
Permitting	5,000					1	5,000	5,000
Site Preparation	1,500					7	10,500	10,500
RC Drilling	28					1,500	42,000	42,000
HQ Core Drilling	50					600	30,000	30,000
Geologic Personnel	1,850					11	20,350	20,350
Analyses	39					210	8,148	8,148
Abandonment/ Reclamation	3,700					7	25,900	25,900
SubTotal					**33,060**		**149,398**	**182,458**
Total			**219,679**		**403,350**		**149,398**	**772,427**

26.4 ENVIRONMENTAL

As the project advances towards development, information needs to assist in permitting activities are needed. To evaluate these needs within the project area the following information should be collected:

- Evaluate tailings geochemistry to determine the need for a liner under the tailings area.
- Determine the seepage rate of the material under the tailings area.
- Evaluate geochemical characteristics of the waste rock and low grade ore stockpiles to ensure non-toxic, non-hazardous.

26.5 WATER

The source of water is an important component of this project. Studies to date have indicated the presence of water within the Wah Wah Valley. Test bores have verified the water surface at the depths anticipated. Work by ESI in the 1970's showed that the aquifer was able to produce water at the rates needed. The ability of the aquifer under SITLA ground, in the southern portion of the Wah Wah Valley, to produce water at the rate needed for the project needs to be verified.

26.6 MINING

Geotechnical investigations of the roads, pads, pit walls, waste rock piles, tailings materials, and collection and settlement pond embankments need to be conducted

26.7 INFRASTRUCTURE

The following studies should be considered as part of developing this project in future efforts. These reports will assist in providing better definition of the project scope and designs.

- Geotechnical - for foundation for the processing plant and other facilities including any rail lines
- Product storage requirements, capacity, liner requirements size distribution of SOP
- Reclaim system – options to meet product requirements
- Complete rail siting study to determine the route of the short line rail system to the project area.

26.8 SUMMARY OF RECOMMENDATIONS

Based on the positive results of this pre-feasibility study, it is recommended that this project be continued to the next phase of development. The recommendations for additional work have been outlined above. A summary of the work and estimated costs are noted in Table 26.2.

TABLE 26.2 ESTIMATED COSTS OF ADDITIONAL WORK

Description	Range of Estimated Costs ($millions)
Mineral Processing and Metallurgical Testing	$1.5 to $3.0
Continued Evaluation of Recovery Methods	Included in Feasibility Study
Development Drilling	$0.75 to $1.0
Permitting	$2.0 to $3.0
Feasibility Study	$6.0 to $8.0
Total	**$10.5 to $15.0**

27 REFERENCES

Abbott, J.T., Best, M.G., and Morris, H.T. (1983) Geologic map of the Pine Grove-Blawn Mountain area, Beaver County, Utah: US Geological Survey Map I-1479, two sheets, scale 1:24,000.

Ackerman, J. B. (1978). Alunite Flotation, M.S. Thesis, Department of Metallurgy and Metallurgical Engineering, University of Utah, Salt Lake City, UT.

AGRC (2011) National Agricultural Imagery Program (NAIP), 1 meter resolution color orthophotography: http://gis.utah.gov/aerial-photography/2009-naip-1-meter-orthophotography, accessed June 2011.

ALS Metallurgy Kamloops. (June 5, 2013). Metallurgical Assessment of The Blawn Mountain Project, Utah, United States. Work Performed on behalf of Potash Ridge Corp. Kamloops, BC, Canada.

Anonymous. (February, 1989). The NG Alunite Property of Earth Sciences, Inc. Beaver County, Utah.

Berge, Dale (1974) Preliminary report on archeological investigations in southern Wah Wah Mountains, Beaver County Utah: Provo, Brigham Young University, unpublished report of the Museum of Archeology and Ethnology.

Bowen, F. J. et al. (July 7, 1972). Metallurgical Studies of Utah Alunite Ore, for Earth Sciences, Inc. Hazen Project 1132, Hazen Research, Inc. Golden, CO.

Bowen, F. J. et al. (April 12, 1973). Metallurgical Studies of Utah Alunite Ore, for Earth Sciences, Inc. Hazen Project 1235, Hazen Research, Inc. Golden, CO.

Chapman, E.P., Jr. (1974) Interim report for Earth Sciences, Inc., on the NG Alunite Deposit Wah Wah Mountains, Beaver County, Utah, U.S.A.: Unpublished reserve report by Chapman, Wood, and Griswold, Ltd., 18 p.

Couzens, T.R. (1975) Alumet project mine planning for NG Alunite Area C: Unpublished report for Earth Sciences, Inc. by Pincock, Allen & Holt, Inc., 118 p.

David Chinloy, Hatch Ltd., Mississauga, ON. (March 9, 2012). Potash Ridge Alunite Processing, Project Memo to Potash Ridge: Ross Phillips/Paul Hampton. H340277-0000-90-220-0001, Rev.0.

Earth Sciences, Inc. (1974) Alunite mine/processing plant complex, western Beaver County, Utah: Environmental Impact Assessment prepared by Earth Sciences, Inc. for the U.S. Bureau of land Management, two volumes, 744 p.

Gillespie, Daniel W. (March 1, 2012). Comminution Testing. Letter report from to Ross Phillips of Potash Ridge Corp. Hazen Project 11468-01, Hazen Research, Inc. Golden, CO.

Green, Don W., and Perry, Robert H. (2008). Perry's Chemical Engineers' Handbook, 8th ed. McGraw-Hill.

Grzmil, B. U. and B. Kic. (2005). "Single-Stage Process for Manufacturing of Potassium Sulfate from Sodium Sulfate," in Chemical Papers, Institute of Chemistry, Slovak Academy of Sciences, Vol. 59(6b), pp. 476-480.

Hall, R.B. (1978) World Nonbauxite Aluminum Resources – Alunite: U.S. Geological Survey Professional Paper 1076-A, p. 35.

Hall, R.B., and Bauer, C.W. (1983) Alunite, in LaFond, S.J., Industrial minerals and rocks, 5th edition: Littleton, Society for Mining, Metallurgy, and Exploration, Inc., p. 417-434.

Hanson, Richard C. (March 18, 1977). ***Crushing and Grinding of Alunite Ore,*** for Alumet Co.

Hatch Ltd., Mississauga, ON. (March 19, 2012). Potash Ridge: Alunite Processing Process Design Basis. H340277-0000-109-0001, Rev. B.

Hazen Research, Inc. (July 7, 1972). Metallurgical Studies of Utah Alunite Ore. HRI Project No. 1132.

Hazen Research, Inc. (June 24, 2013). Alunite Process development and Pilot Plant Program—Interim report. HRI Project No. 11687.

Hodgson, Rodney and Thomas, P. N. (April 12, 1972). ROASTING AND LEACHING OF ALUNITE ORE, Progress Report No. 1, for Earth Sciences, Inc. Hazen Project 1132, Hazen Research, Inc. Golden, CO.

Hofstra, A.H. (1984) Geology, Alteration and Genesis of the NG Alunite Area, Southern Wah Wah Range, Southwestern Utah: Golden, Colorado School of Mines M.S. Thesis, 130 p.

Hu, Yue-hua, Li Har-pu, and Jiang, Yu-ren (October 2002). "Effect of Hydroxamic Acid Starch on Reverse Flotation Desilicate from Diasporic Bauxite," Trans. Nonferrous Met. Soc. China, Vol. 12, No. 5, pp. 974-978.

Imhof, R., Battersby, M., Parra, F., and Sanchez-Pino, S. (June 6 - 9, 2005). "The Successful Application of Pneumatic Flotation Technology for the removal of Silica by Reverse Flotation at the Iron Ore Pellet plant of Compania Minera Huasco, Chile," Centenary of Flotation Symposium, Brisbane, Queensland, Australia.

Jennings, L. D., and Bisque, R. E. (November 13, 1970). Preliminary Tests to Evaluate the Extraction of Alumina and Potassium Sulfate from Alunite Ores, for Earth Sciences, Inc.

Kesler, G. H. (February 10, 1977). Chemical Thermodynamics of Alunite Processing. Report No. SR-77-1, Alumet Company, Golden, CO.

Krahulec, Ken (2007) Mineral potential of the Blawn Wash Alunite area, Beaver County, Utah: Unpublished report prepared by the Utah Geological Survey for the Utah School and Institutional Trust Lands Administration under a Memorandum of Understanding, 29 p.

Lange, Norbert A., and Forker, Gordon M. (1967). Handbook of Chemistry, 10th ed. McGraw-Hill.

Liu, Wen-gang, Wei, De-zhou, Wang, Ben-ying, Fang, Ping, Wang, Xiao-hui, and Cui, Bao-yu (February 18, 2009) "A New Collector Used for Flotation of Oxide Minerals," Science Digest, Elsevier Science Press.

Loost, Kent W. (June 21, 1977). Recovery of Aluminum from Alunite Ore Using Acid-Leach to Purify the Residue for Bayer Leach, US Patent 4,031,182, filed March 24, 1976, and issued June 21, 1977.

MECS, Inc. (June 2009). MECS Acid Technology, Corporate brochure.

Mellon, Robert J. (May 21, 2012). Alunite Development Program, prepared for Utah Alunite LLC and Potash Ridge Corporation. Hazen Project 11468, Hazen Research, Inc. Golden, CO.

Mellon, Robert, Hazen Research, Inc. (March 28, 2013), Comminution Testing: Hazen Project 11687, Report and Appendices.

Mellon, Robert, Hazen Research, Inc. (2013). Periodic Updates on Bench-scale and Pilot Plant Test Results and Personal Communication.

Miller, G.M. (1966) Structure and stratigraphy of southern part of Wah Wah Mountains, southwest Utah: Bulletin of the American Association of Petroleum Geologists, v. 50, p. 858-900.

Myertons, Ralph (May 27, 1976). Flotation of Alunite and Alumina, prepared for Alumet Company, Golden, CO.

NorFalco. (April 2007). Sulfuric Acid Handbook.

Norwest Corporation (2012) Technical Report, Blawn Mountain Project, Beaver County, Utah: CIM Technical Report for Potash Ridge Corporation, effective date April 16, 2012, Published on SEDAR.com.

Norwest Corporation. (November 5, 2012). Technical report: Preliminary Economic Assessment Blawn Mountain Project, Beaver County, Utah. Submitted to Potash Ridge Corporation effective date October 30, 2012, Published on SEDAR.com.

Norwest Corporation (November 8, 2013). Prefeasibility Study Report: Blawn Mountain Project, Beaver County, Utah, submitted to Potash Ridge Corporation.

Oxtoby, David. W., Gillis, H. P., and Campion, Alan. (2008). Principles of Modern Chemistry, Thomson Books/Cole.

Parkinson, Gerald (1974) Golden pilot plant points way to 500,000-tpy alumina-from-Alunite mine and plant in Utah: Engineering and Mining Journal, August, p. 75-78.

Perry, Harry (1977_ Mining plan for NG Alunite area C – first five years: Unpublished Earth Sciences, Inc. report, 122 p.

Potash Ridge Corporation, Toronto, and Hatch Ltd., Mississauga, ON. (February 16, 2012). Blawn Mountain Alunite Project: Project Update. Power Point Presentation.

Rinckhoff, J. B. (1959). "Modern American Contact Plants, Section I. Chemico-Designed Contact Plants," in The Manufacture of Sulfuric Acid, American Chemical Society, pp. 230-245.

Rowley, P.D., Lipman, P.W., Mehnert, H.H., Lindsay, D.A., and Anderson, J.J. (1978) Blue Ribbon lineament, an east-trending structural zone within the Pioche mineral belt of southwestern Utah and eastern Nevada: U.S. Geological Survey Journal of Research, v. 6, p. 175-192.

Salisbury, Tyler J. (June 24, 2013). "Alunite Process Development and Pilot Plant Program Interim Report," Hazen Project 11687, Hazen Research, Inc. Golden, CO.

Shawe, D.R., and Stewart, J.H. (1976) Ore deposits as related to tectonics and magmatism, Nevada and Utah: American Institute of Mining Engineers Transactions, v. 260, p. 225-232.

Shouldice, Tom, Sloan, Robert, and Pojhan, Aralia, ALS Metallurgy Kamloops. (June 5, 2013). Metallurgical Assessment of The Blawn Mountain project, Utah, United States. KM3816.

Sinnott, R. K. (2005). Coulson & Richardson,Chemical Engineering, Vol. 6: Chemical Engineering Design, 4th ed. Elsevier.

Sisselman, R. (October 1975). Engineering and Mining Journal (E&MJ), Vol. 176, N0. 10, pp. 79-84.

Stephens, J.C. (1974) Hydrologic reconnaissance of the Wah Wah Valley drainage basin, Beaver and Millard Counties, Utah: U.S. Geological Survey Open-File Report 137, 79 p. [also Utah Department of Natural Resources Technical Publication 47].

Susarla, V. R. K., Chudasama, K. M., Mohandas, V. P., and Ghosh, P. K. (June 2007). "Glaserite Preparation by Sodium Sulfate and Potassium Chloride," in Journal of Scientific and Industrial Research, Vol. 66, pp. 444-449.

Tippin, R. B., and Huiatt, J. L. (March 7, 1977). A New potash Flotation Process from Laboratory Testing Through Pilot Plant Studies and on to Full-Scale Operation, Paper presented at the 106th AIME Annual meeting, Atlanta, GA.

Tippin, R. B. (July 18, 1977). "Potash Flotation Process Handles Variable Feed," in Chemical Engineering, pp. 73-75.

Thompson, David L. (December 9, 2011). Recovery of K_2SO_4 and H_2SO_4 (and Possibly Alumina) from Alunite Deposits in Southwestern Utah, for Potash Ridge Corporation. HAZEN Project No. 11416, Hazen Research, Inc. Golden, CO.

Thompson, A.J.B. (1991) Characteristics of acid-sulfate alteration in the Marysvale-Pioche mineral belt: a guide to gold mineralization: Utah Geological and Mineral Survey Miscellaneous Publication 91-2, 29 p.

U.S. Bureau of Land Management (1977) Alunite Project – Final Environmental Statement: U.S. Bureau of Land Management, U.S. Department of the Interior, variously paginated.

U.S. Bureau of Land Management (2011)[a] Cedar City Field Office Resource Management Plan, Major Rights-of-Way Map: Online at, https://www.blm.gov/epl-front-office/projects/lup/7100/17354/17554/CCFO_ROW_42x50L.pdf, accessed 07/05/2011.

U.S. Bureau of Land Management (2011)[b] Cedar City Field Office Resource Management Plan, Allotment Map: Online https://www.blm.gov/epl-front-office/projects/lup/7100/17353/17553/CCFO_HMA-Allotment_42x50L.pdf, accessed 07/05/2011.

U.S. Bureau of Land Management (2011)[c] Cedar City Field Office Resource Management Plan, Wilderness Study Areas and Recreation Map: Online < https://www.blm.gov/epl-front-office/projects/lup/7100/17355/17555/CCFO_Wilderness-3PeakSRMA_42x50L.pdf>, accessed 7.511.

U.S. Bureau of Land Management (2011)[d] Cedar City Field Office Resource Management Plan, Wildlife map: Online https://www.blm.gov/epl-front-office/projects/lup/7100/17352/17552/CCFO_Wildlife_42x50L.pdf, accessed 7/5/2011.

U.S. Bureau of Land Management (2011)[e] Cedar City Field Office Resource Management Plan, Renewable Energy Resources Map: Online at, https://www.blm.gov/epl-front-office/projects/lup/7100/17350/17550/CCFO_RenewableResources_42x50L.pdf, accessed 07/05/2011.

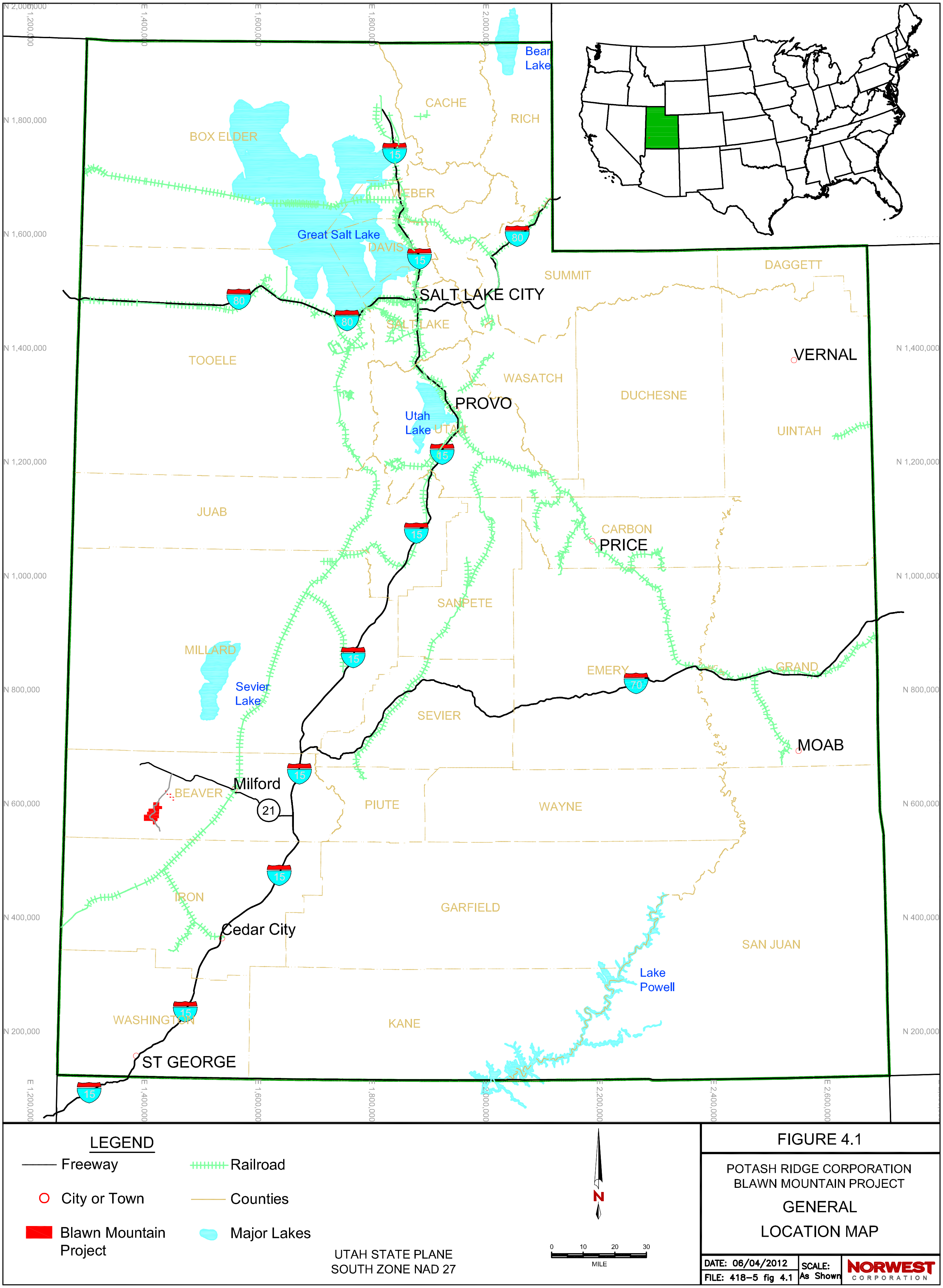
U.S. Department of Energy (2011) West-wide Energy Corridor Programmatic Environmental Impact Statement: Online at: http://corridoreis.anl.gov/, accessed July 6, 2011.

Walker, W.W. (1972) Report of results phase III exploration program on the NG Alunite property Beaver County, Utah: Unpublished report for National -Southwire Aluminum Co., and Earth Sciences, Inc., 123 p., appendices, and plates.

Walker, M.W. (1973) Evaluation of the tonnage and grade of the White Mountain Alunite deposit, Beaver County, UT: Unpublished report for Earth Sciences, Inc., 21 p., appendices and plates.

Whelan, J.A. (1965) Hydrothermal Alteration and Mineralization, Staats Mine and Blawn Mountain Areas, central Wah Wah Range, Beaver County, Utah: Utah Geological and Mineralogical Survey Special Studies 12, 31 p.

28 ILLUSTRATIONS



Bear Lake
CACHE
RICH
BOX ELDER
WEBER
Great Salt Lake
DAVIS
SUMMIT
DAGGETT
SALT LAKE CITY
SALT LAKE
TOOELE
VERNAL
WASATCH
DUCHESNE
PROVO
Utah Lake
UTAH
UINTAH
JUAB
CARBON
PRICE
SANPETE
MILLARD
Sevier Lake
EMERY
GRAND
SEVIER
MOAB
Milford
BEAVER
21
PIUTE
WAYNE
IRON
GARFIELD
Cedar City
SAN JUAN
Lake Powell
WASHINGTON
KANE
ST GEORGE
15
70
80
N 1,800,000
N 1,600,000
N 1,400,000
N 1,200,000
N 1,000,000
N 800,000
N 600,000
N 400,000
N 200,000
E 1,200,000
E 1,400,000
E 1,600,000
E 1,800,000
E 2,000,000
E 2,200,000
E 2,400,000
E 2,600,000
LEGEND
Freeway
Railroad
City or Town
Counties
Blawn Mountain Project
Major Lakes
UTAH STATE PLANE SOUTH ZONE NAD 27
N
0 10 20 30
MILE
FIGURE 4.1
POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT
GENERAL LOCATION MAP
DATE: 06/04/2012
FILE: 418-5 fig 4.1
SCALE: As Shown
NORWEST CORPORATION

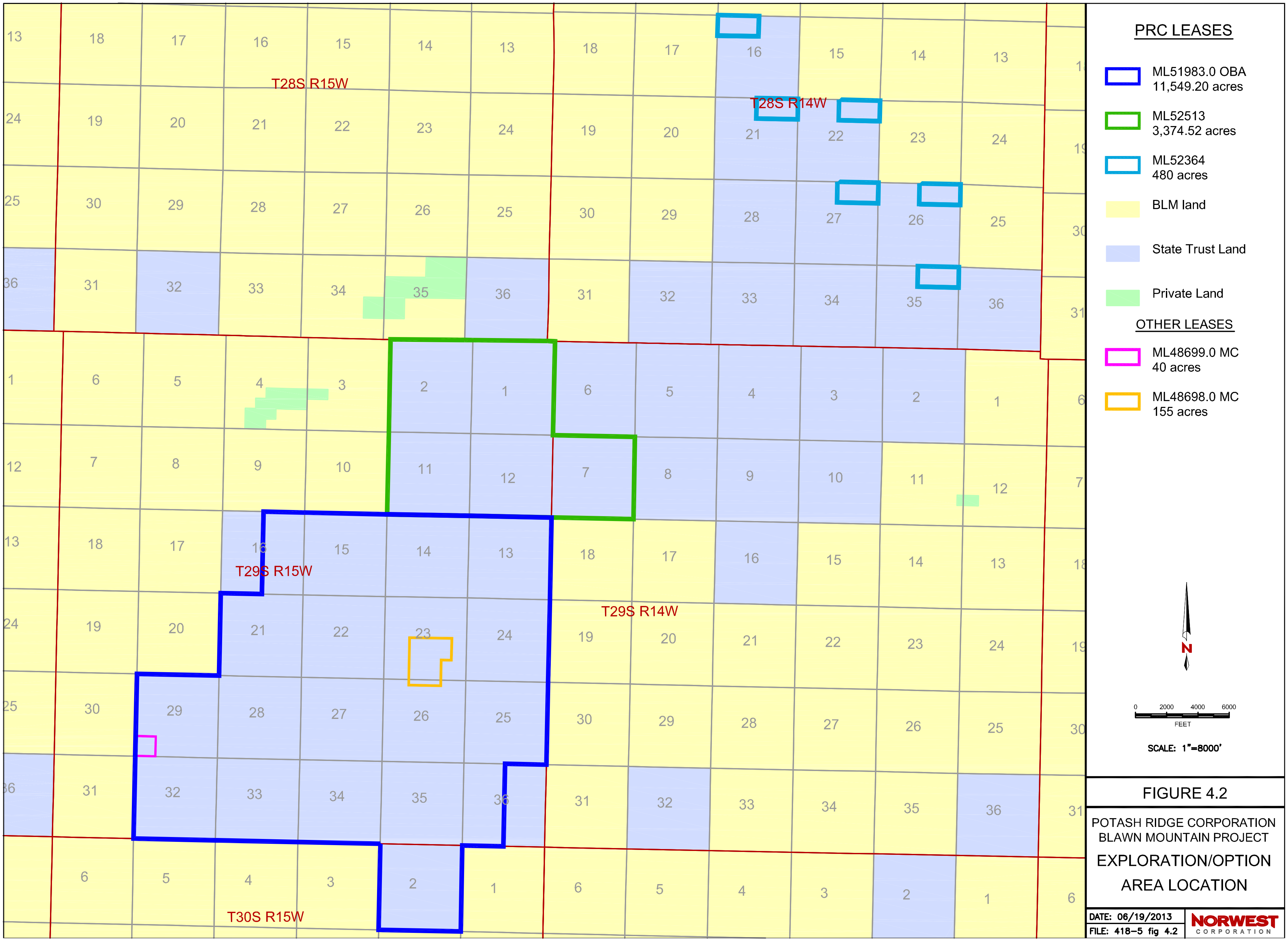


FIGURE 4.2

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

EXPLORATION/OPTION AREA LOCATION

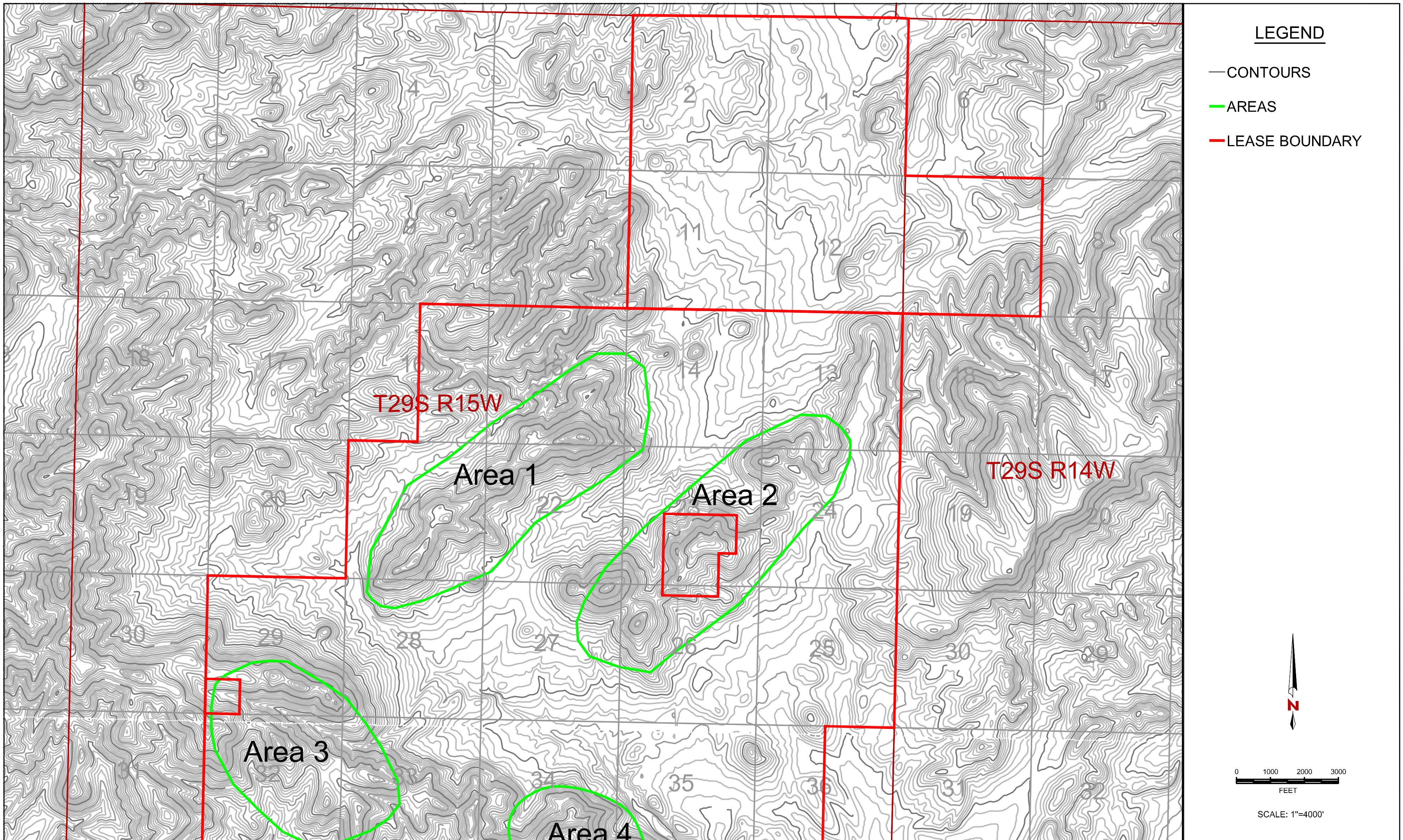


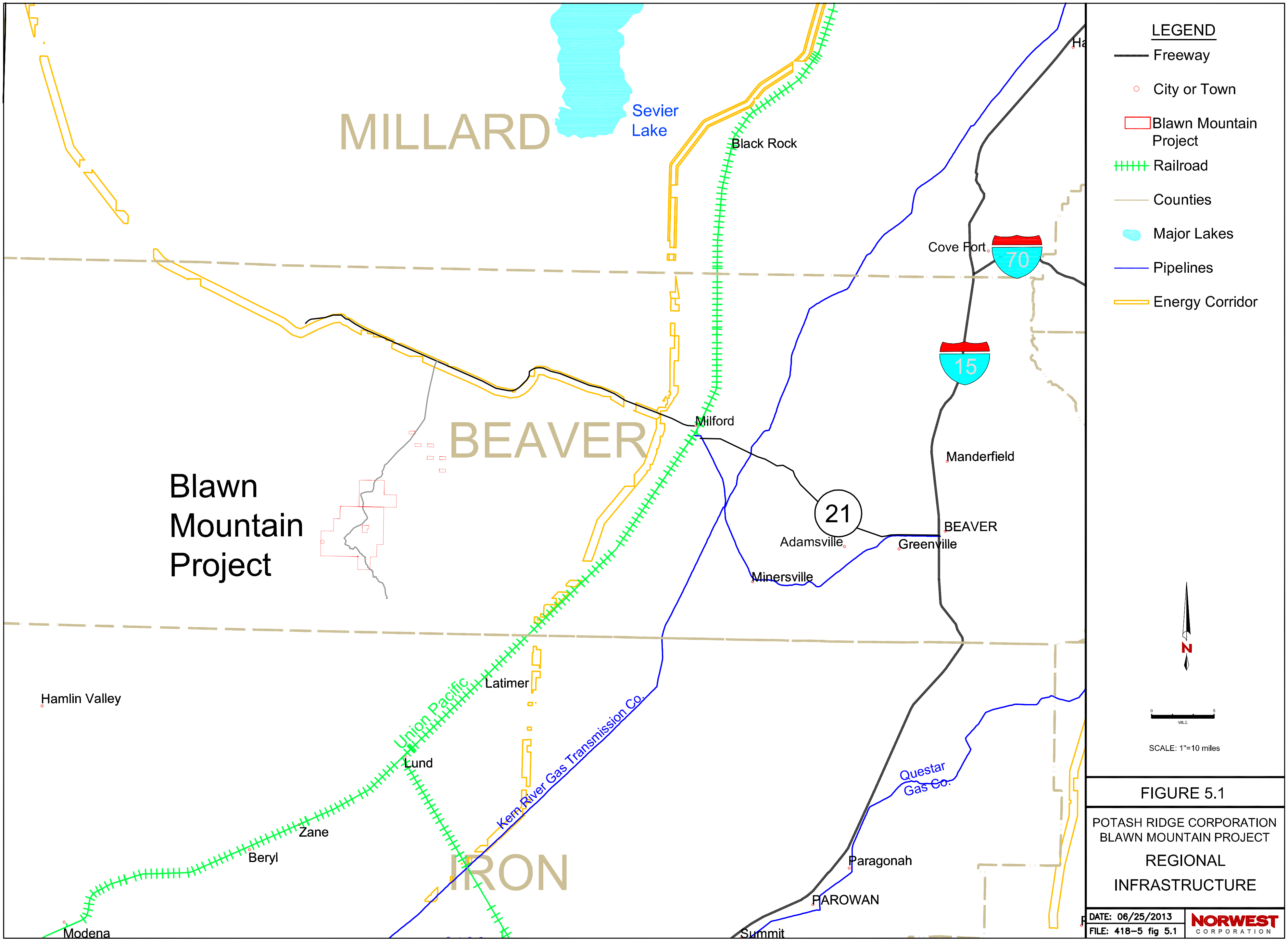
FIGURE 4.3

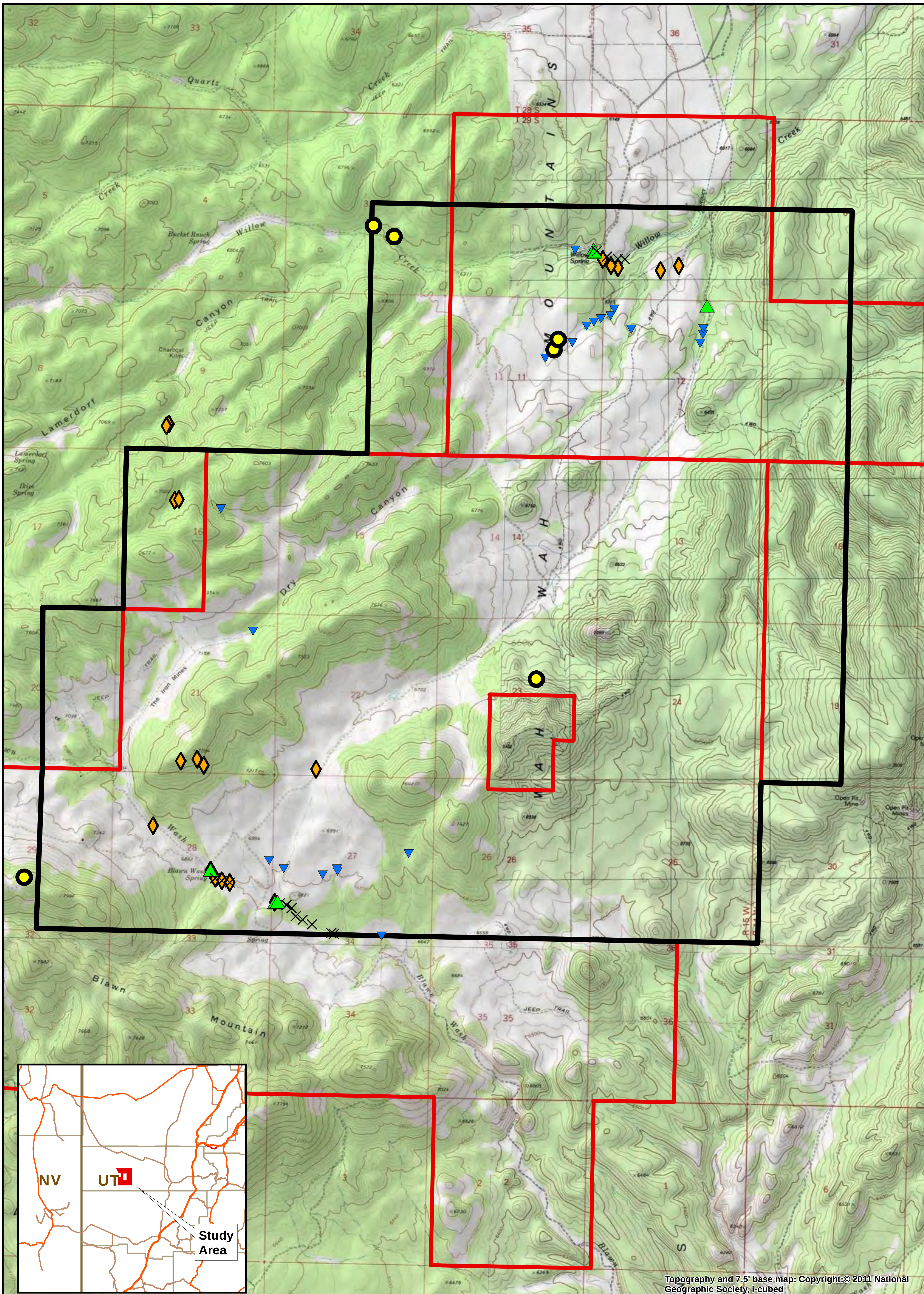
POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

MINERALIZATION ZONES

DATE: 06/25/2013
FILE: 418-5 fig 4.3

NORWEST CORPORATION


MILLARD
Sevier Lake
Black Rock
Cove Fort
70
15
BEAVER
Milford
Manderfield
Blawn Mountain Project
21
BEAVER
Adamsville
Greenville
Minersville
Hamlin Valley
Latimer
Union Pacific
Lund
Kern River Gas Transmission Co.
Questar Gas Co.
Zane
Beryl
IRON
Paragonah
PAROWAN
Modena
Summit
LEGEND
Freeway
City or Town
Blawn Mountain Project
Railroad
Counties
Major Lakes
Pipelines
Energy Corridor
N
SCALE: 1"=10 miles
FIGURE 5.1
POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT
REGIONAL INFRASTRUCTURE
DATE: 06/25/2013
FILE: 418-5 fig 5.1
NORWEST CORPORATION



Legend

- Study Area Boundary
- Lease Boundary
- Historic Water Sources

2013 Survey

- Spring
- Seep
- Observation point
- Notable Vegetation

N

0 3,000 6,000
Feet

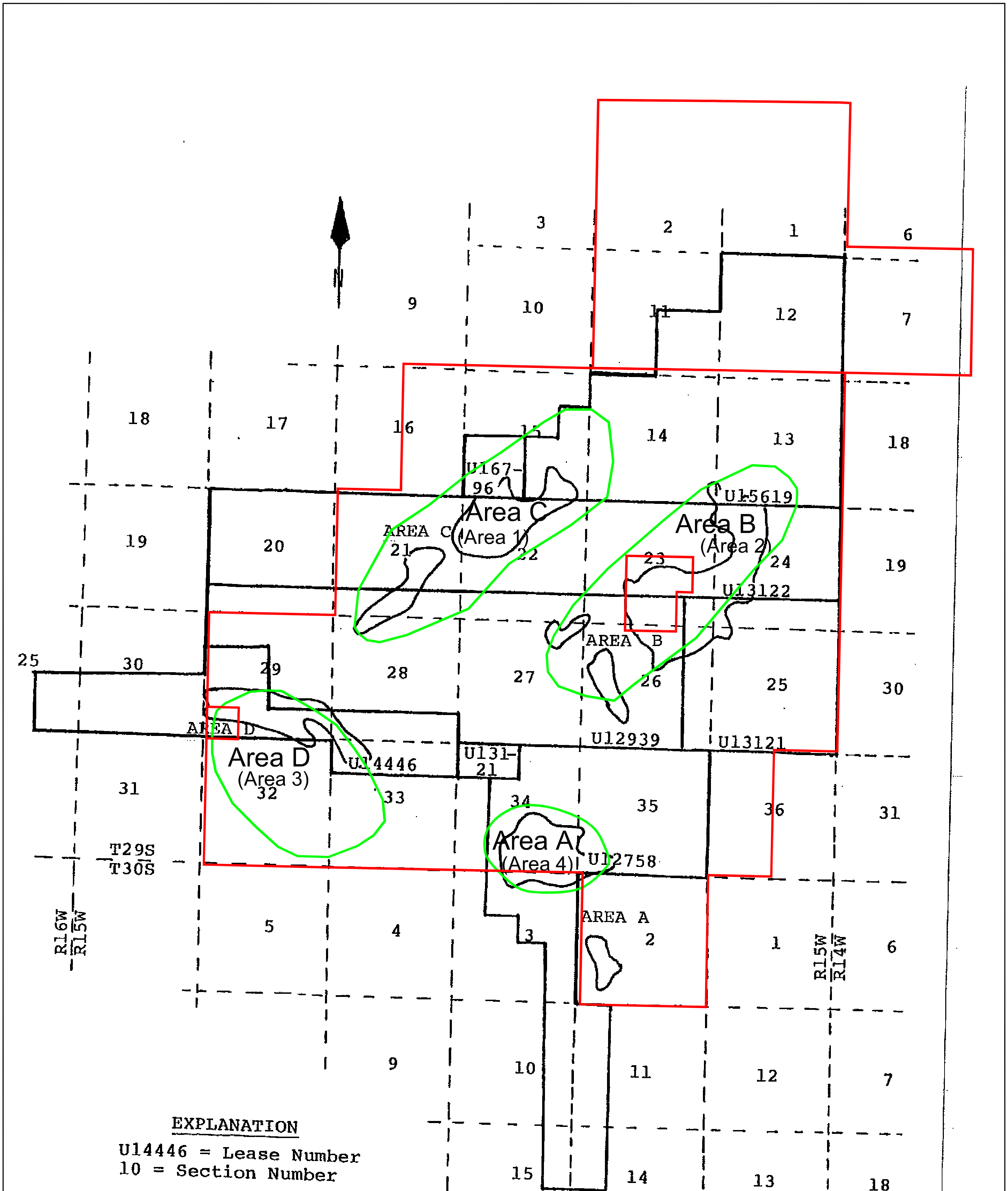
Figure 5.2

Potash Ridge Corporation
Blawn Mountain Project
Spring & Seep Survey
April-May 2013

Date: 6/25/2013
Map by: J. Cain

Scale:
1 " = 3,000 '

NORWEST CORPORATION



LEGEND

Current Blawn Mountain Property

Former NG Alunite Boundary

FIGURE 6.1

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

FORMER NG
ALUNITE PROJECT AREAS

DATE: 06/25/2013
FILE:418-5 fig 6.1
SCALE: 1"=5600'
NORWEST CORPORATION

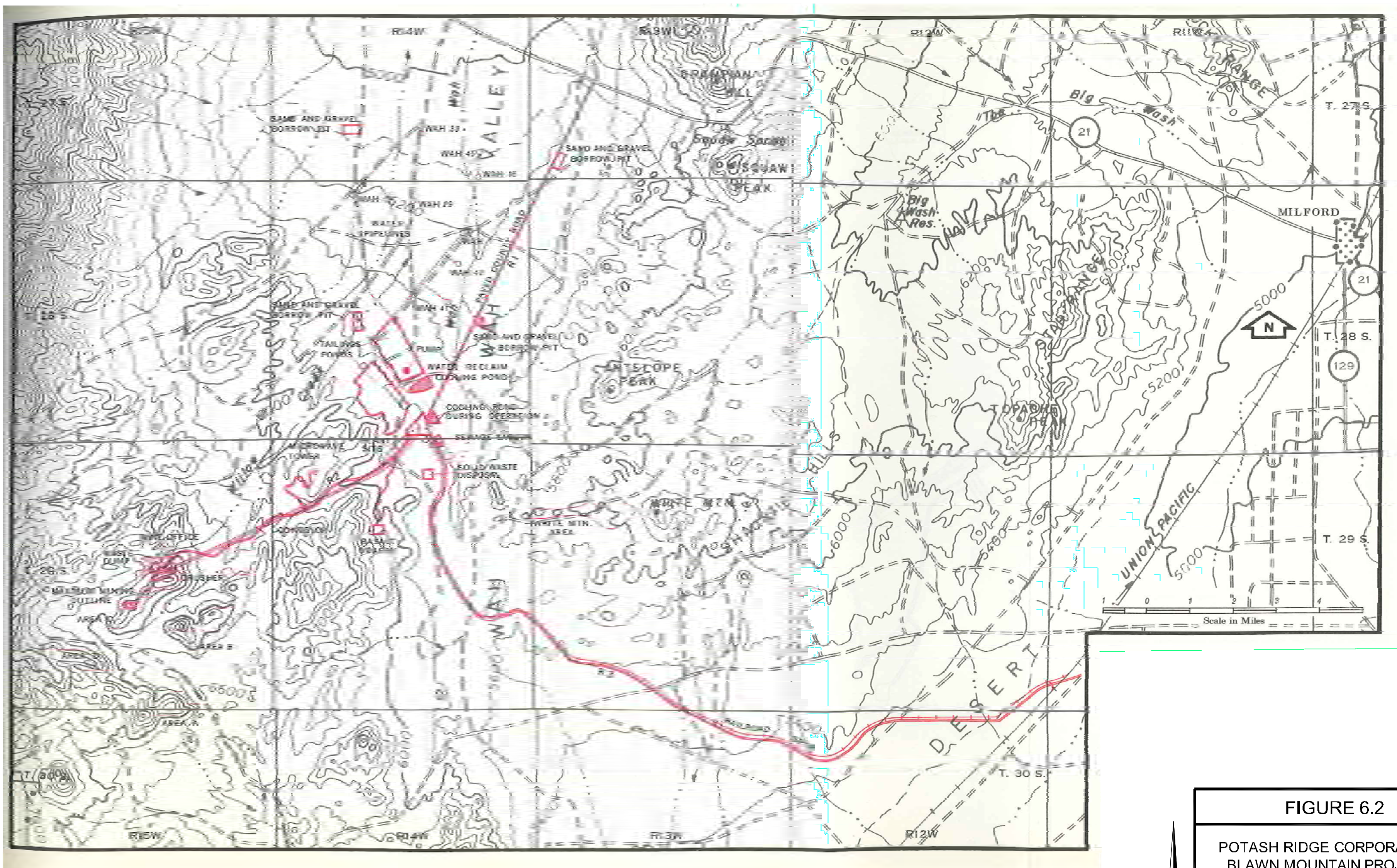
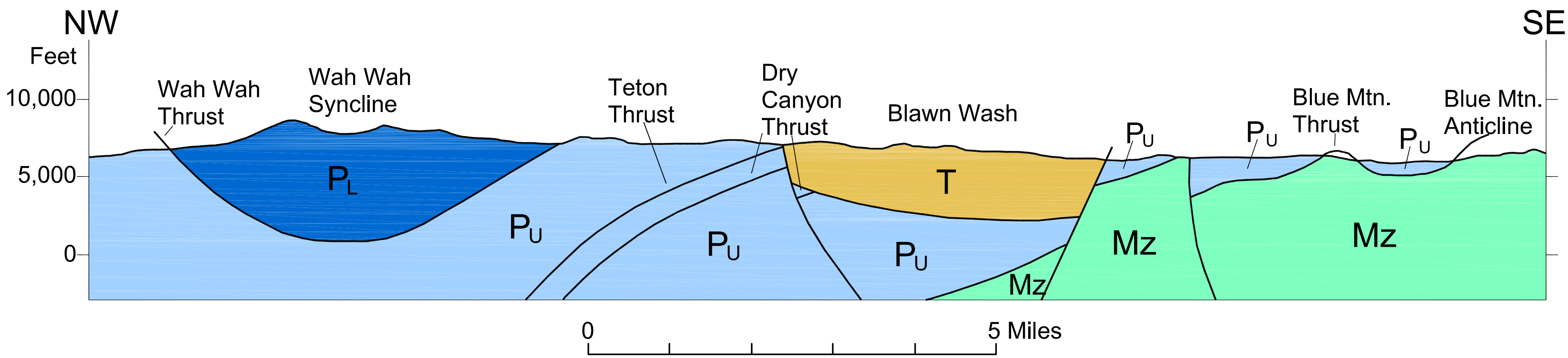


FIGURE 6.2

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

FORMER NG PROPERTY
DEVELOPMENT PLAN

DATE: 04/16/2012	PROJECT: 418-3	NORWEST CORPORATION
FILE: 418-3FIG 6.2		



T	Tertiary volcanic rocks
Mz	Mesozoic block beneath the Blue Mountain Thrust
P_U	Upper Paleozoic block between the Blue Mountain and Wah Wah Thrusts
P_L	Lower Paleozoic block above the Wah Wah Thrust

Modified from Abbot and others (1983) and Hintze and others (1994)

FIGURE 7.1

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

REGIONAL THEMATIC GEOLOGICAL CROSS SECTION

DATE: 11/20/2013
FILE: 418-5 fig7.1
PROJECT: 418-5

NORWEST CORPORATION

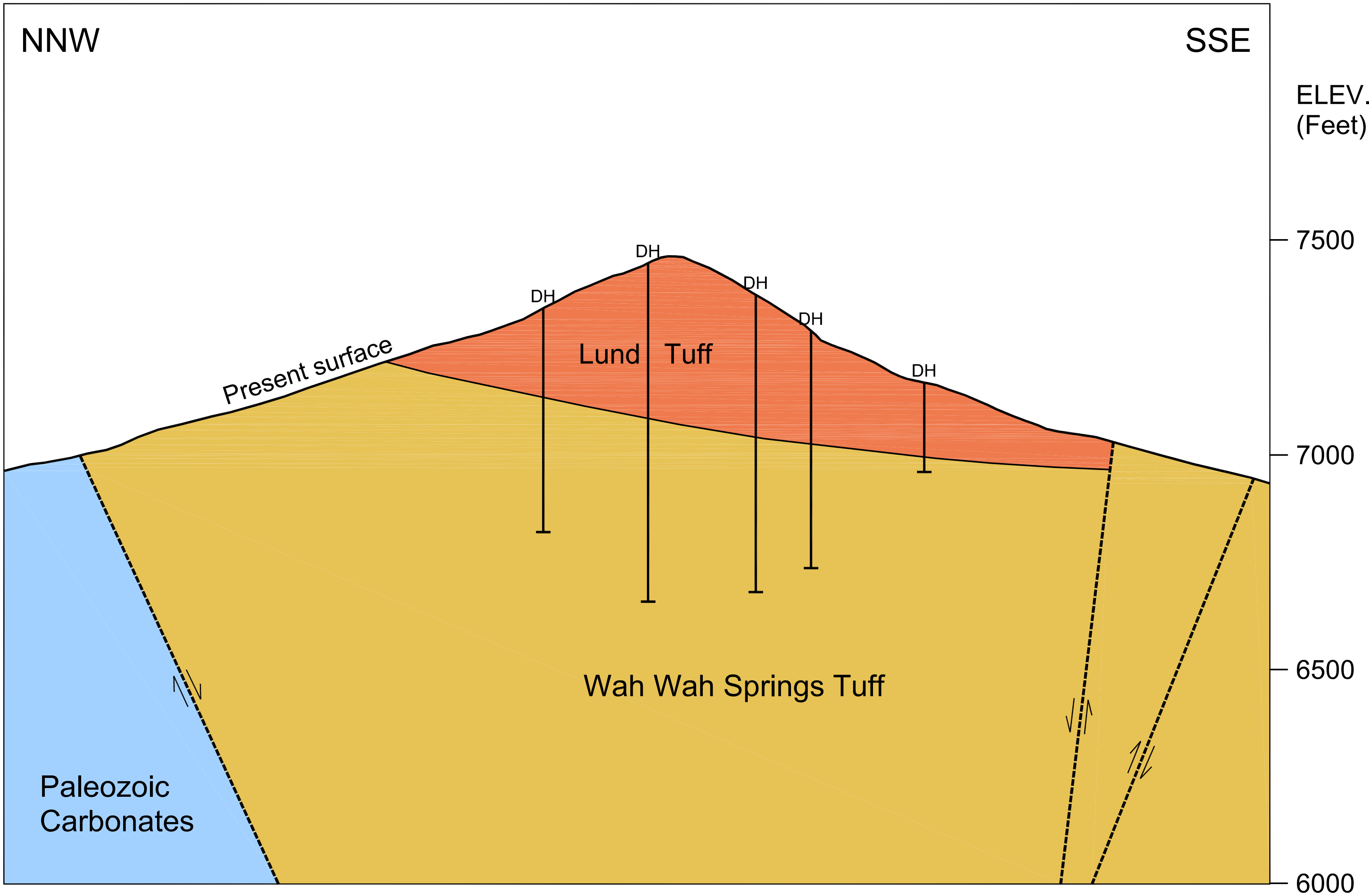


FIGURE 7.2

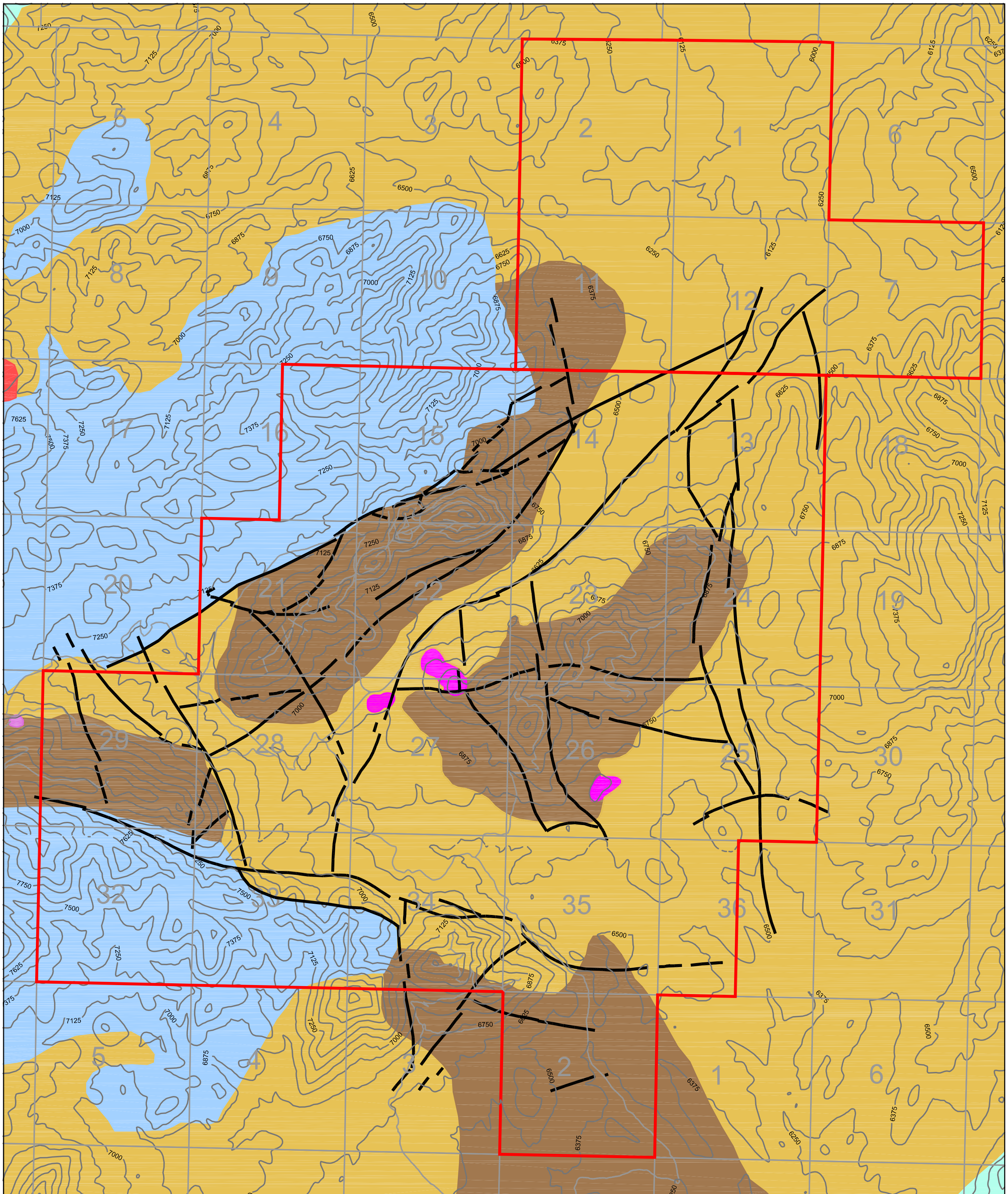
POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

DIAGRAMTIC
LOCAL GEOLOGICAL
CROSS SECTION

DATE: 04/12/2012
FILE: 418-3\XSEC

NORWEST CORPORATION

Modified from Hofstra (1984)



LEGEND

VOLCANIC ROCKS

- Trp — RHYOLITE PORPHYRY
- Tva — ALTERED IGNIMBRITES
- Tvu — UNDIFFERENTIATED VOLCANICS

SEDIMENTARY ROCKS

- Tcg — CONGLOMERATE
- Mzu — MESOZOIC SANDSTONE AND SHALE
- Pzu — PALEOZOIC CARBONATES AND QUARTZITES

- ROADS
- LEASE BOUNDARY
- NORMAL FAULTS
- THRUST FAULTS

0 2000 4000 6000

FEET

FIGURE 7.3

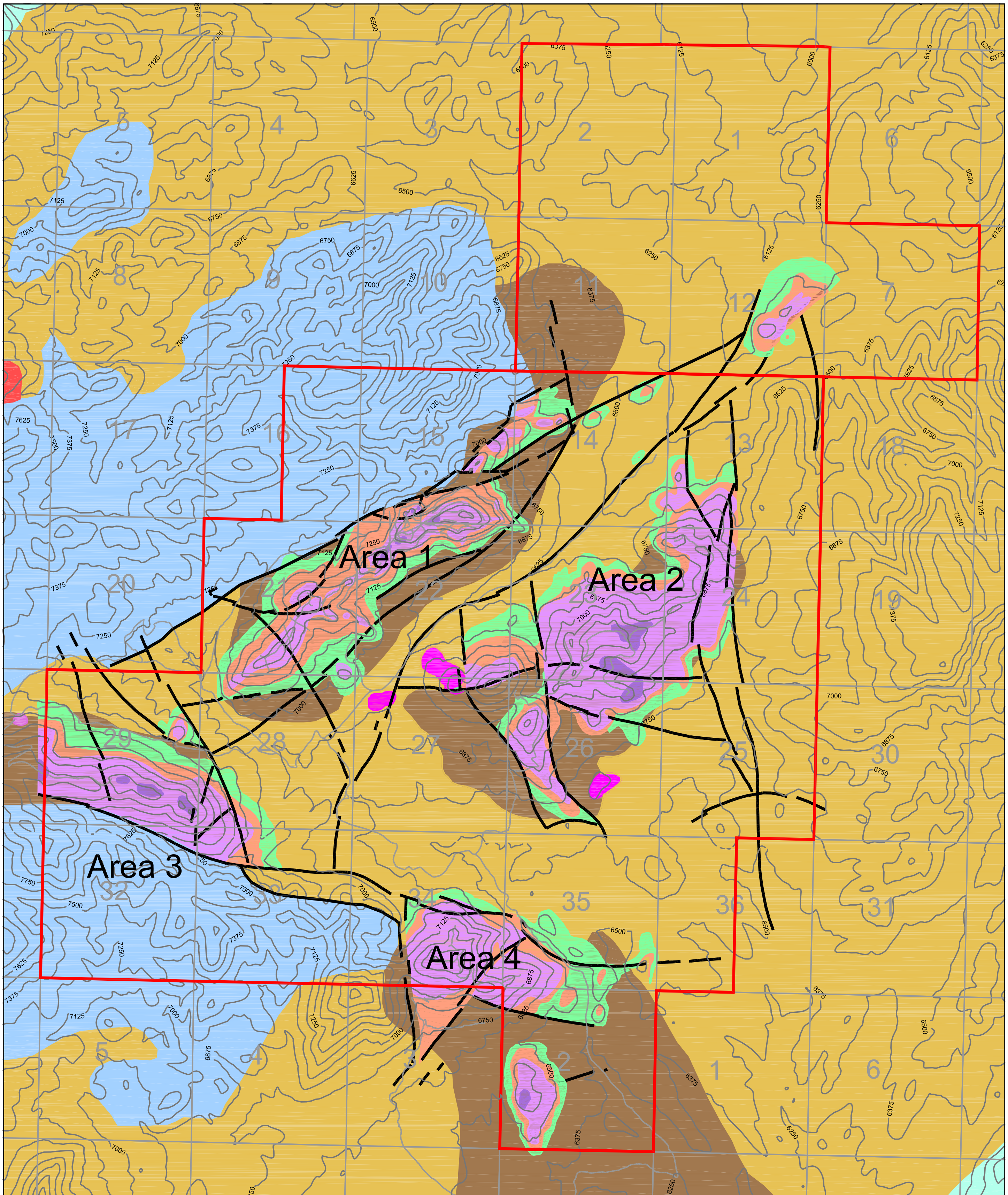
POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

SURFACE
GEOLOGY MAP

N

DATE: 11/20/2013
FILE: 418-5 fig 7.3
SCALE: 1:55,000

NORWEST CORPORATION



LEGEND

VOLCANIC ROCKS

- Trp RHYOLITE PORPHYRY
- Tva ALTERED IGNIMBRITES
- Tvu UNDIFFERENTIATED VOLCANICS

SEDIMENTARY ROCKS

- Tcg CONGLOMERATE
- Mzu MESOZOIC SANDSTONE AND SHALE
- Pzu PALEOZOIC CARBONATES AND QUARTZITES

ALTERATION

- SILICA ZONE
- QUARTZ-ALUNITE
- HEMATITE-CLAY
- PROPYLITIC ZONE

- ROADS
- LEASE BOUNDARY
- NORMAL FAULTS
- THRUST FAULTS

0 2000 4000 6000

FEET

FIGURE 7.4

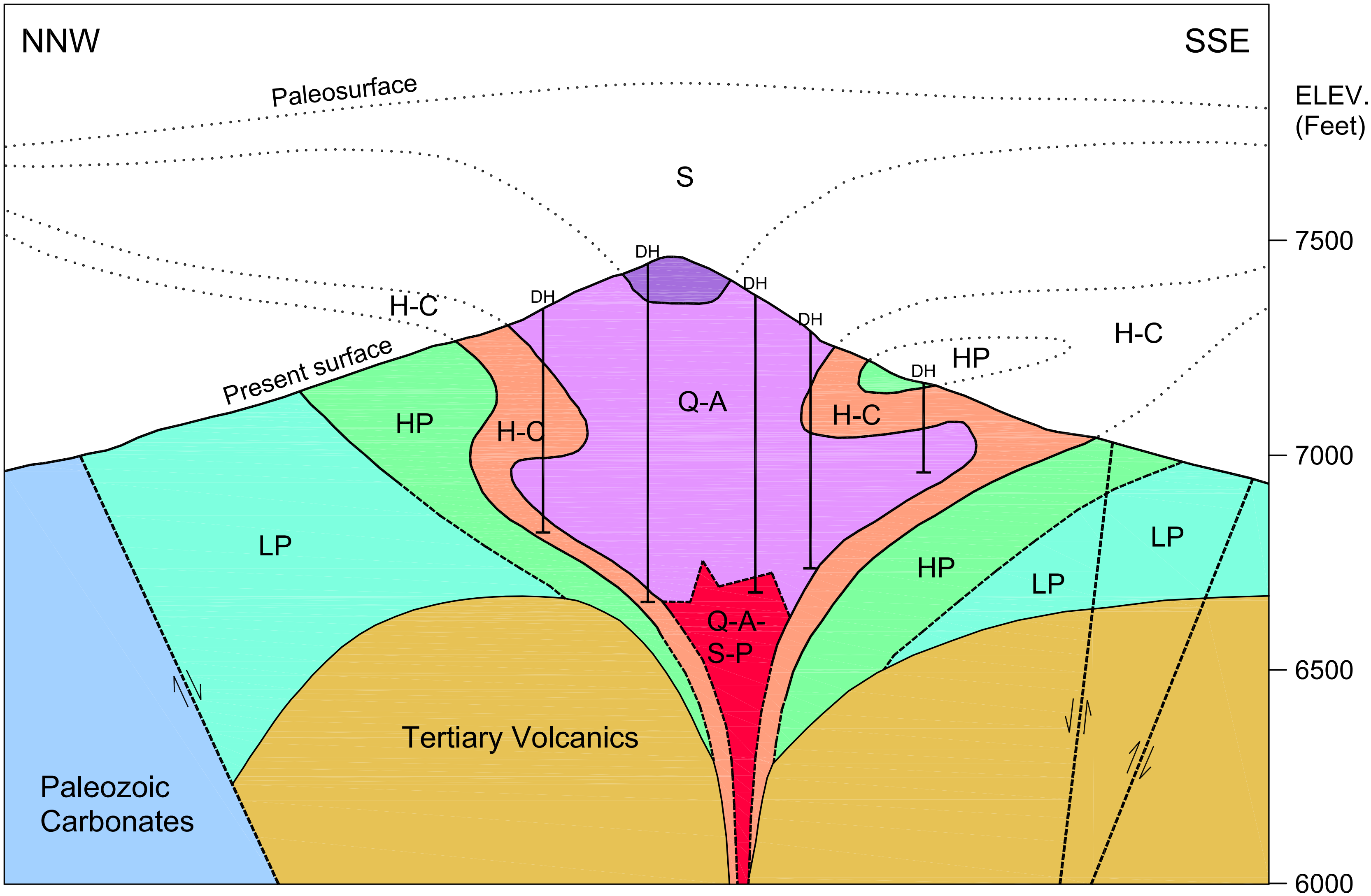
N

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

ALUNITE
ALTERATION AREAS

DATE: 11/20/2013
FILE: 418-5 fig 7.4
SCALE: 1:55,000

NORWEST CORPORATION



Q-A-S-P	Quartz-Alunite-Sericite-Pyrite Zone
Q-A	Quartz-Alunite Zone
S	Silica Zone
H-C	Hematite-Clay Zone
HP	High Propylitic Zone
LP	Low Propylitic Zone

Modified from Hofstra (1984)

FIGURE 7.5

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

DIAGRAMATIC
ALUNITE ALTERATION
CROSS SECTION

DATE: 11/20/2013
FILE: 418-3\FIG7.5

NORWEST CORPORATION

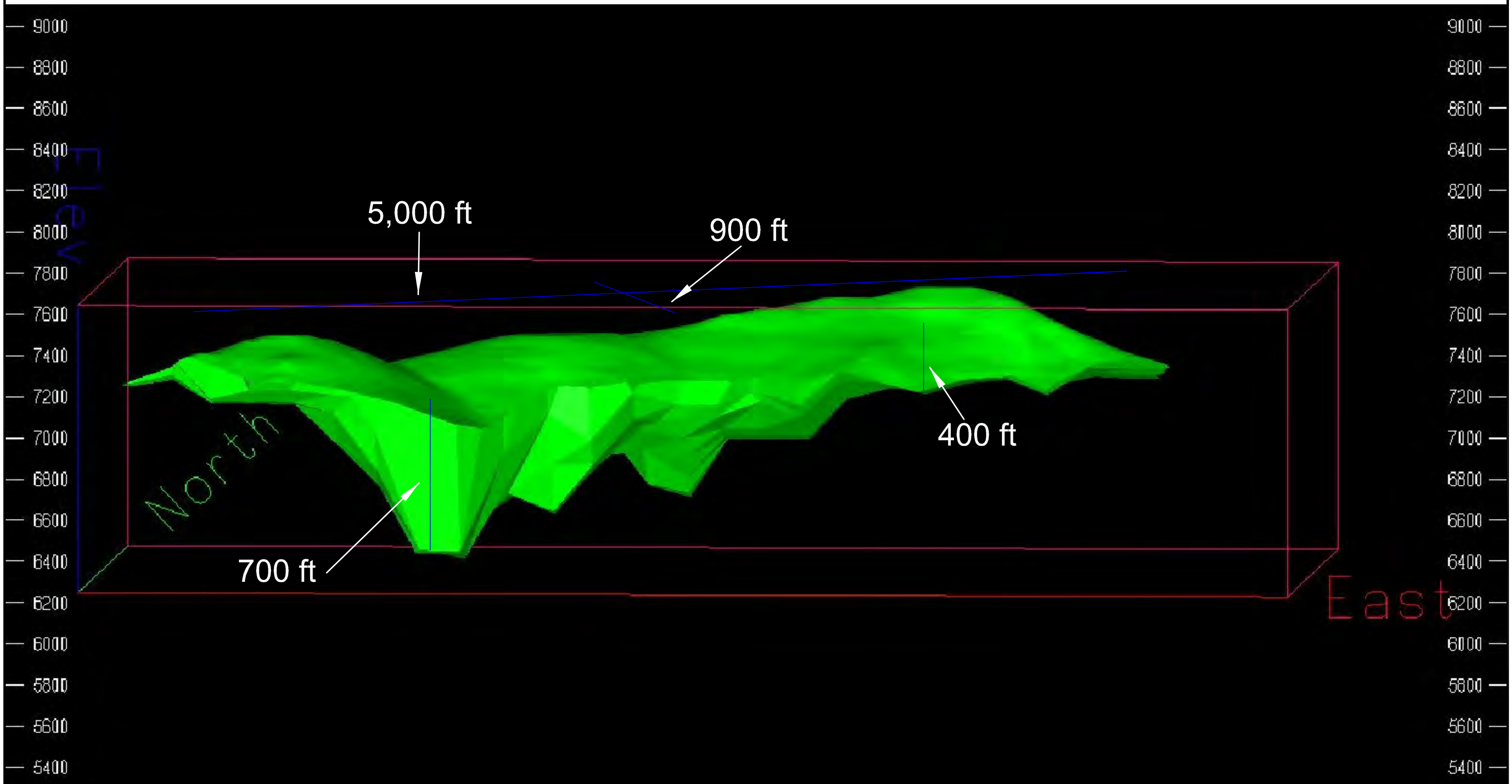


FIGURE 7.6

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

3DBM REPRESENTATION
OF AREA 1 ALUNITE ZONE

DATE: 06/25/2013	SCALE:	NORWEST CORPORATION
FILE: 418-5 fig 7.7	as noted	

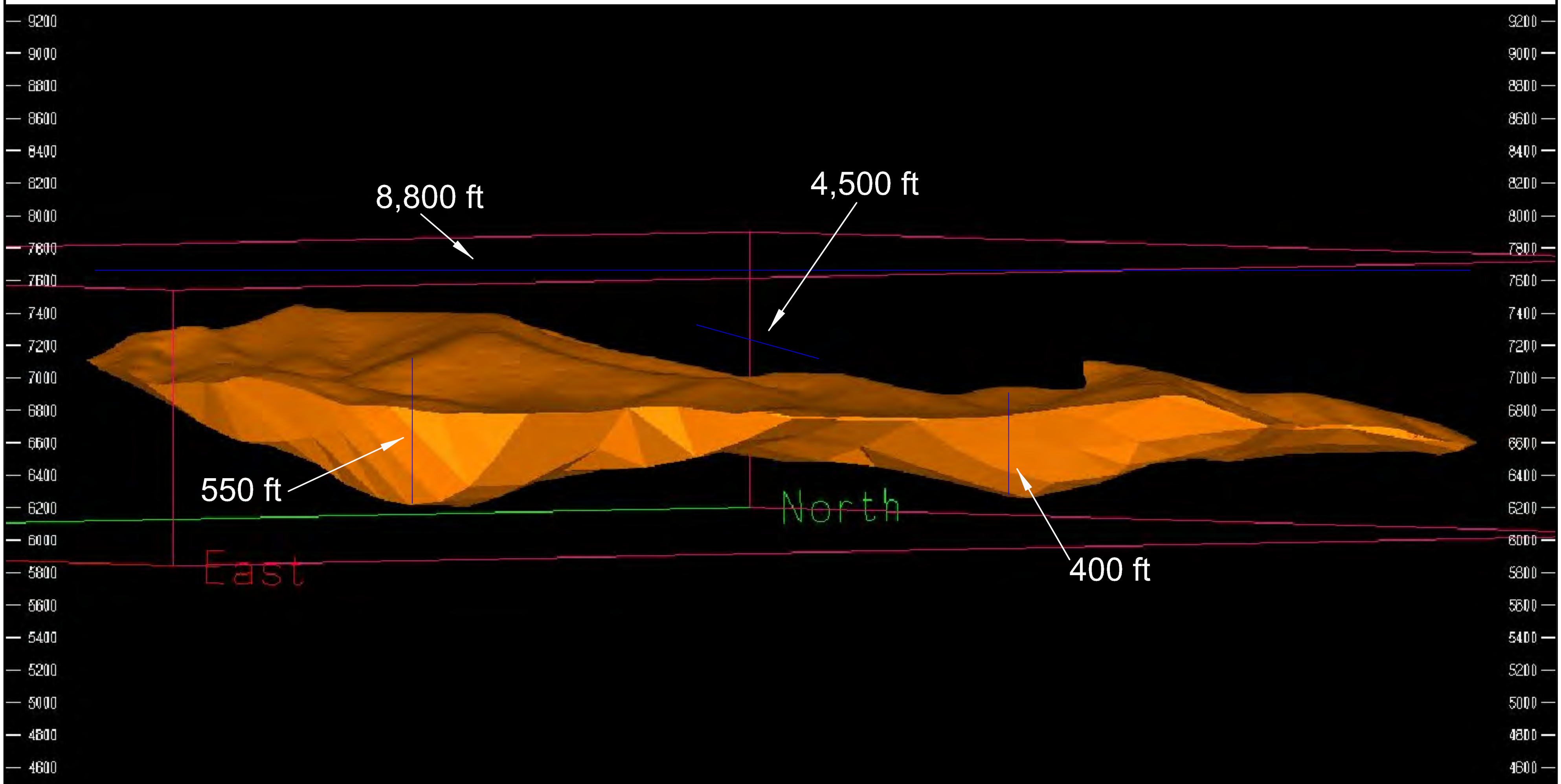


FIGURE 7.7

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

3DBM REPRESENTATION
OF AREA 2 ALUNITE ZONE

DATE: 06/25/2013	SCALE:
FILE: 418-5 fig 7.7	as noted

NORWEST CORPORATION

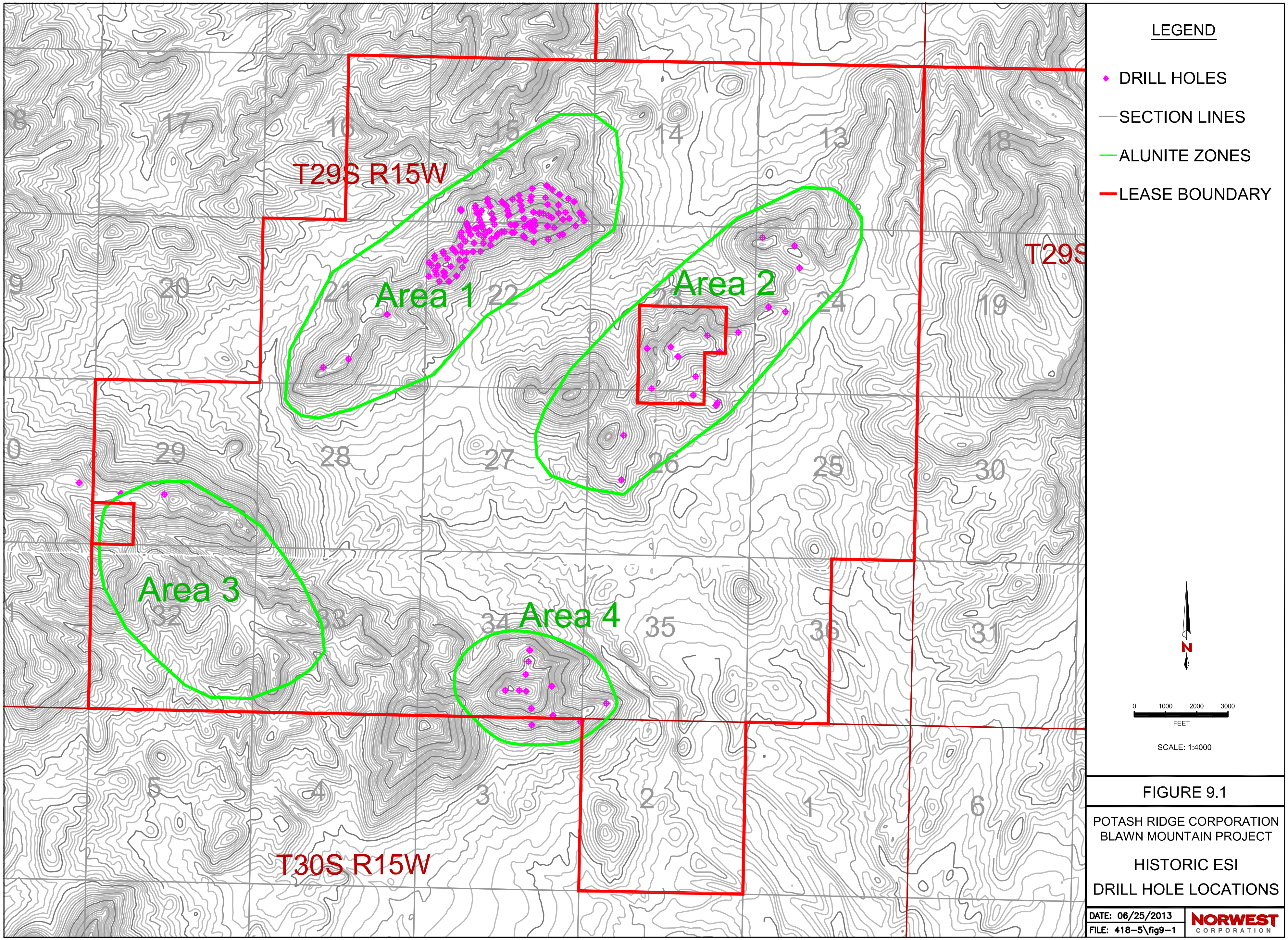
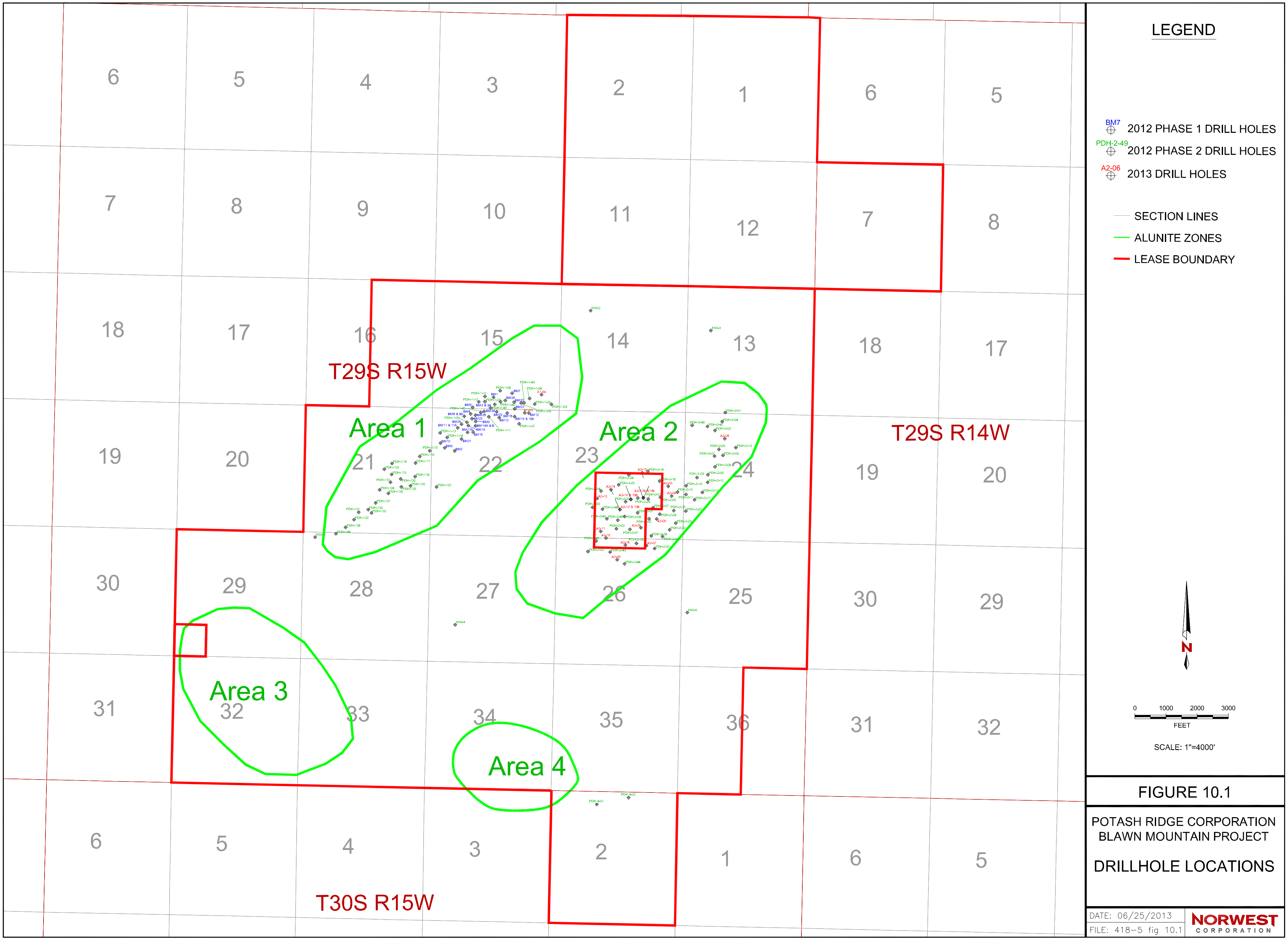


FIGURE 9.1

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

HISTORIC ESI
DRILL HOLE LOCATIONS


T29S R15W
T29S R14W
T30S R15W
Area 1
Area 2
Area 3
Area 4
6
5
4
3
2
1
6
5
7
8
9
10
11
12
7
8
18
17
16
15
14
13
18
17
19
20
21
22
23
24
19
20
30
29
28
27
26
25
30
29
31
32
33
34
35
36
31
32
6
5
4
3
2
1
6
5
LEGEND
BM7
2012 PHASE 1 DRILL HOLES
PDH-2-49
2012 PHASE 2 DRILL HOLES
A2-06
2013 DRILL HOLES
SECTION LINES
ALUNITE ZONES
LEASE BOUNDARY
N
0
1000
2000
3000
FEET
SCALE: 1"=4000'
FIGURE 10.1
POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT
DRILLHOLE LOCATIONS
DATE: 06/25/2013
FILE: 418-5 fig 10.1
NORWEST
CORPORATION

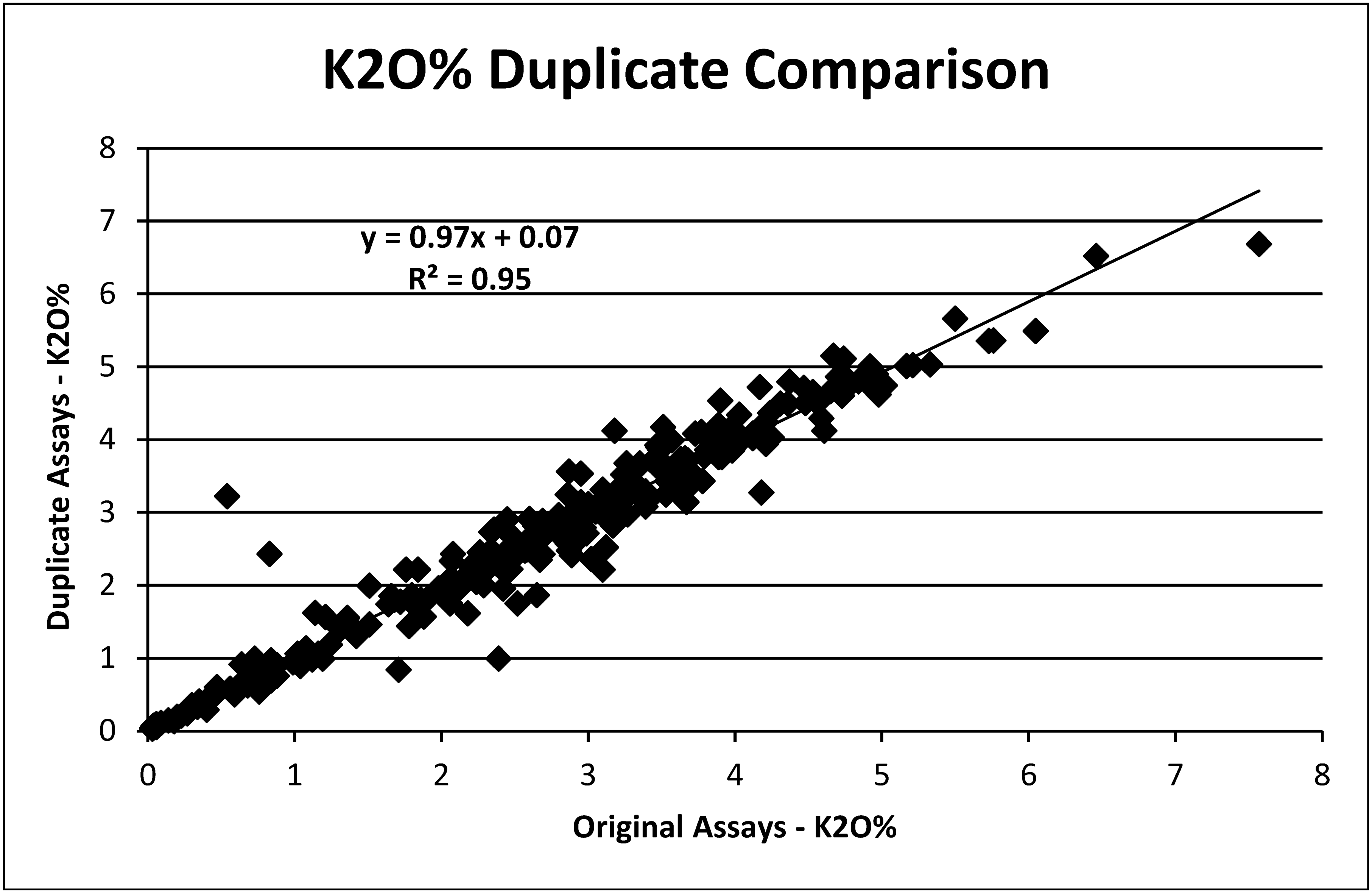
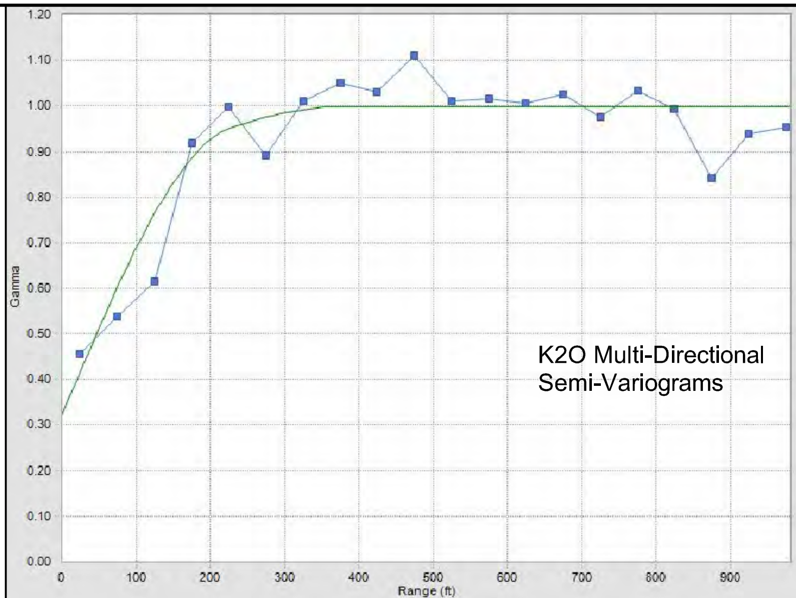


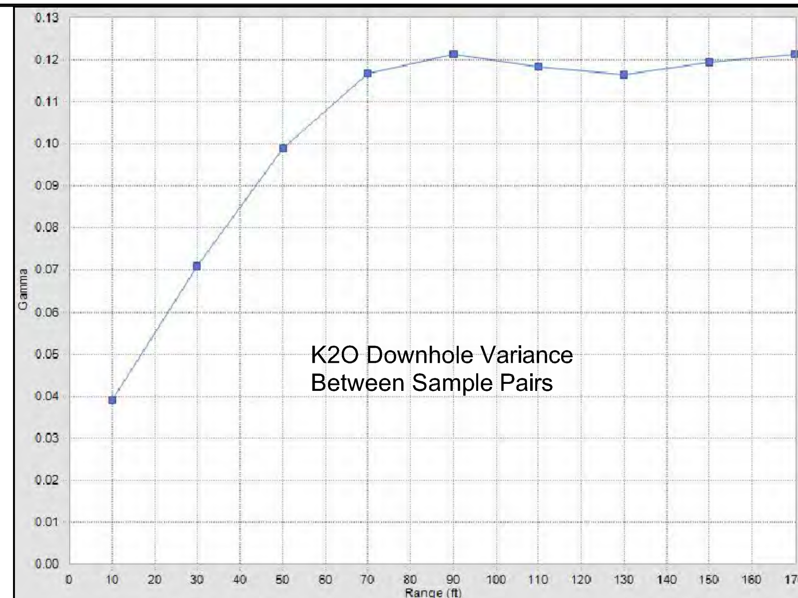
FIGURE 12.1

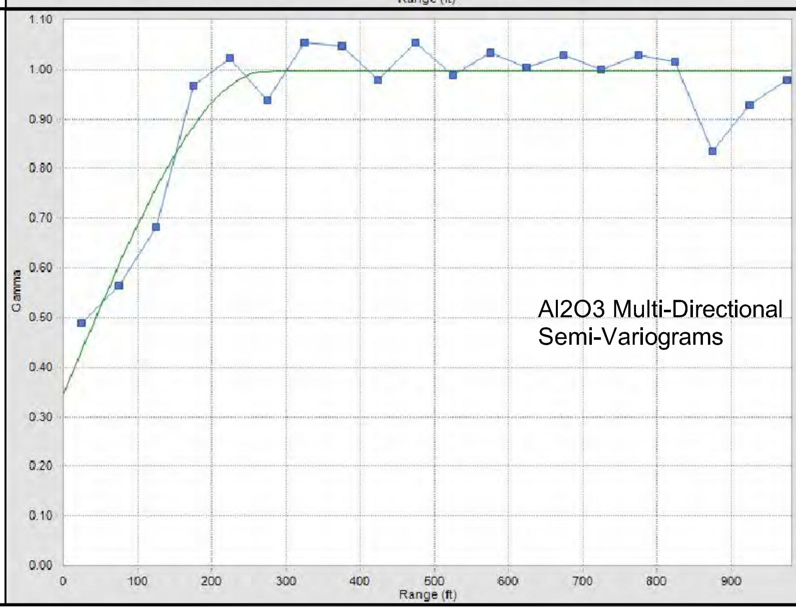
POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

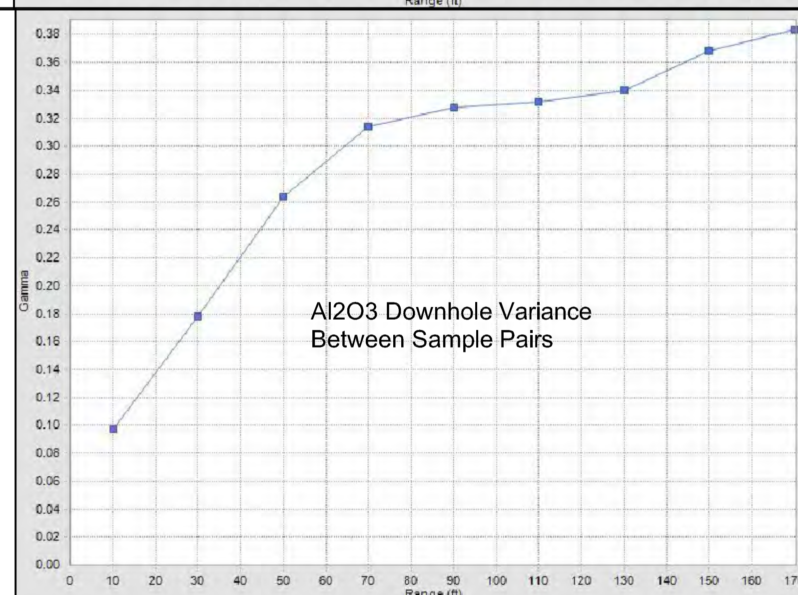
K2O COMPARISONS FOR ORIGINAL and DUPLICATE ASSAYS

DATE: 06/06/2013	SCALE:	NORWEST CORPORATION
FILE: 418-5/fig12.1	as noted	









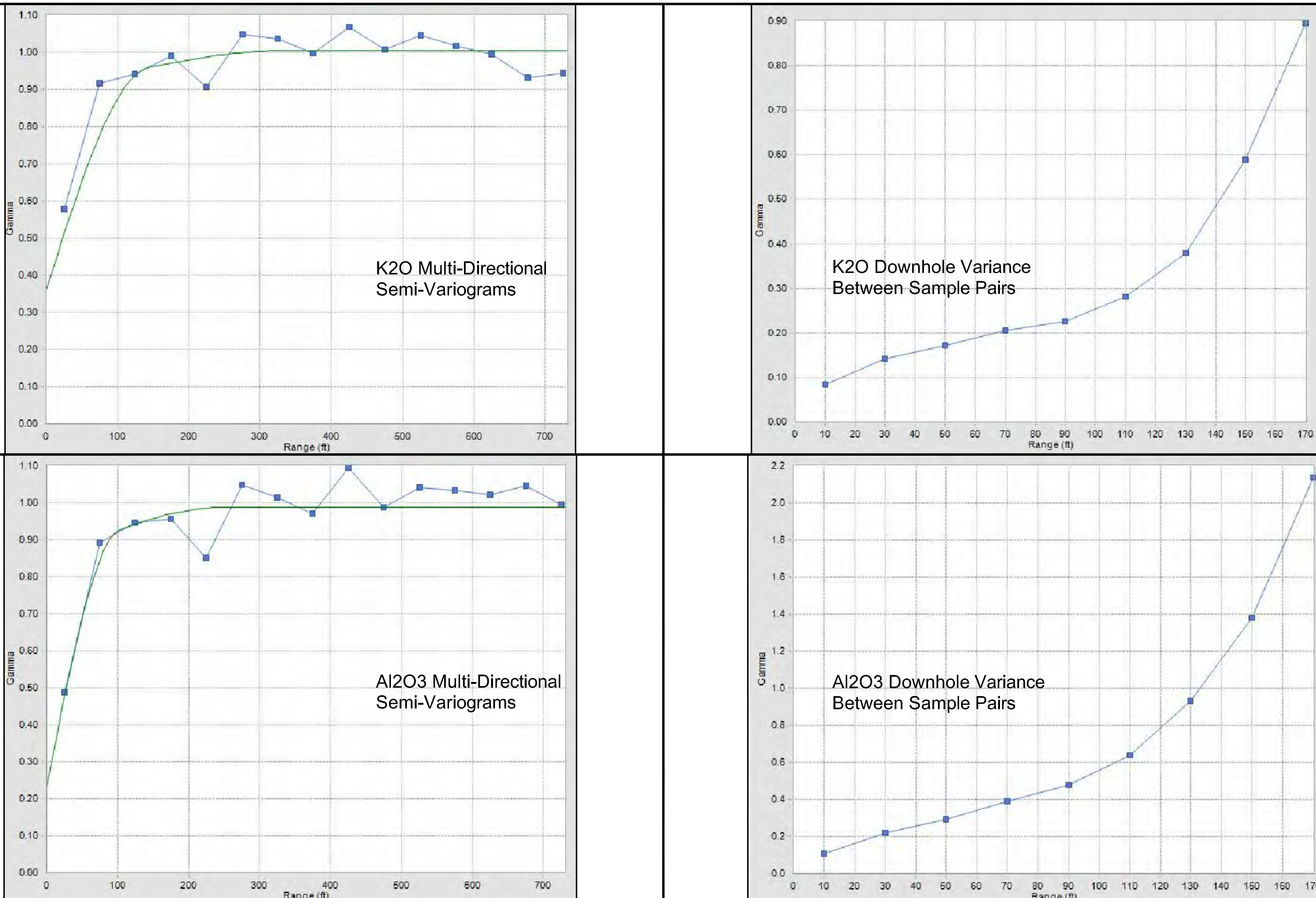
Best Fit Experimental
Semi-Variograms (Spherical Model)

FIGURE14.1

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

AREA 1 VARIOGRAMS OF
ANALYTICAL RESULTS

DATE: 06/03/2013	SCALE:	NORWEST CORPORATION
FILE: 418–5 figures	as noted	



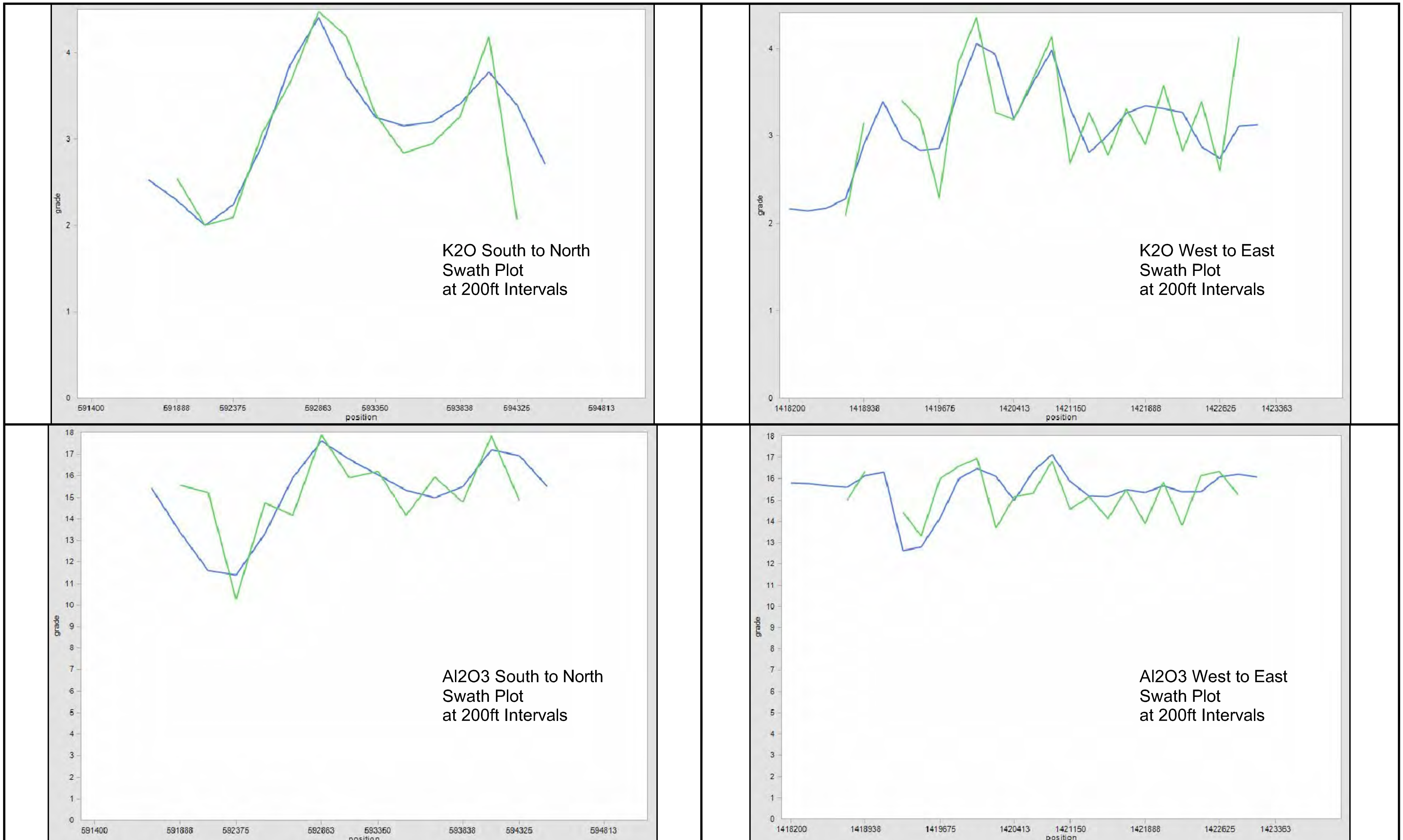
Best Fit Experimental
Semi-Variograms (Spherical Model)

FIGURE14.2

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

AREA 2 VARIOGRAMS OF
ANALYTICAL RESULTS

DATE: 06/03/2013	SCALE:	NORWEST CORPORATION
FILE: 418–5 figures	as noted	



Drillhole Data

Block Model Data

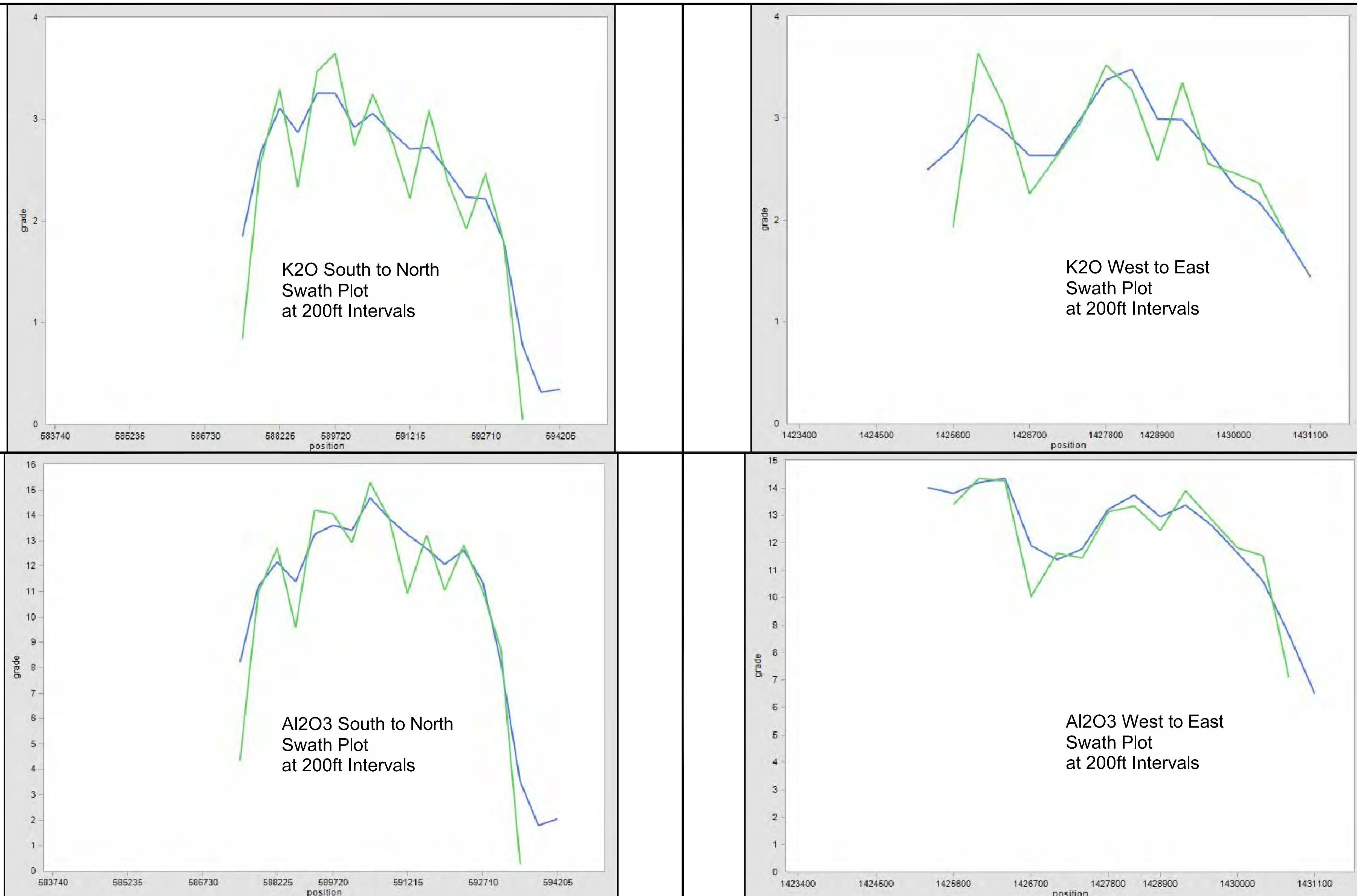
FIGURE 14.3

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

AREA 1 VALIDATION

SWATH PLOTS

DATE: 06/03/2013	SCALE:	NORWEST CORPORATION
FILE: 418-5 figures	as noted	



Drillhole Data

Block Model Data

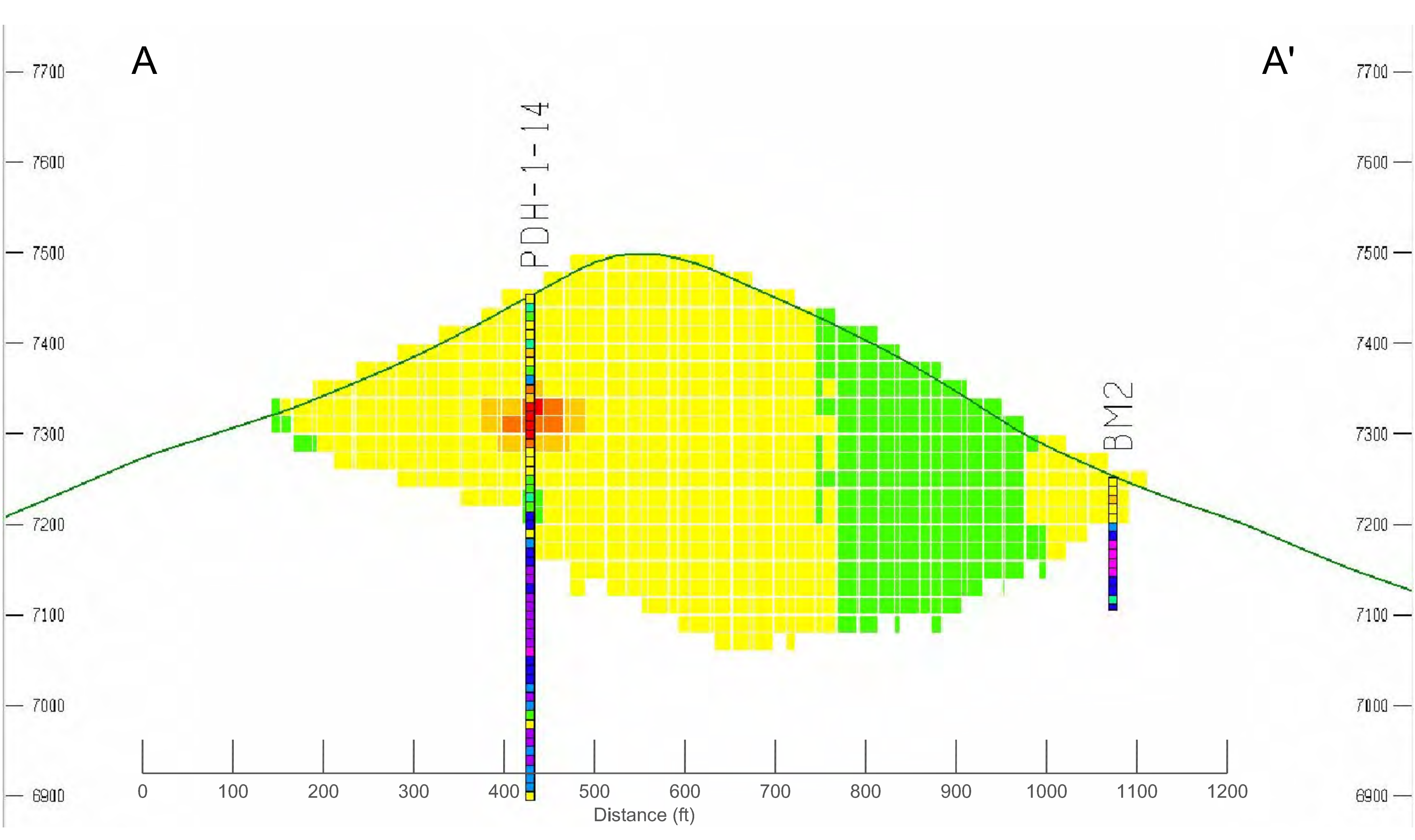
FIGURE 14.4

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

AREA 2 VALIDATION

SWATH PLOTS

DATE: 06/03/2013	SCALE:	NORWEST CORPORATION
FILE: 418-5 figures	as noted	



Drillholes displayed can be up to 100 feet from section

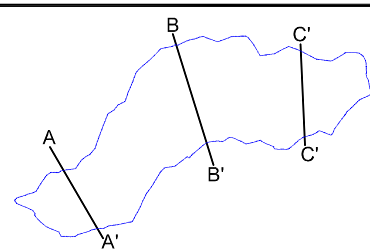


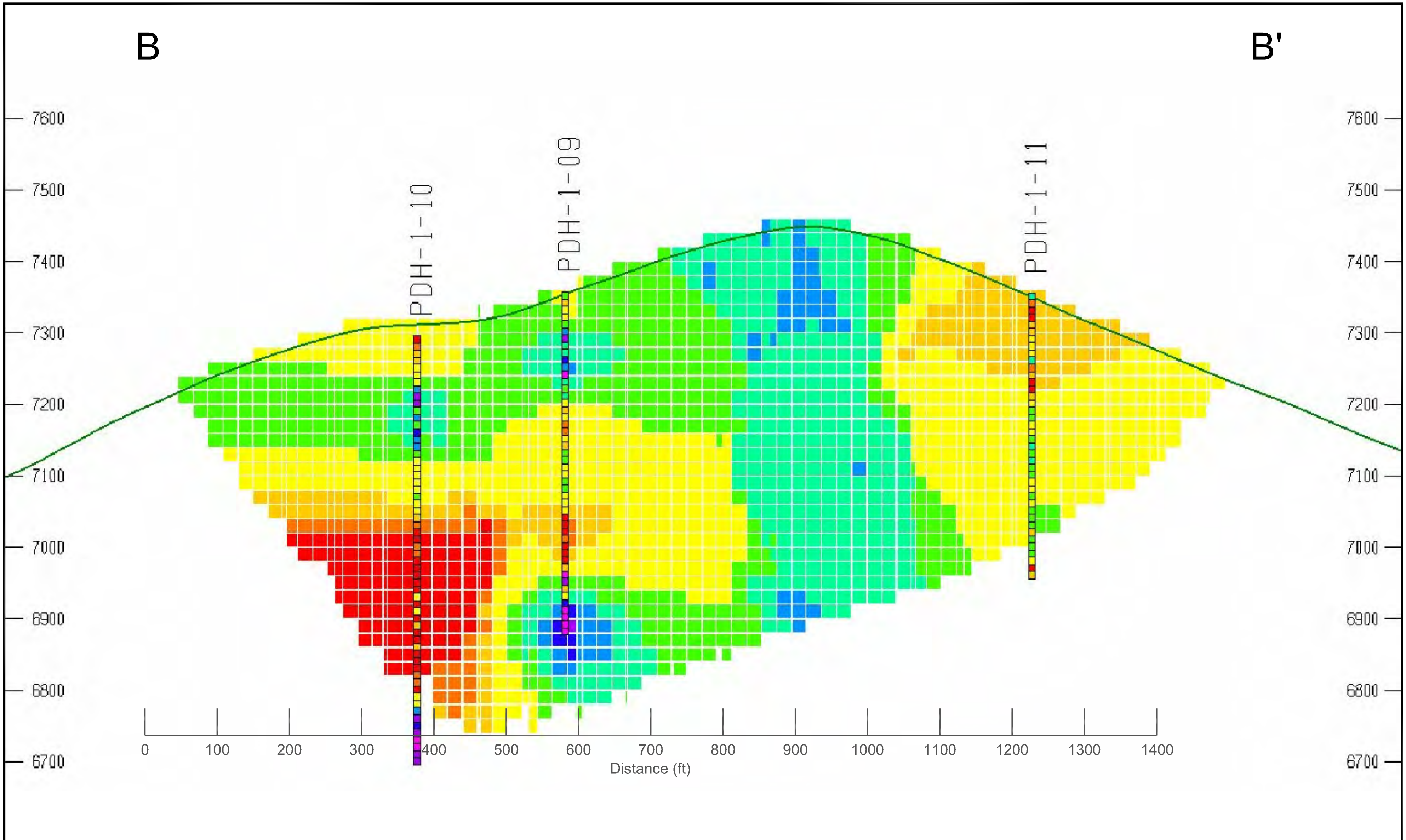
FIGURE 14.5

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

AREA 1 BLOCK MODEL

CROSS-SECTION A-A'

DATE: 06/03/2013	SCALE:	NORWEST CORPORATION
FILE: 418-5 figures	as noted	



K2O%

0
0.5
1
1.5
2
2.5
3
3.5
4
4.5
5

Drillholes displayed can be
up to 100 feet from section

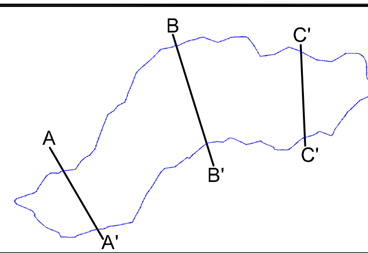


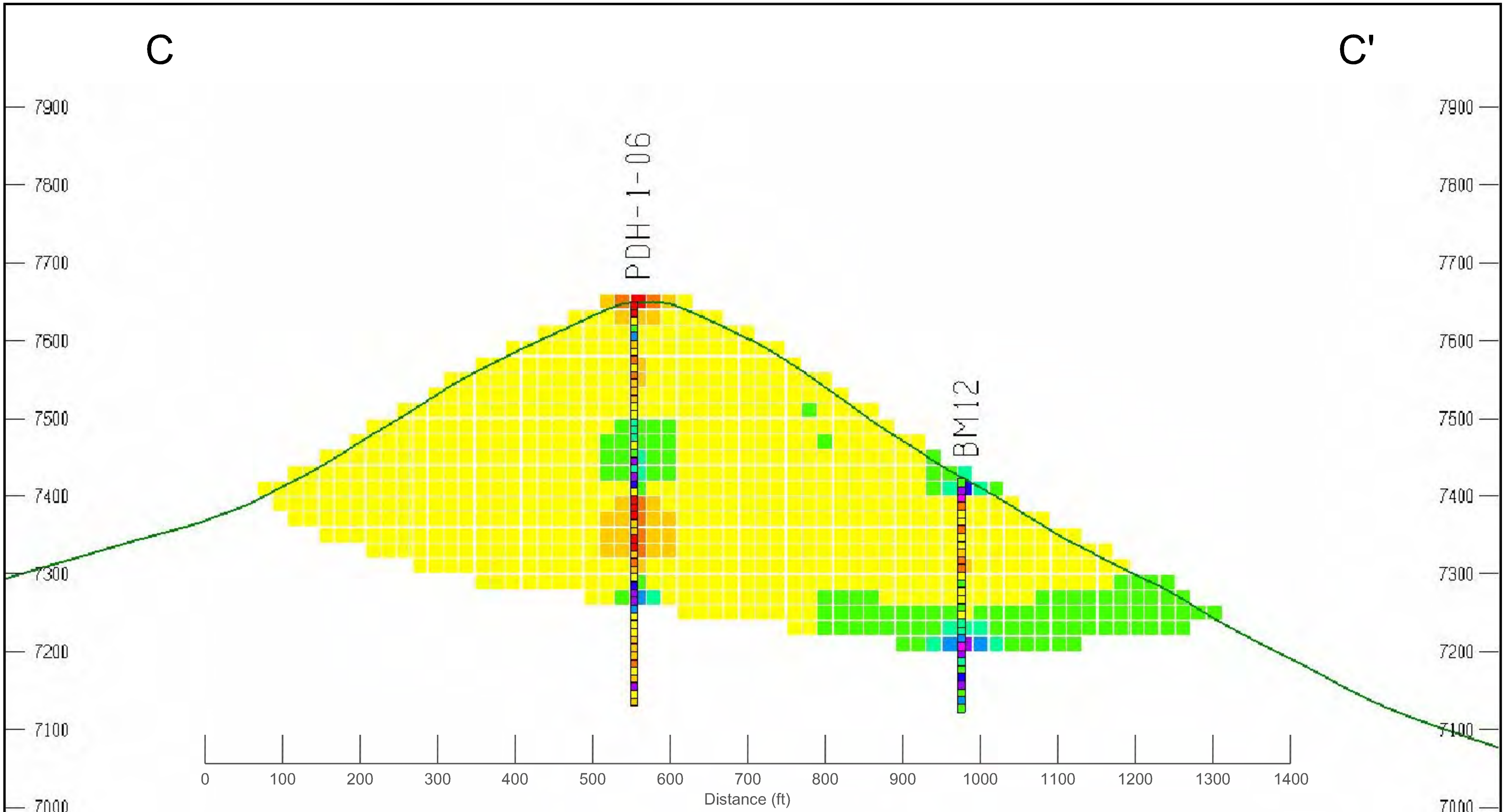
FIGURE 14.6

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

AREA 1 BLOCK MODEL

CROSS-SECTION B-B'

DATE: 06/03/2013	SCALE:	NORWEST CORPORATION
FILE: 418-5 figures	as noted	



K2O%

0
0.5
1
1.5
2
2.5
3
3.5
4
4.5
5

Drillholes displayed can be
up to 100 feet from section

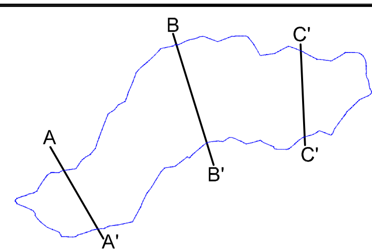


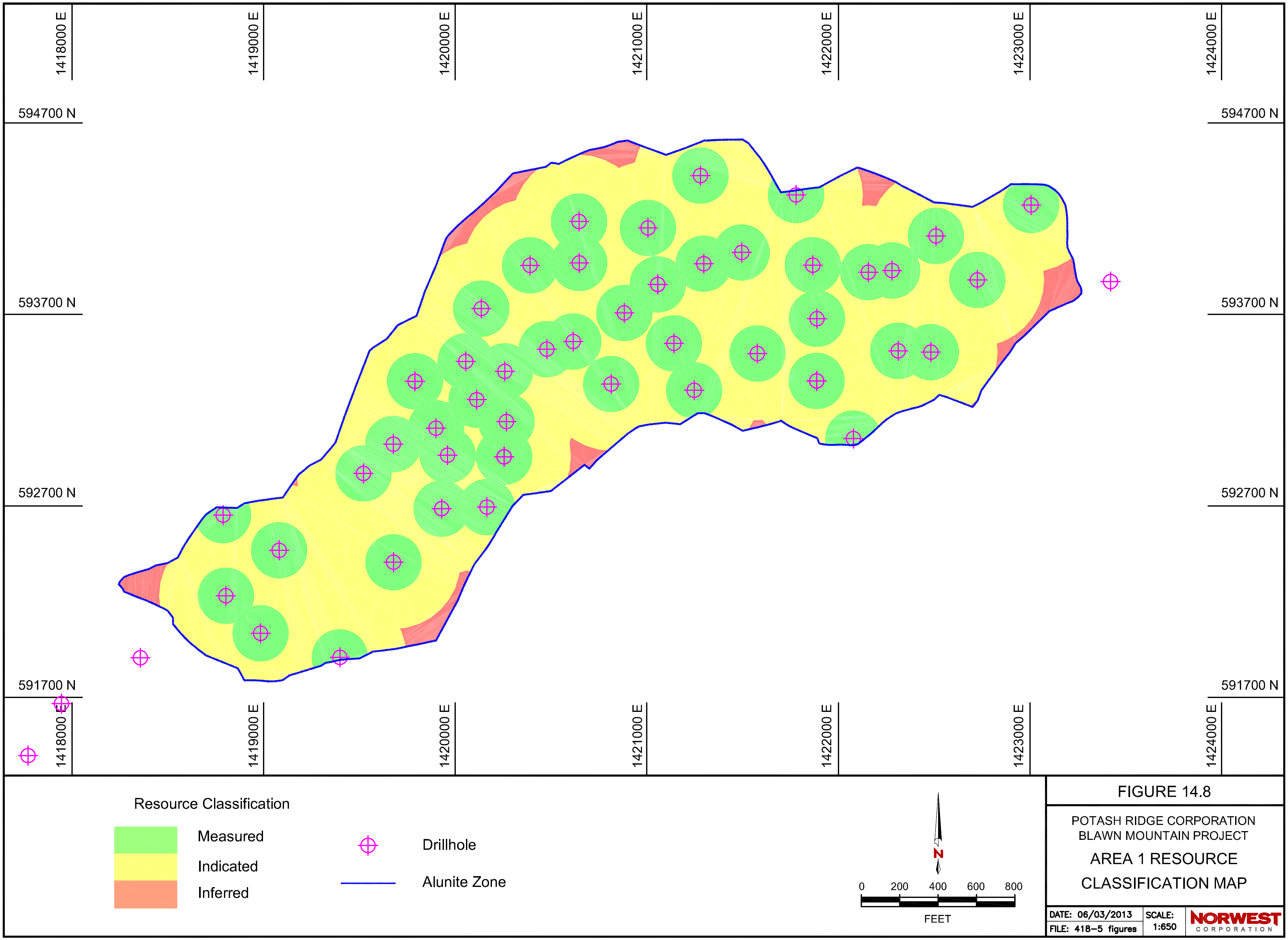
FIGURE 14.7

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

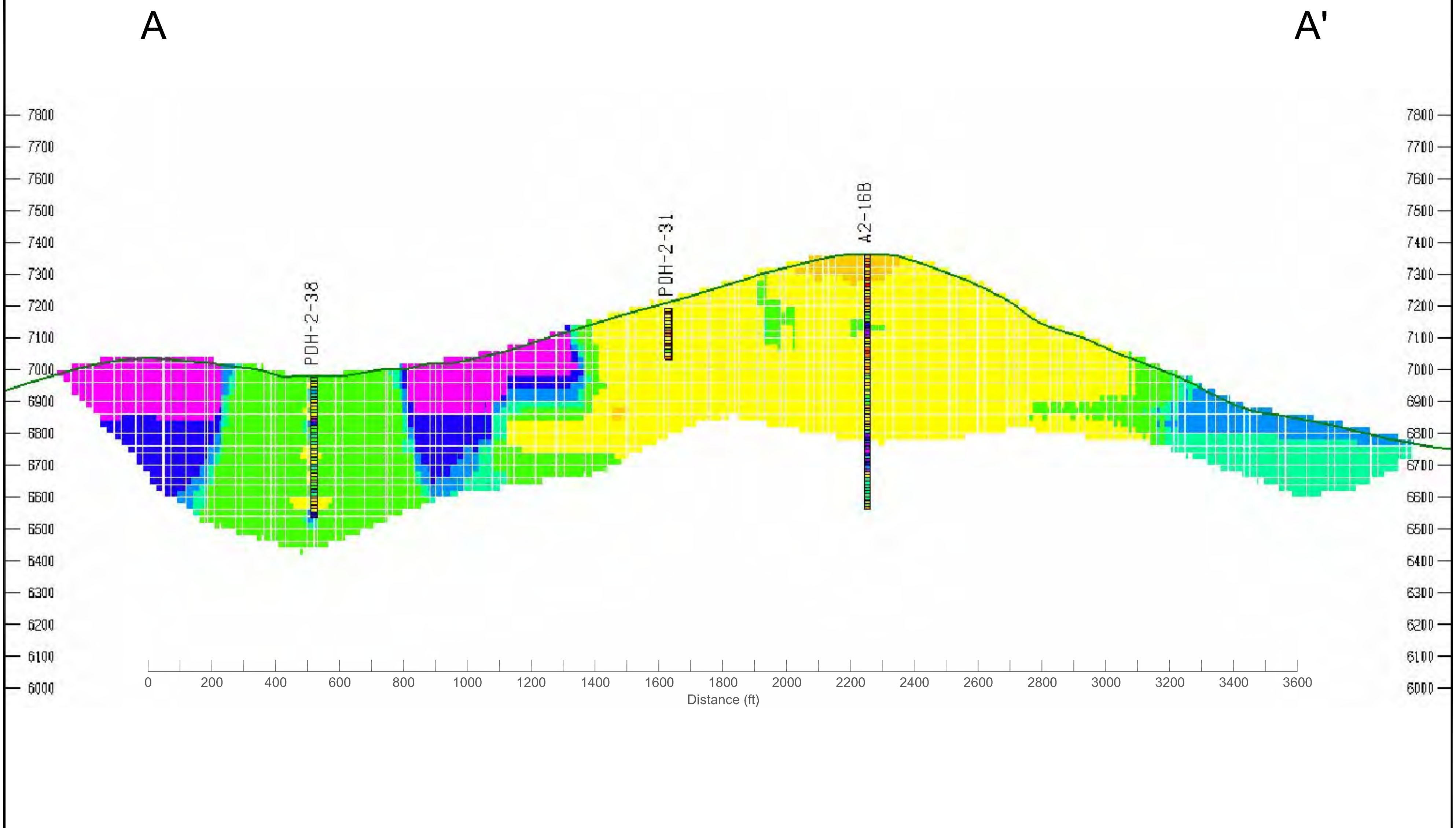
AREA 1 BLOCK MODEL

CROSS-SECTION C-C'

DATE: 06/03/2013	SCALE:	NORWEST CORPORATION
FILE: 418-5 figures	as noted	



1418000 E
1419000 E
1420000 E
1421000 E
1422000 E
1423000 E
1424000 E
594700 N
593700 N
592700 N
591700 N
Resource Classification
Measured
Indicated
Inferred
Drillhole
Alunite Zone
N
0
200
400
600
800
FEET
FIGURE 14.8
POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT
AREA 1 RESOURCE
CLASSIFICATION MAP
DATE: 06/03/2013
FILE: 418-5 figures
SCALE: 1:650
NORWEST
CORPORATION



Drillholes displayed can be up to 100 feet from section

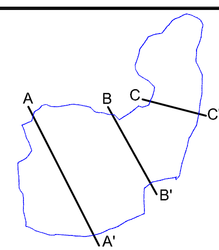


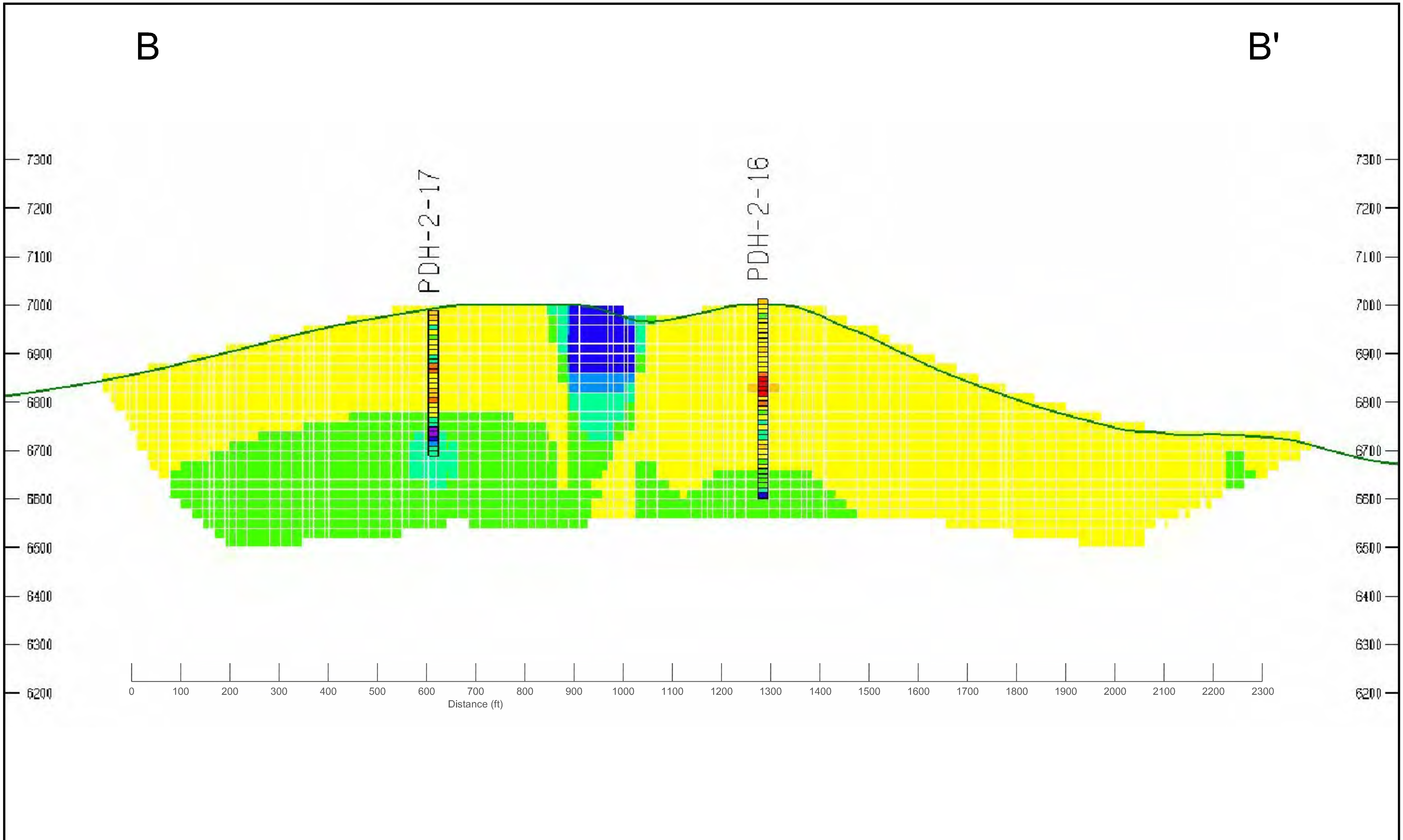
FIGURE 14.9

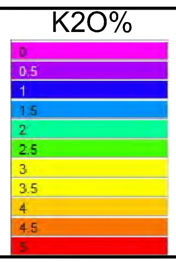
POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

AREA 2 BLOCK MODEL

CROSS-SECTION A-A'

DATE: 06/03/2013	SCALE:	NORWEST CORPORATION
FILE: 418-5 figures	as noted	





Drillholes displayed can be
up to 100 feet from section

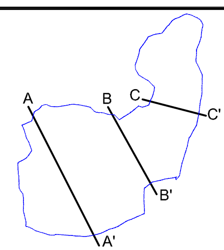
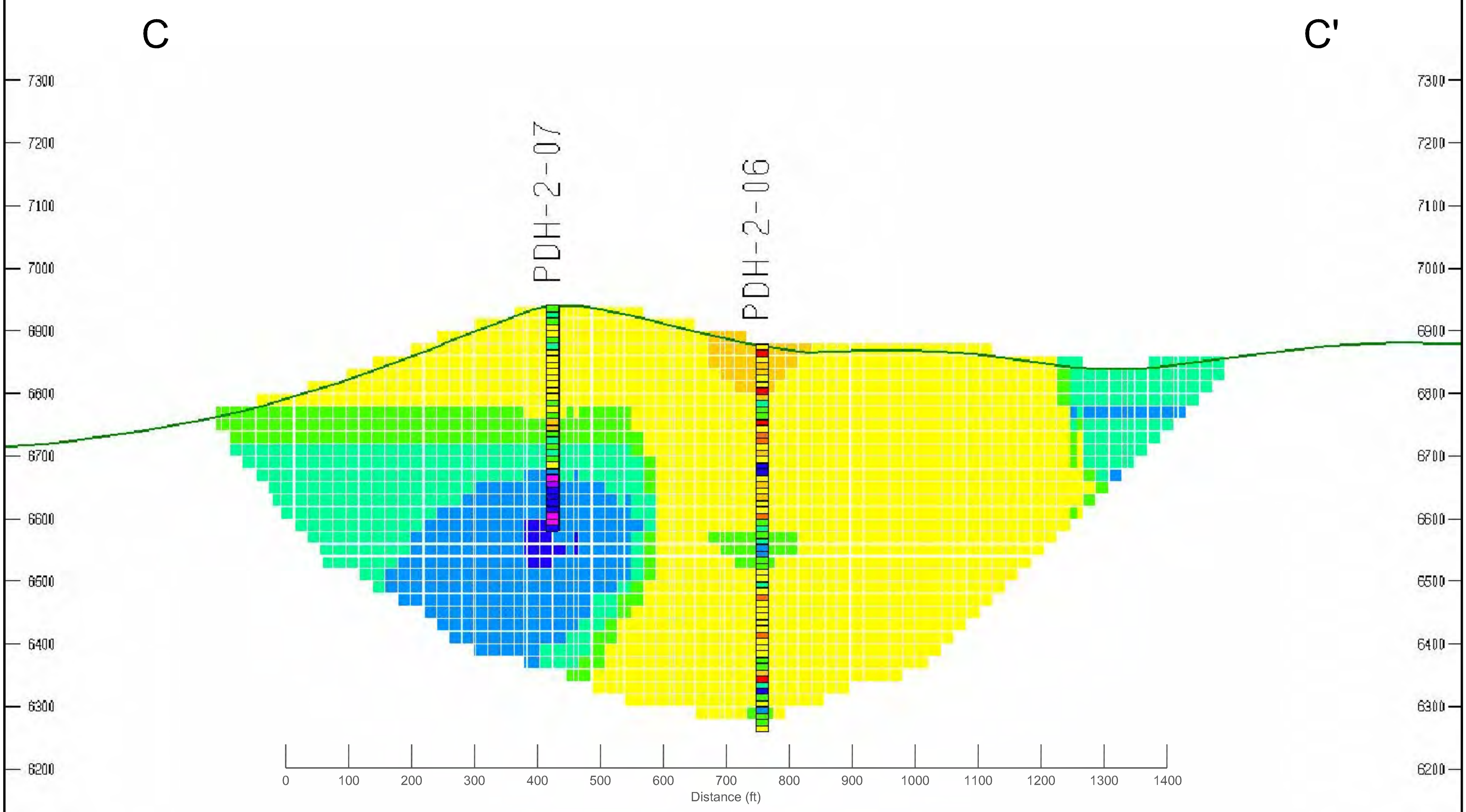


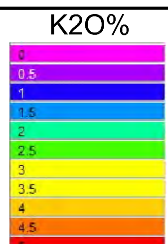
FIGURE 14.10

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT
AREA 2 BLOCK MODEL
CROSS-SECTION B-B'

DATE: 06/03/2013 | SCALE: as noted
FILE: 418-5 figures

NORWEST CORPORATION





Drillholes displayed can be up to 100 feet from section

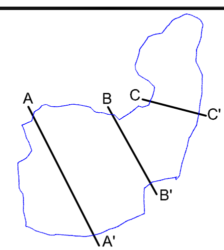


FIGURE 14.11

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

AREA 2 BLOCK MODEL

CROSS-SECTION C-C'

DATE: 06/03/2013	SCALE:	NORWEST CORPORATION
FILE: 418-5 figures	as noted	

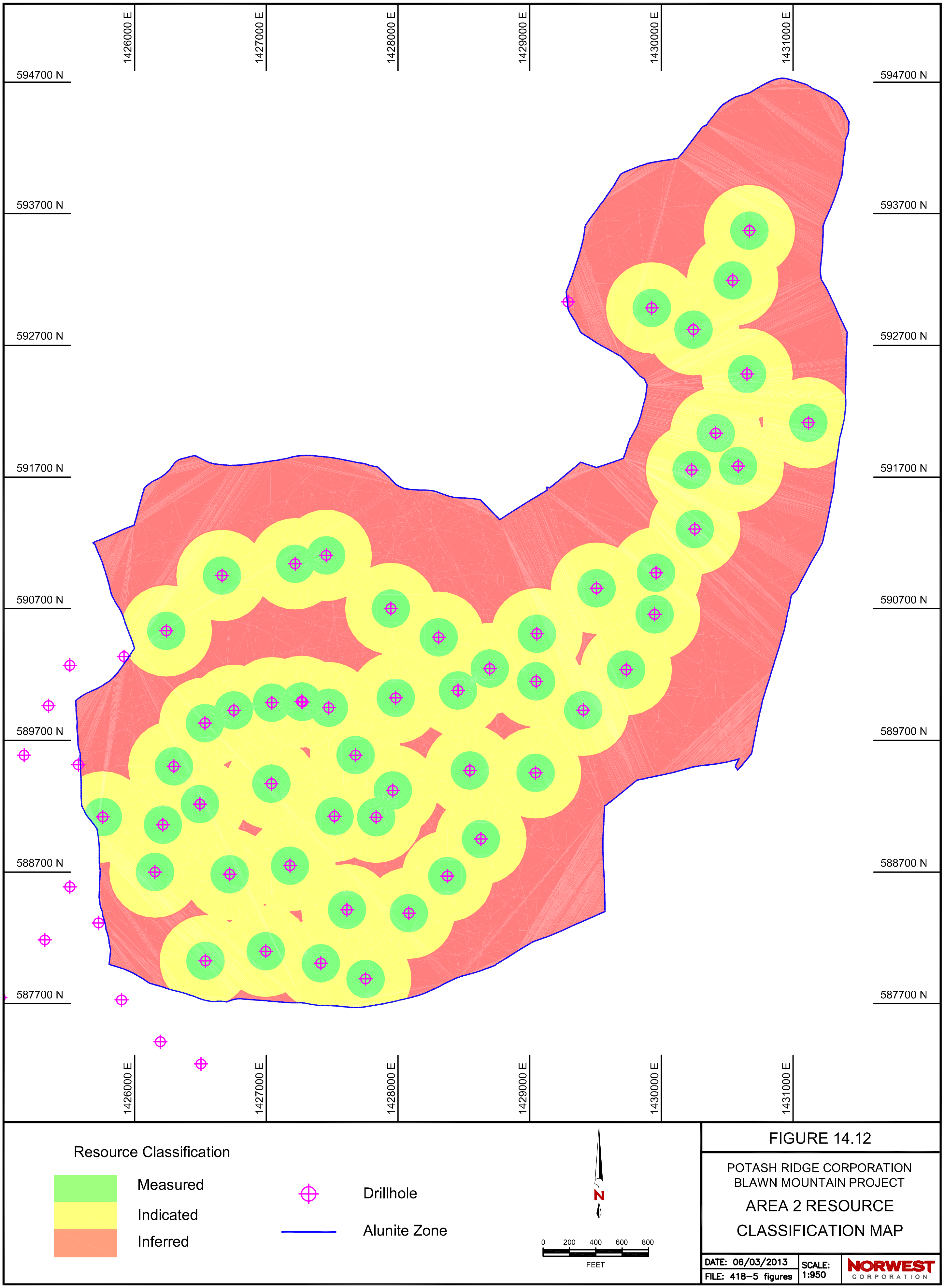


FIGURE 14.12

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

AREA 2 RESOURCE CLASSIFICATION MAP

DATE: 06/03/2013 | SCALE: 1:950
FILE: 418-5 figures

NORWEST CORPORATION

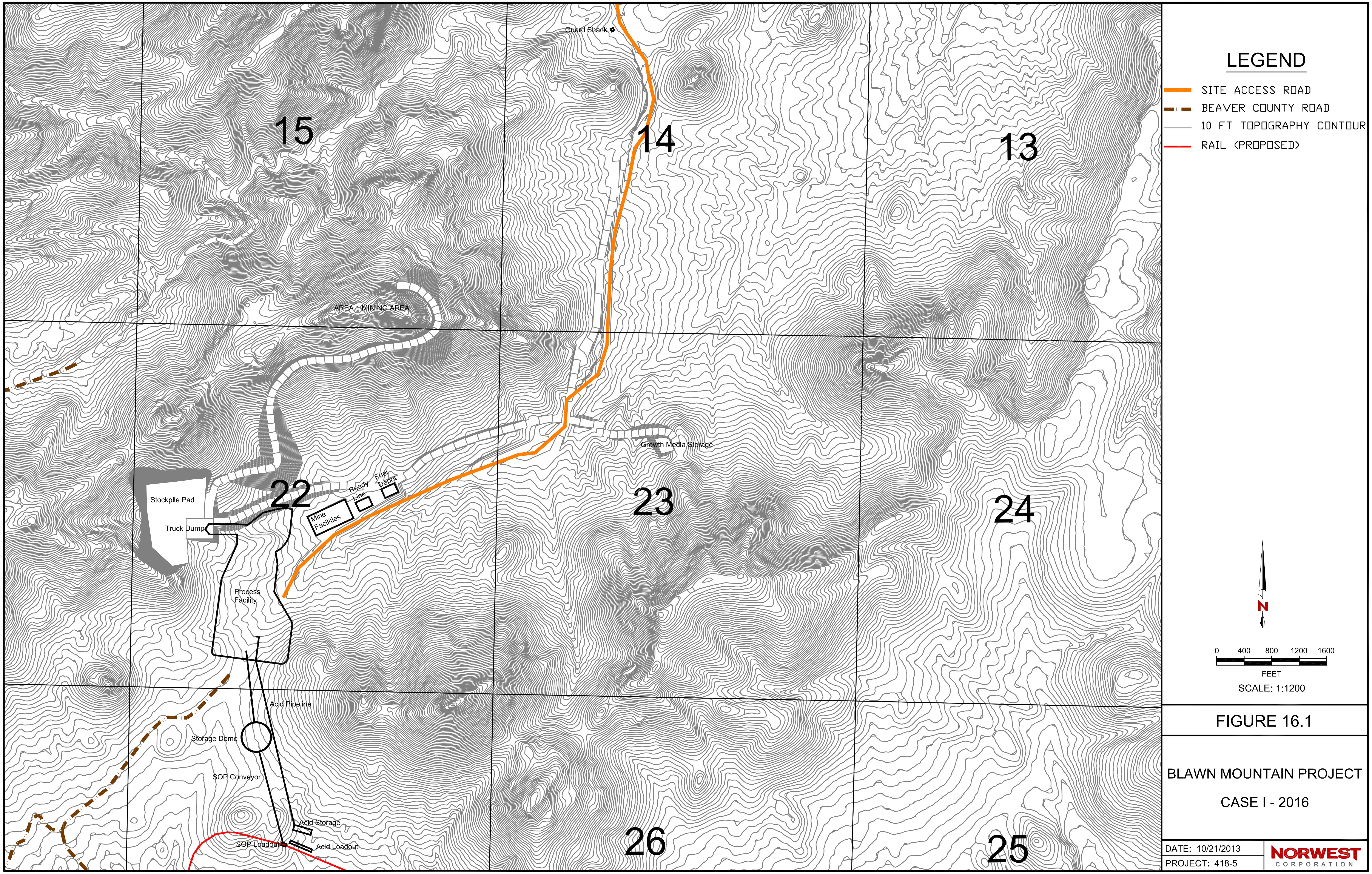


FIGURE 16.1

BLAWN MOUNTAIN PROJECT

CASE I - 2016

DATE: 10/21/2013

PROJECT: 418-5

NORWEST CORPORATION

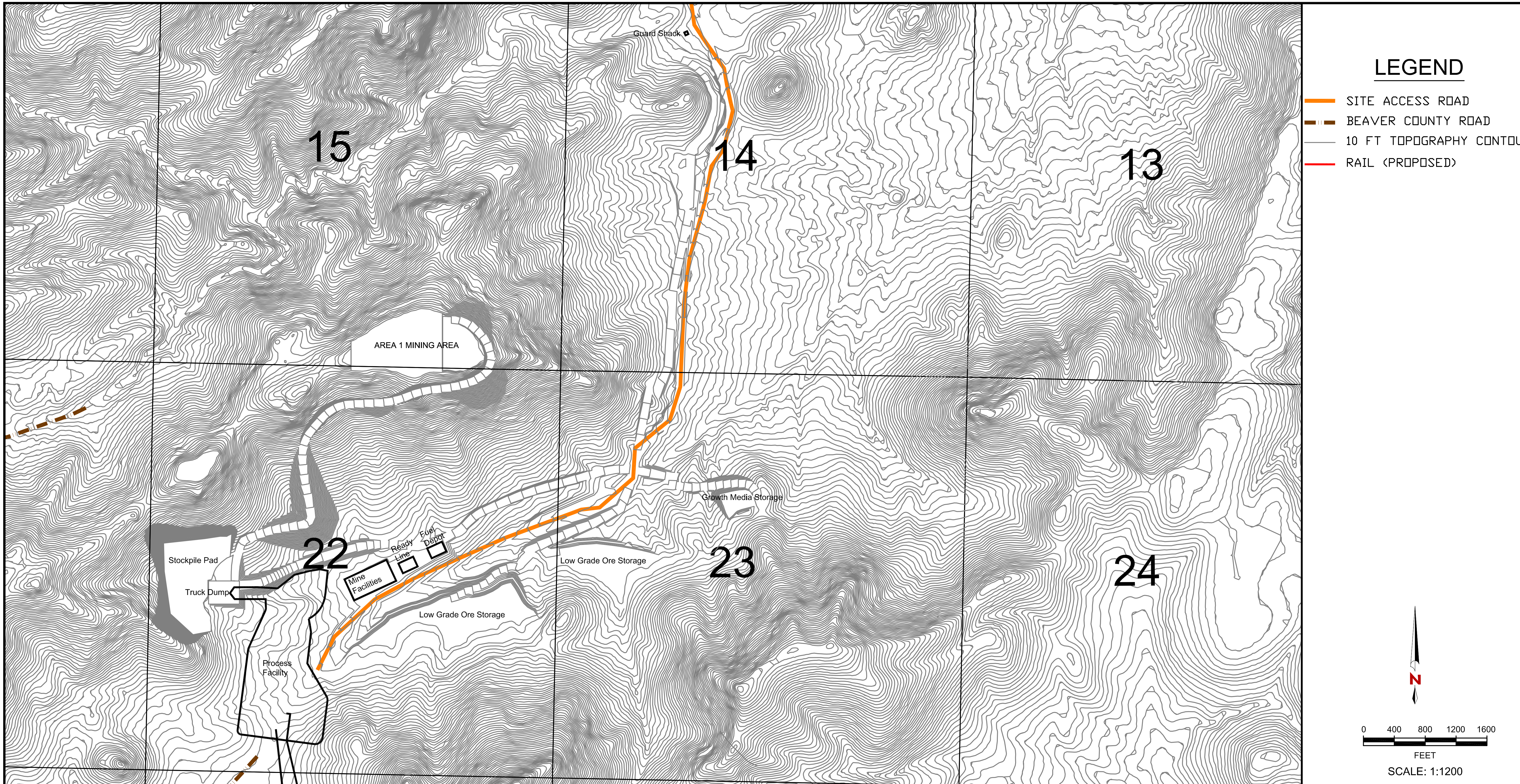


FIGURE 16.2

BLAWN MOUNTAIN PROJECT

CASE I -2017

DATE: 10/21/2013

PROJECT: 418-5

NORWEST CORPORATION

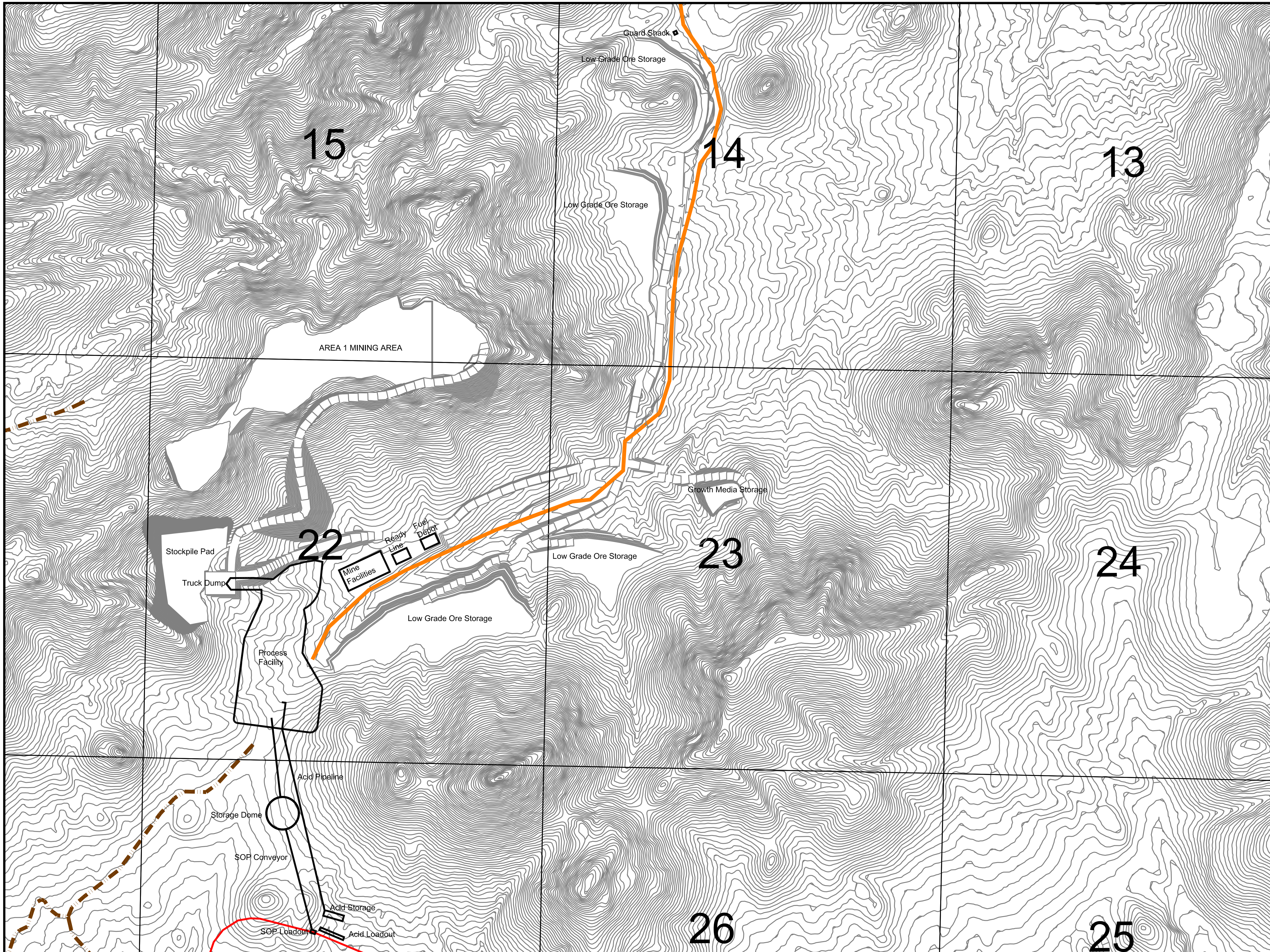


FIGURE 16.3

BLAWN MOUNTAIN PROJECT

CASE I - 2018

DATE: 10/21/2013

PROJECT: 418-5

NORWEST CORPORATION

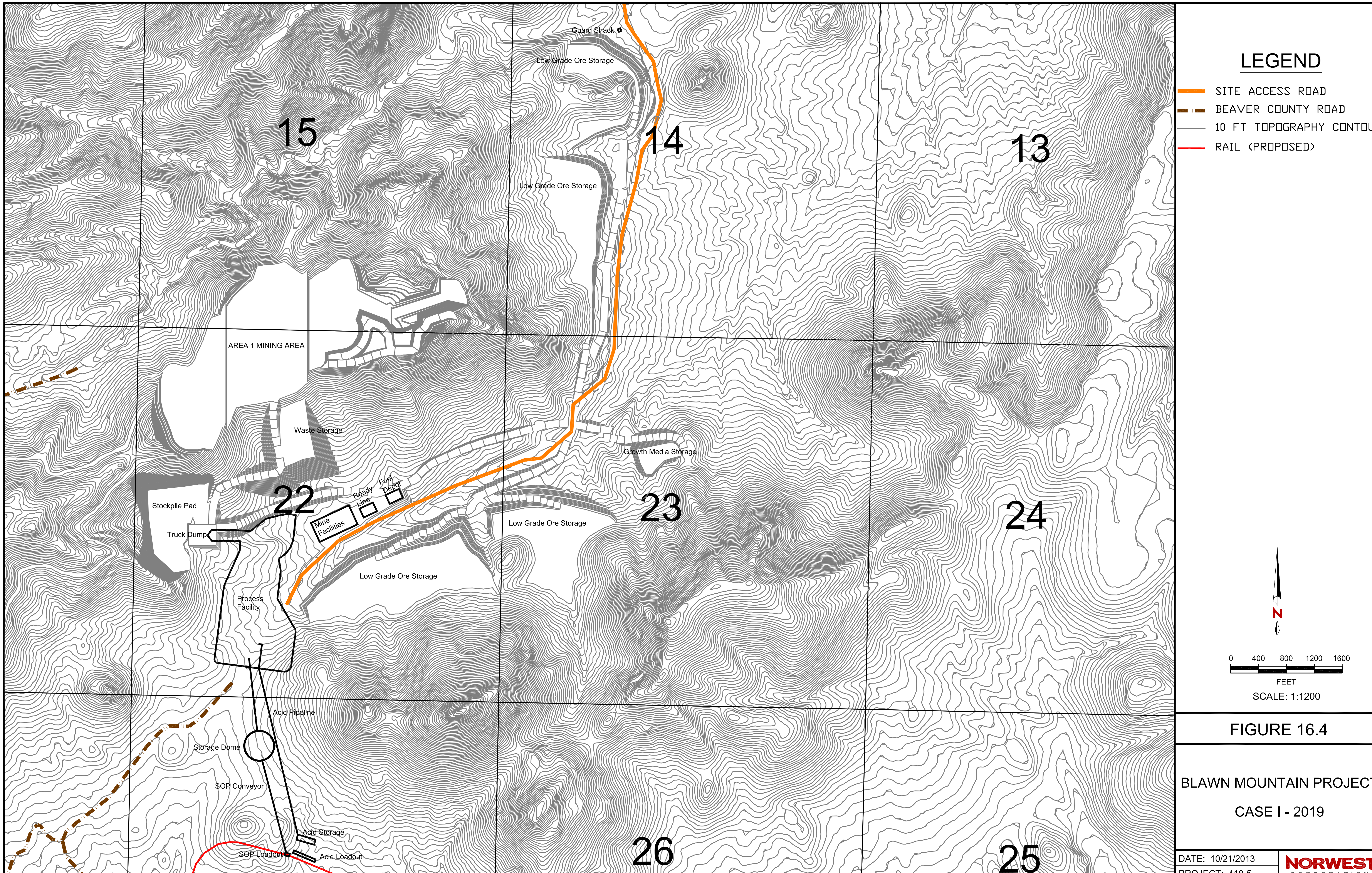


FIGURE 16.4

BLAWN MOUNTAIN PROJECT

CASE I - 2019

DATE: 10/21/2013

PROJECT: 418-5

NORWEST CORPORATION

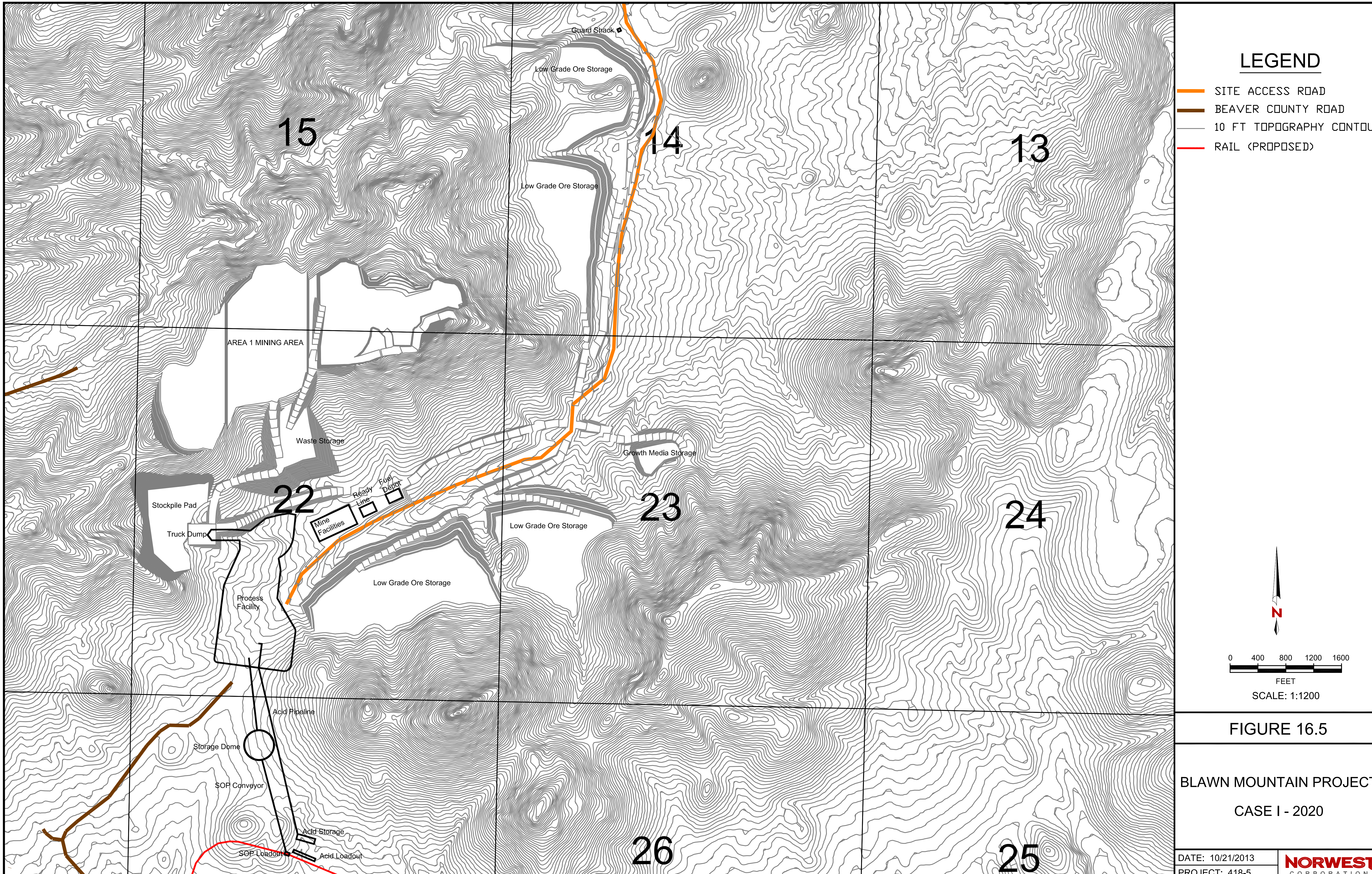
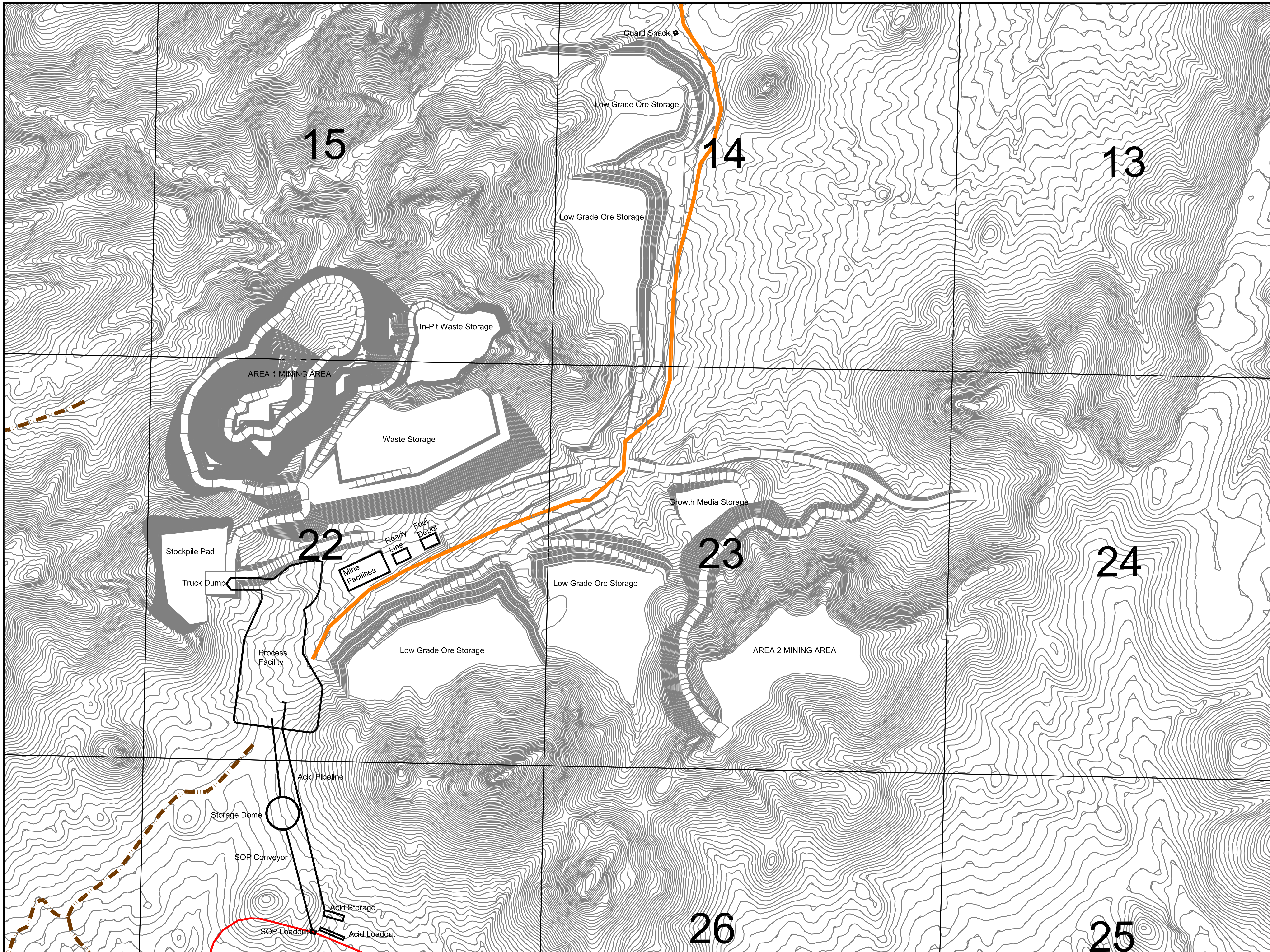


FIGURE 16.5

BLAWN MOUNTAIN PROJECT

CASE I - 2020



LEGEND

- SITE ACCESS ROAD
- BEAVER COUNTY ROAD
- 10 FT TOPOGRAPHY CONTOUR
- RAIL (PROPOSED)

N

0 400 800 1200 1600

FEET

SCALE: 1:1200

FIGURE 16.6

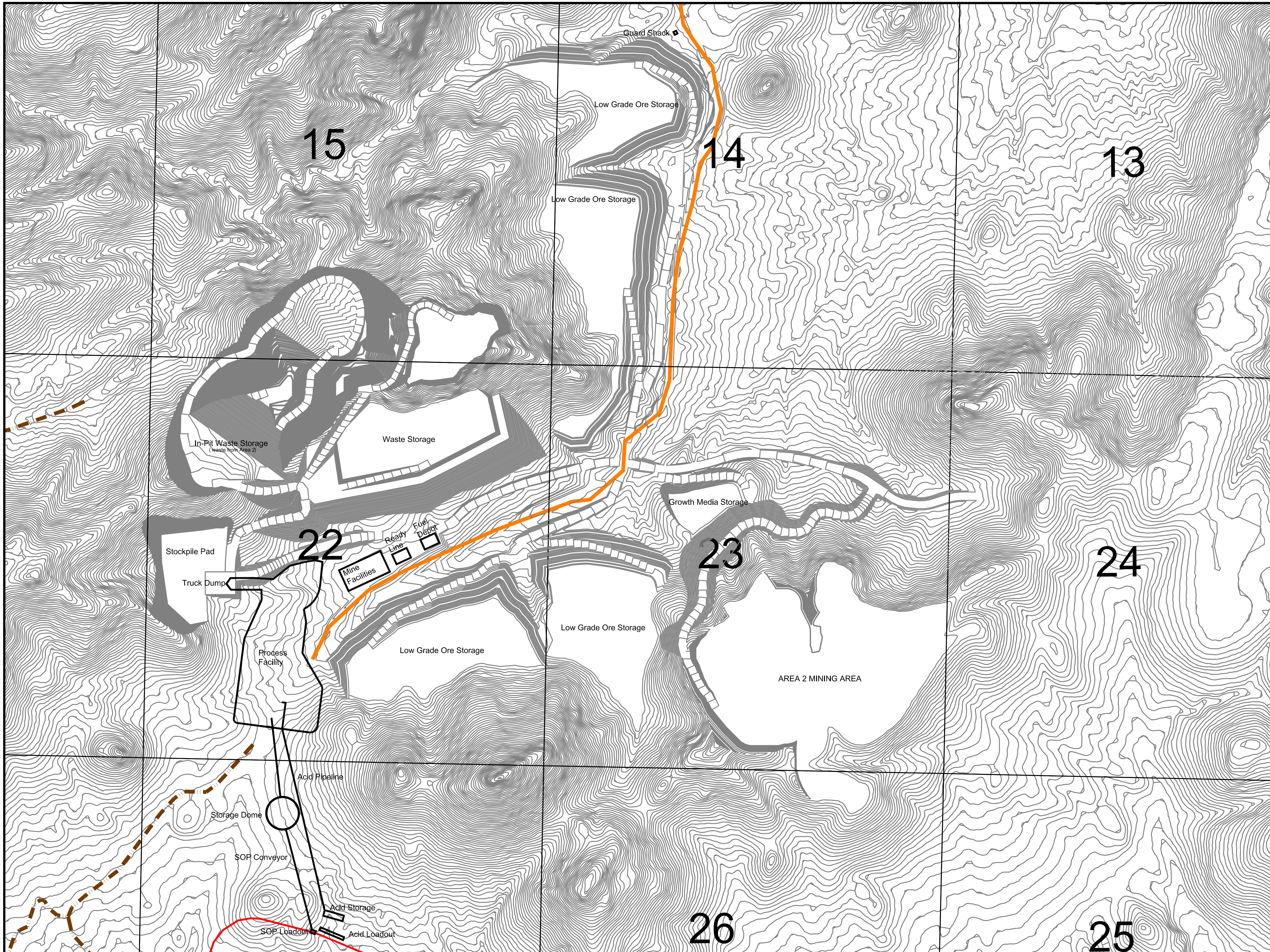
BLAWN MOUNTAIN PROJECT

CASE I - 2024

DATE: 10/21/2013

PROJECT: 418-5

NORWEST CORPORATION



LEGEND

SITE ACCESS ROAD
BEAVER COUNTY ROAD
10 FT TOPOGRAPHY CONTOUR
RAIL (PROPOSED)

N

0 400 800 1200 1600
FEET
SCALE: 1:1200

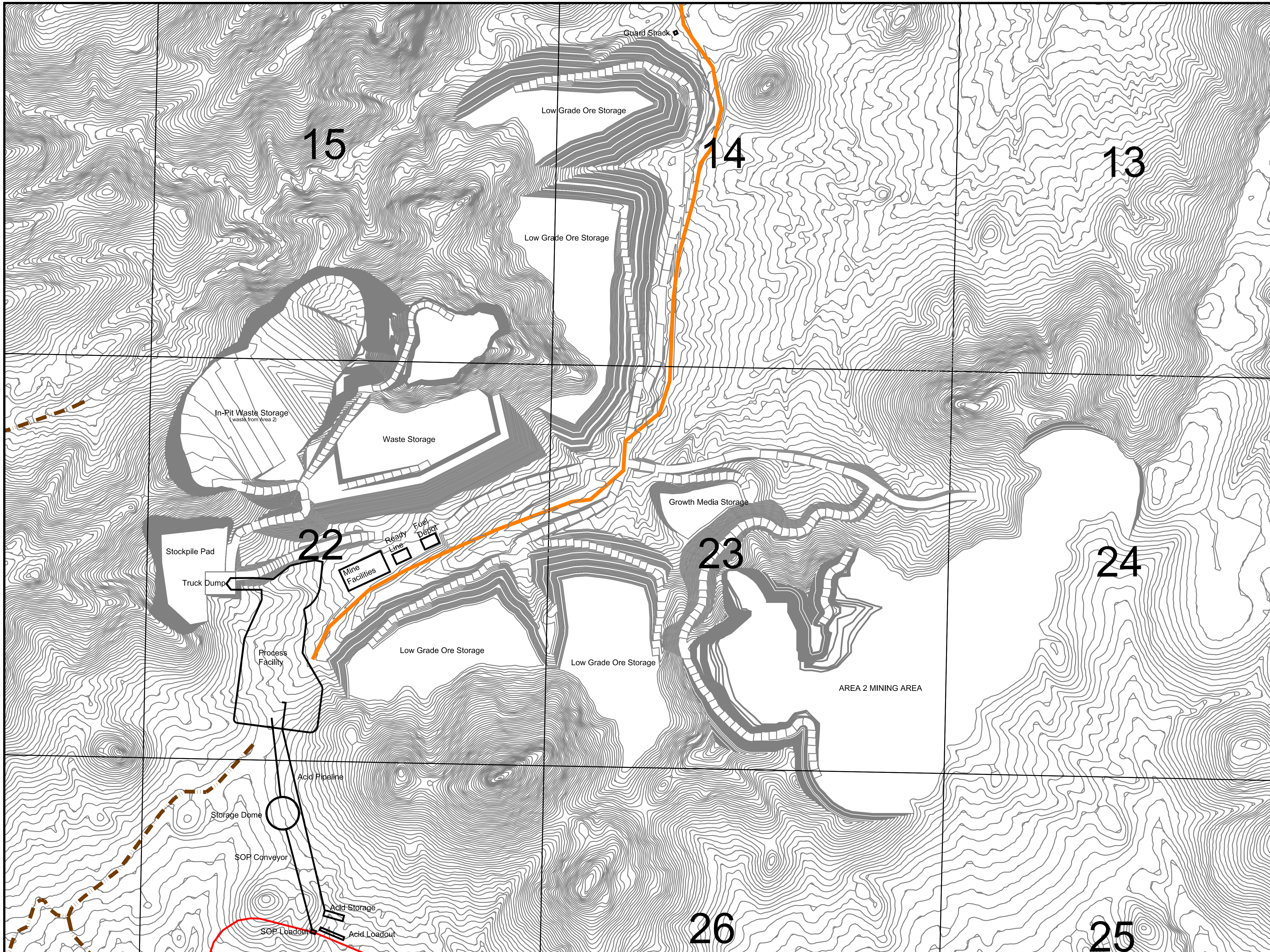
FIGURE 16.7

BLAWN MOUNTAIN PROJECT

CASE I - 2027

DATE: 10/21/2013
PROJECT: 418-5

NORWEST CORPORATION



LEGEND

SITE ACCESS ROAD
BEAVER COUNTY ROAD
10 FT TOPOGRAPHY CONTOUR
RAIL (PROPOSED)

FIGURE 16.8

BLAWN MOUNTAIN PROJECT

CASE I - 2032

DATE: 10/21/2013	NORWEST CORPORATION
PROJECT: 418-5	

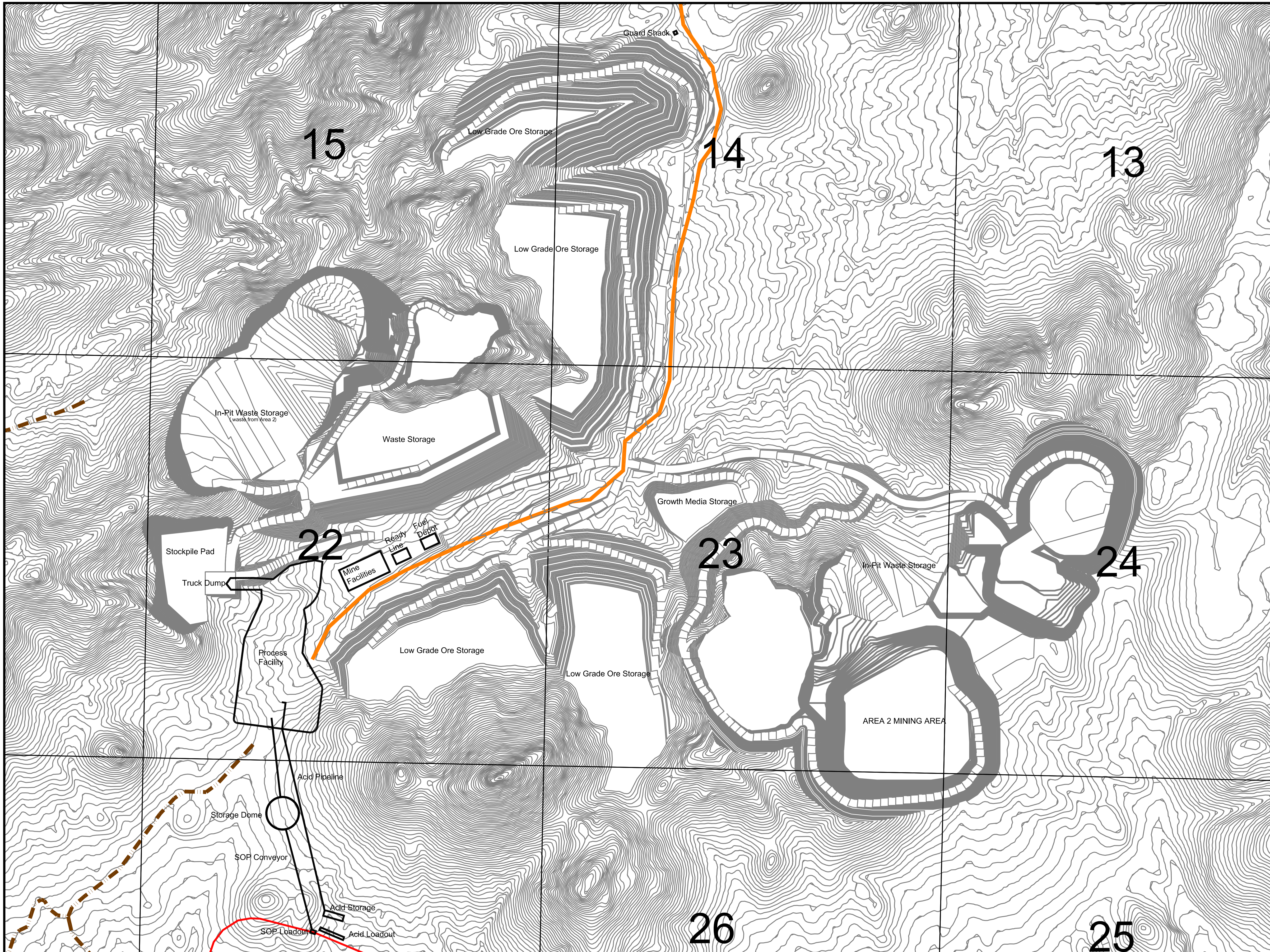


FIGURE 16.9

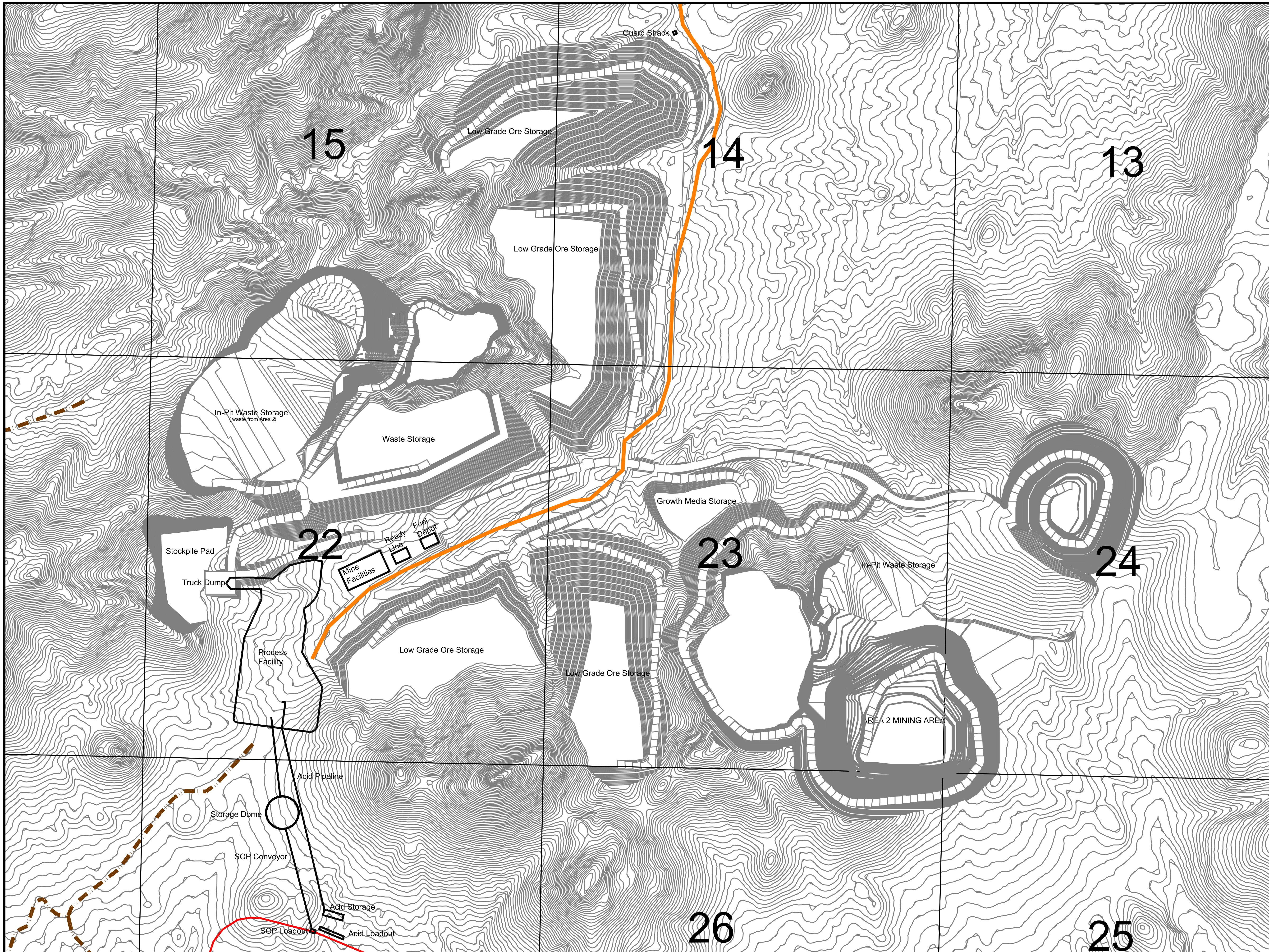
BLAWN MOUNTAIN PROJECT

CASE I - 2037

DATE: 10/21/2013

PROJECT: 418-5

NORWEST CORPORATION



LEGEND

- SITE ACCESS ROAD
- BEAVER COUNTY ROAD
- 10 FT TOPOGRAPHY CONTOUR
- RAIL (PROPOSED)

N

0 400 800 1200 1600
FEET
SCALE: 1:1200

FIGURE 16.10

BLAWN MOUNTAIN PROJECT

CASE I - 2041

DATE: 10/21/2013
PROJECT: 418-5

NORWEST CORPORATION

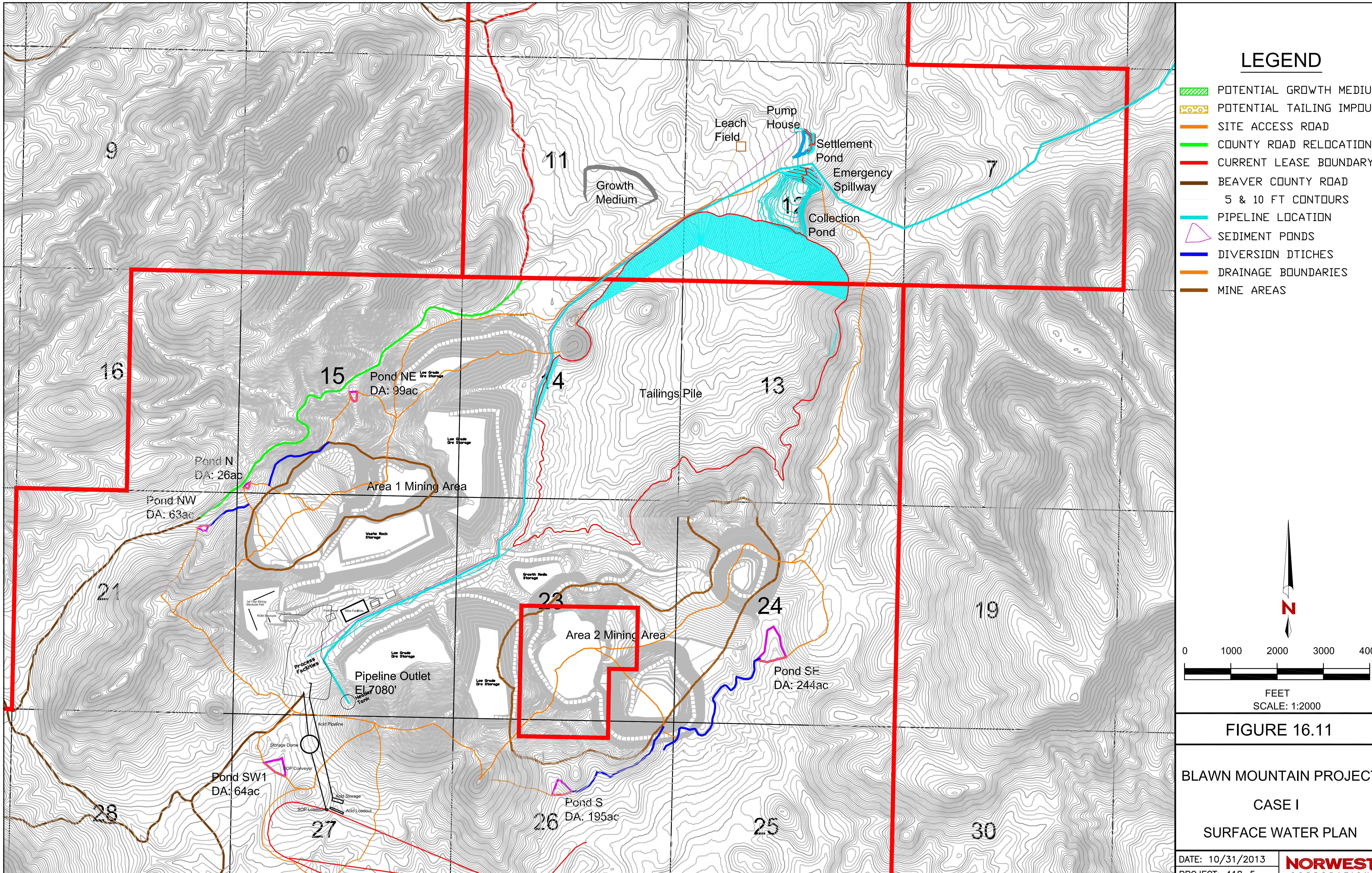


FIGURE 16.11

BLAWN MOUNTAIN PROJECT

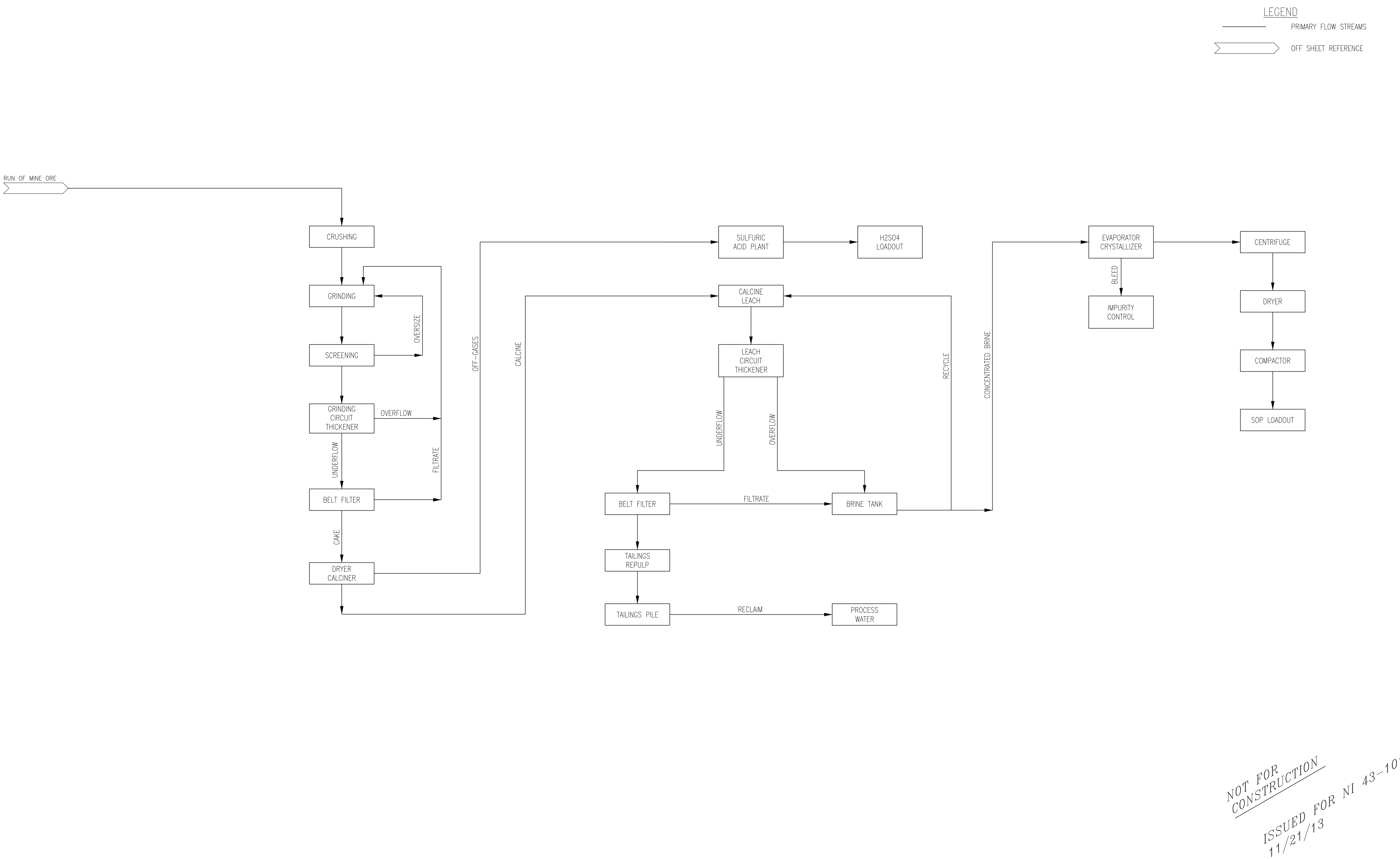
CASE I

SURFACE WATER PLAN

DATE: 10/31/2013

PROJECT: 418-5

NORWEST CORPORATION



FILE NAME: L:\2012 PROJECTS\504-001 PRC BLAWN MOUNTAIN\200 ENG AND DESIGN\201 DRAWINGS\201.09 BLOCK FLOW DIAGRAM\BFD-001.DWG PLOT DATE: 2013/11/21 LAST SAVED BY: ILYMAN

No.	DATE	BY	APP	REVISION
0	11/21/13	IL	BN	ISSUED FOR NI 43-101

REFERENCE DRAWING | NUMBER

POTASH RIDGE

CONFIDENTIAL

Intermountain Consumer Professional Engineers, Inc.
CONSULTING ENGINEERS
1145 E. SOUTH UNION AVE.
MIDVALE, UTAH 84047
BUS. (801) 255-1111 FAX. (801) 566-0088

		Date
Des By:	LRN	10/30/12
Drawn By:	BES	10/30/12
Chk'd. By:	LRN	10/30/12
ENGR.	LRN	10/30/12
APPD.	BN	10/30/12
APPD.		

POTASH RIDGE CORP
BLAWN MOUNTAIN
FIGURE 17.1 – BLOCK FLOW DIAGRAM

SCALE: NONE
Project No. 504-001
DRAWING No. BFD-001
REVISION 0

ISSUED FOR NI 43-101
11/21/13

NOTE:
SEE DWG. GA-101 FOR BUILDING LABELS AND MORE INFORMATION.

No.	DATE	BY	APP	REVISION	REFERENCE DRAWING	NUMBER
0	11/21/13	MSB	BD	ISSUED FOR NI 43-101	---	---

POTASH RIDGE

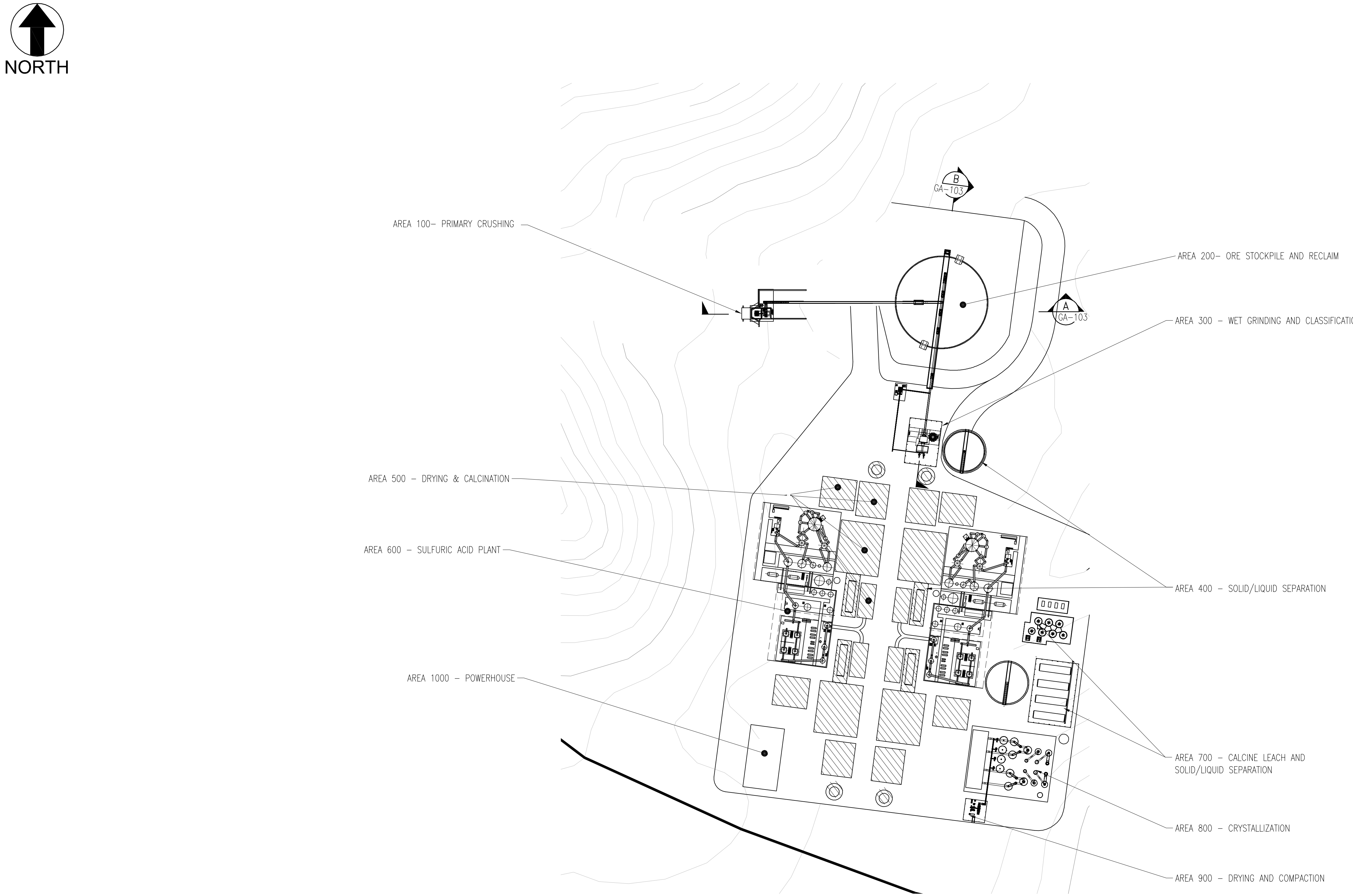
CONFIDENTIAL

Intermountain Consumer Professional Engineers, Inc.
CONSULTING ENGINEERS
1145 E. SOUTH UNION AVE.
MIDVALE, UTAH 84047
BUS. (801) 255-1111 FAX. (801) 566-0088

Des By: M BARTEL	Date 3/26/13	
Drawn By: M BARTEL	Date 3/26/13	
Chk'd. By:	Date	
ENGR.	Date	
APPD. BD	Date 10/31/13	SCALE 1"=500'
APPD.	Date	Project No. 504-001

POTASH RIDGE CORP
BLAWN MOUNTAIN
FIGURE 17.2 – GENERAL ARRANGEMENT
OVERALL SITE PLAN

DRAWING No. GA-001
REVISION 0



ISSUED FOR NI 43-101
11/21/13

PROCESS AREA PLAN
1"=200'

No.	DATE	BY	APP	REVISION	REFERENCE DRAWING	NUMBER
0	11/21/13	MSB	BD	ISSUED FOR NI 43-101	---	---

POTASH RIDGE

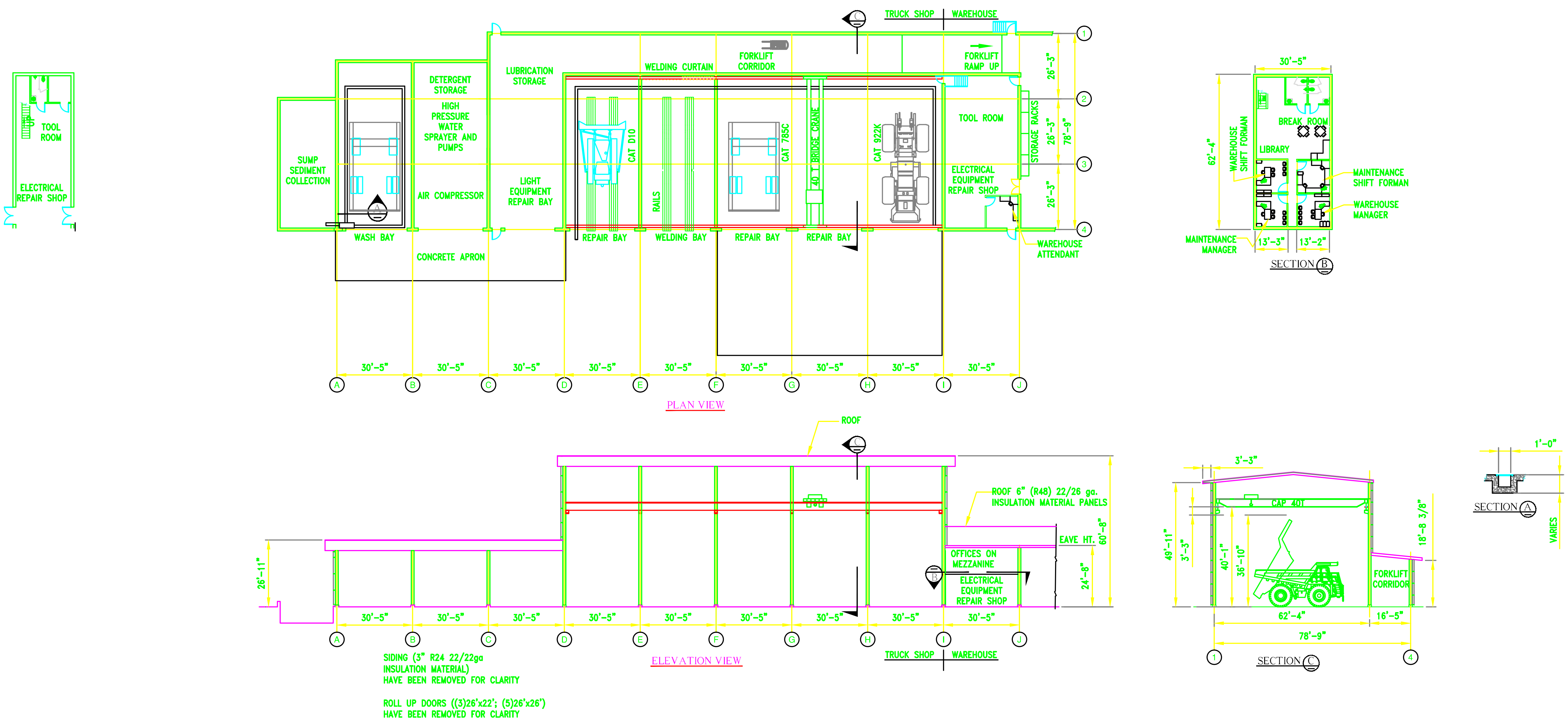
CONFIDENTIAL

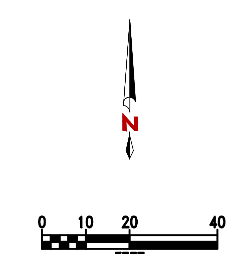
Intermountain Consumer Professional Engineers, Inc.
CONSULTING ENGINEERS
1145 E. SOUTH UNION AVE.
MIDVALE, UTAH 84047
BUS. (801) 255-1111 FAX. (801) 566-0088

Des By:	BD	Date	3/26/13
Drawn By:	IL	Date	3/26/13
Chk'd. By:		Date	
ENGR.		Date	
APPD.	BD	Date	10/31/13
APPD.		Date	

POTASH RIDGE CORP
BLAWN MOUNTAIN
FIGURE 17.3 - GENERAL ARRANGEMENT
PROCESS AREA PLAN

SCALE	DRAWING No.	REVISION
1"=200'	GA-002	0
Project No. 504-001		





SCALE: AS NOTED

FIGURE 18.1

BLAWN MOUNTAIN PROJECT

MINE TRUCK SHOP PLAN & ELEVATION VIEWS

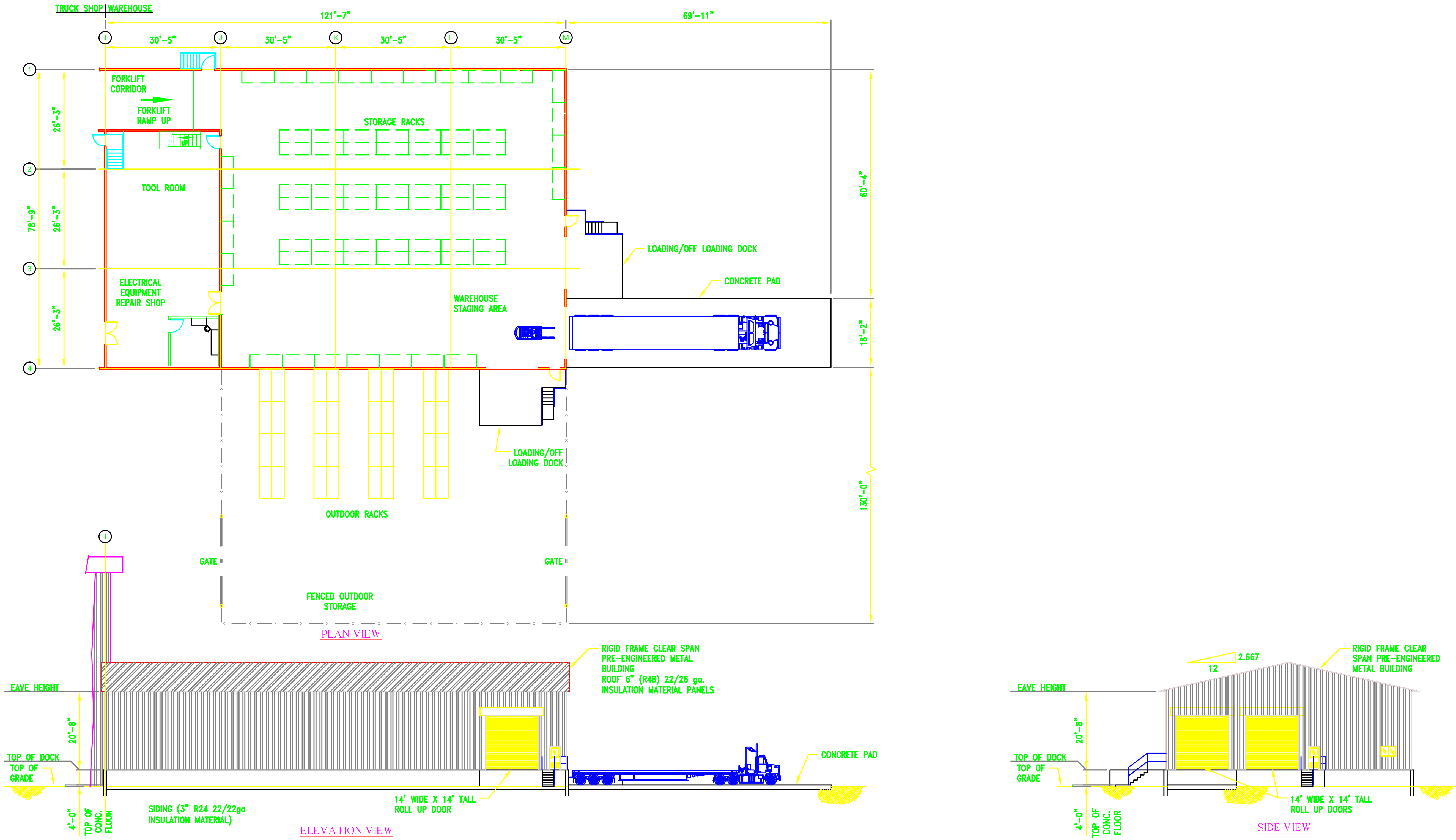
ISSUED FOR REPORT

NOT FOR CONSTRUCTION

DATE: 06/28/2013

PROJECT: 418-5

NORWEST CORPORATION



N

0 10 20 40
FEET

SCALE: AS NOTED

FIGURE 18.2

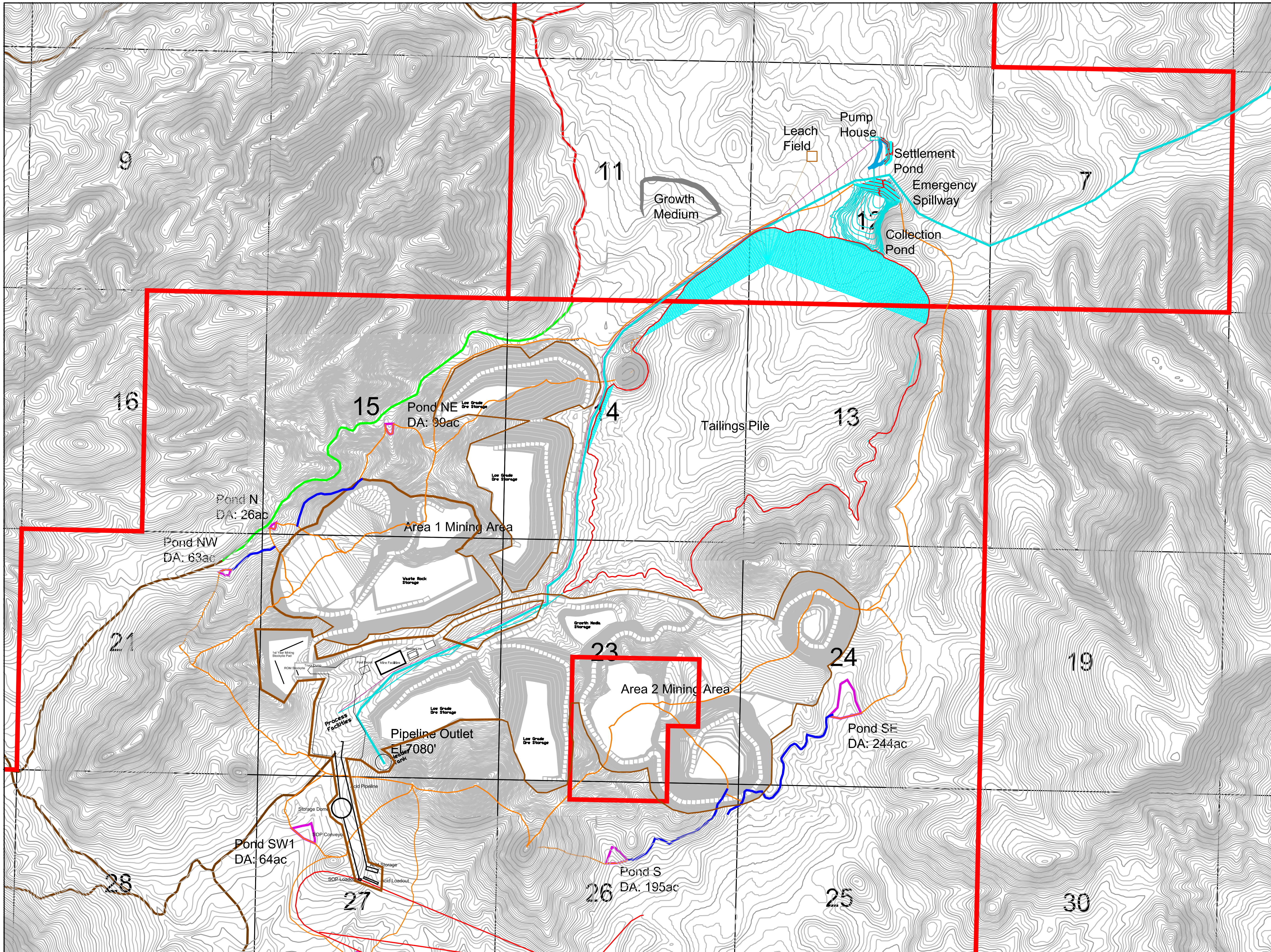
BLAWN MOUNTAIN PROJECT

MINE WAREHOUSE
PLAN & ELEVATION VIEWS

ISSUED FOR REPORT
NOT FOR CONSTRUCTION

DATE: 06/28/2013
PROJECT: 418-5

NORWEST CORPORATION



LEGEND

POTENTIAL GROWTH MEDIUM
POTENTIAL TAILING IMPOUNI
SITE ACCESS ROAD
COUNTY ROAD RELOCATION
CURRENT LEASE BOUNDARY
BEAVER COUNTY ROAD
5 & 10 FT CONTOURS
PIPELINE LOCATION
SEDIMENT PONDS
DIVERSION DTICHES
DRAINAGE BOUNDARIES
MINE AREAS

N

SCALE: 1:2000

FIGURE 18.3

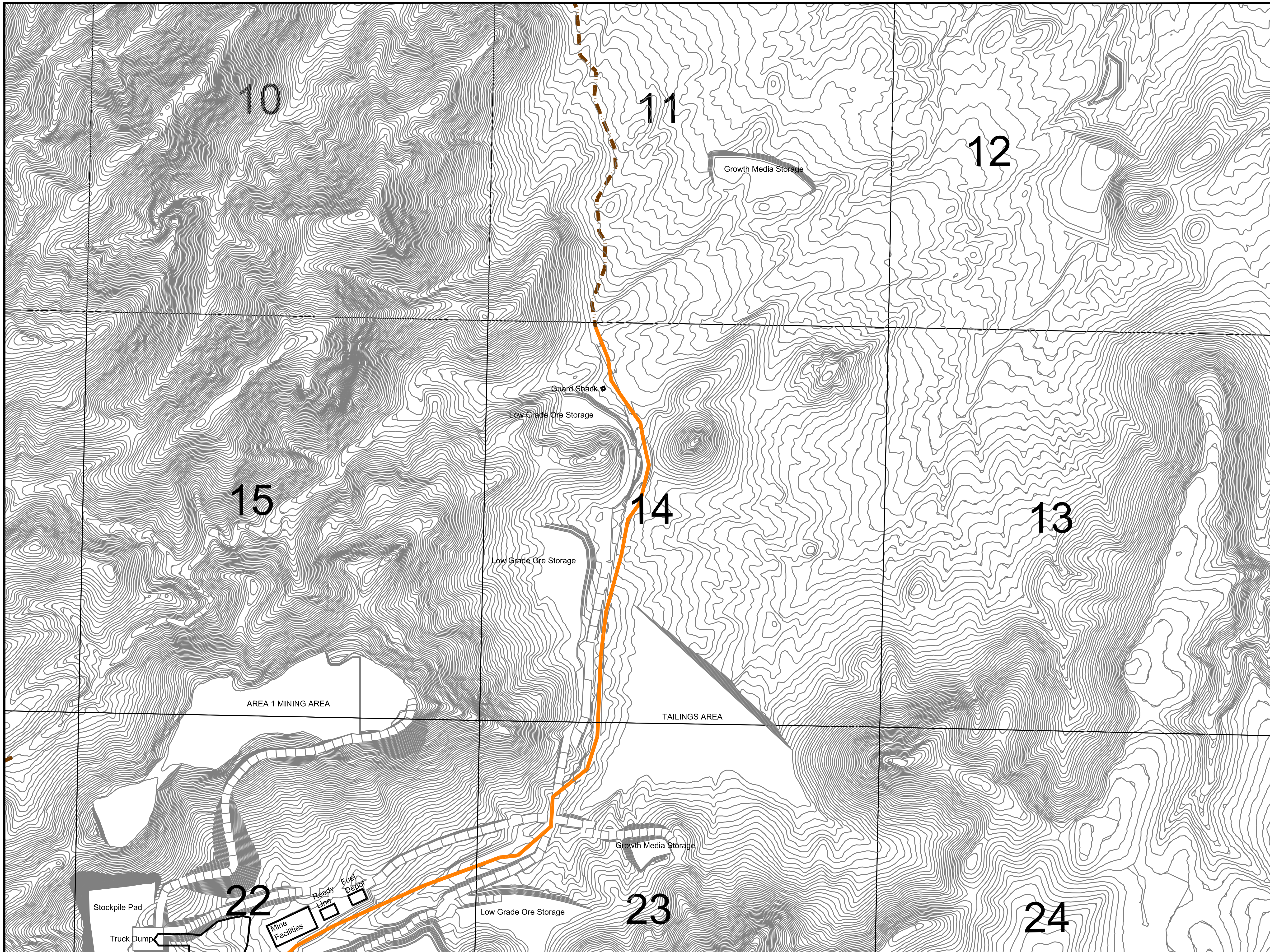
BLAWN MOUNTAIN PROJECT

CASE I

SURFACE WATER PLAN

DATE: 10/30/2013
PROJECT: 418-5

NORWEST CORPORATION



LEGEND

- SITE ACCESS ROAD
- BEAVER COUNTY ROAD
- 10 FT TOPOGRAPHY CONTOUR

N

0 400 800 1200 1600

FEET

SCALE: 1:1200

FIGURE 18.4

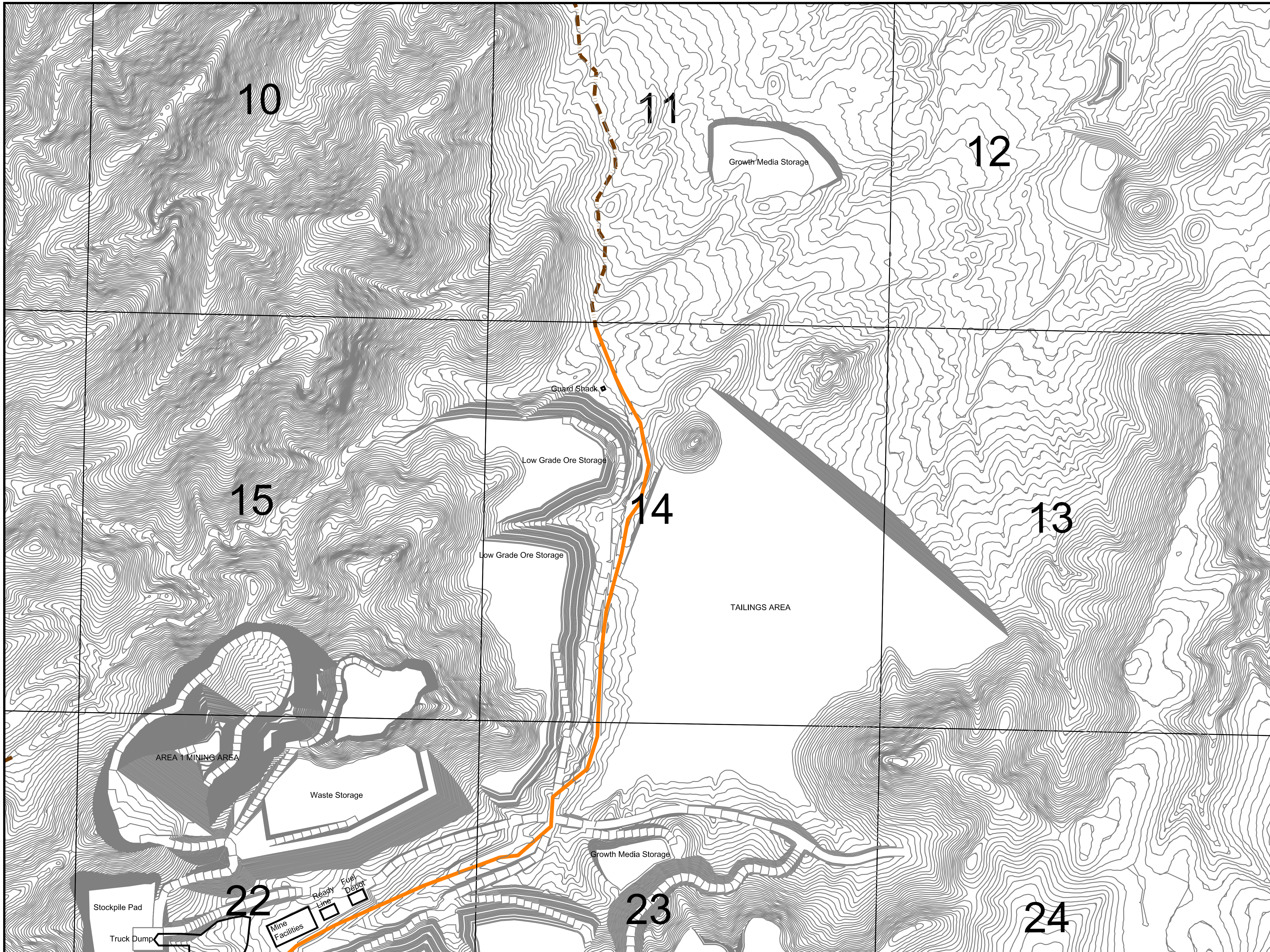
BLAWN MOUNTAIN PROJECT

CASE I - 2018 TAILINGS

DATE: 10/21/2013

PROJECT: 418-5

NORWEST CORPORATION



LEGEND

SITE ACCESS ROAD
BEAVER COUNTY ROAD
10 FT TOPOGRAPHY CONTOUR

N

0 400 800 1200 1600
FEET
SCALE: 1:1200

FIGURE 18.5

BLAWN MOUNTAIN PROJECT

CASE I - 2027 TAILINGS

DATE: 10/21/2013
PROJECT: 418-5

NORWEST CORPORATION

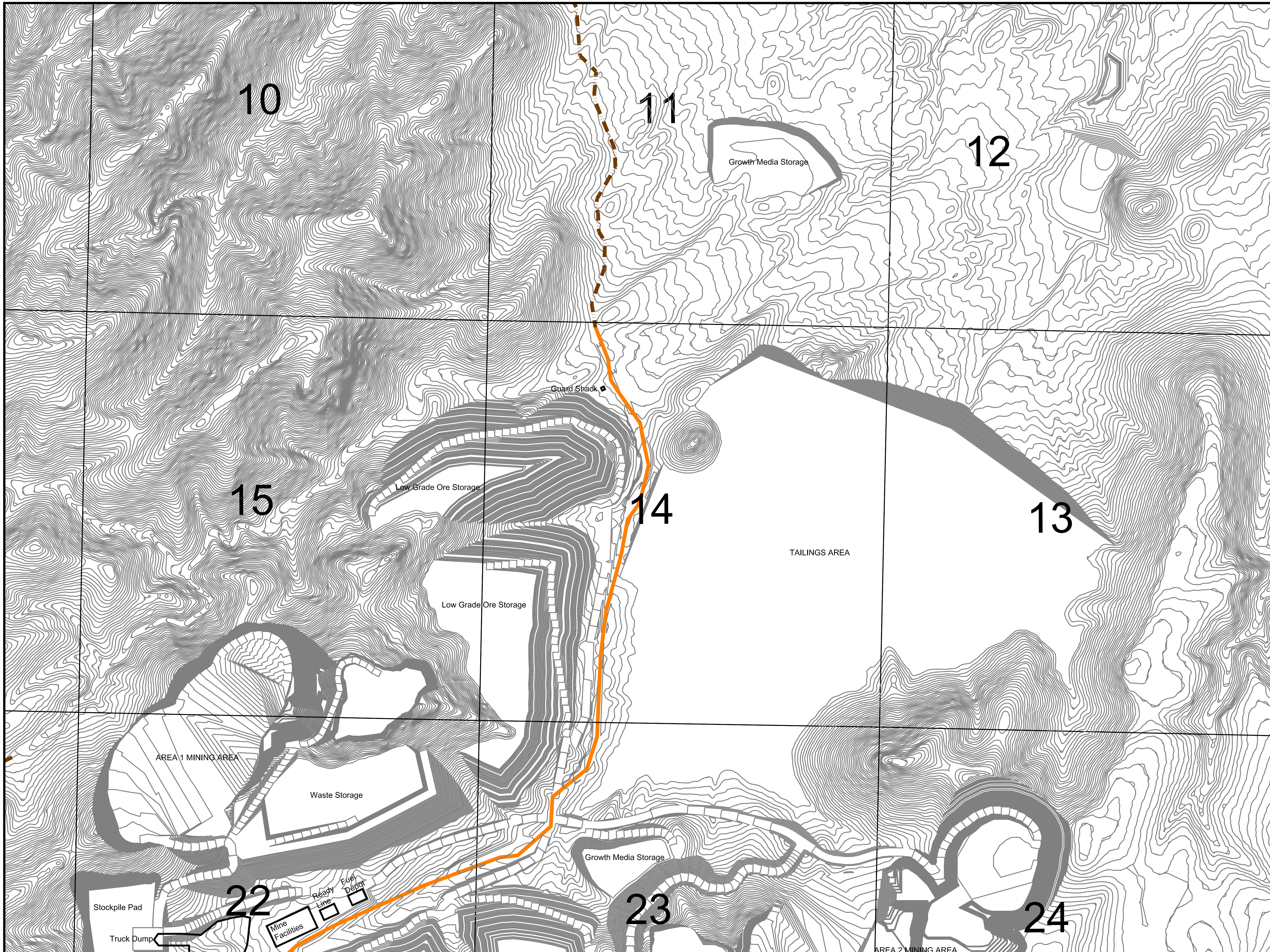


FIGURE 18.6

BLAWN MOUNTAIN PROJECT

CASE I - 2037 TAILINGS

DATE: 10/21/2013

PROJECT: 418-5

NORWEST CORPORATION

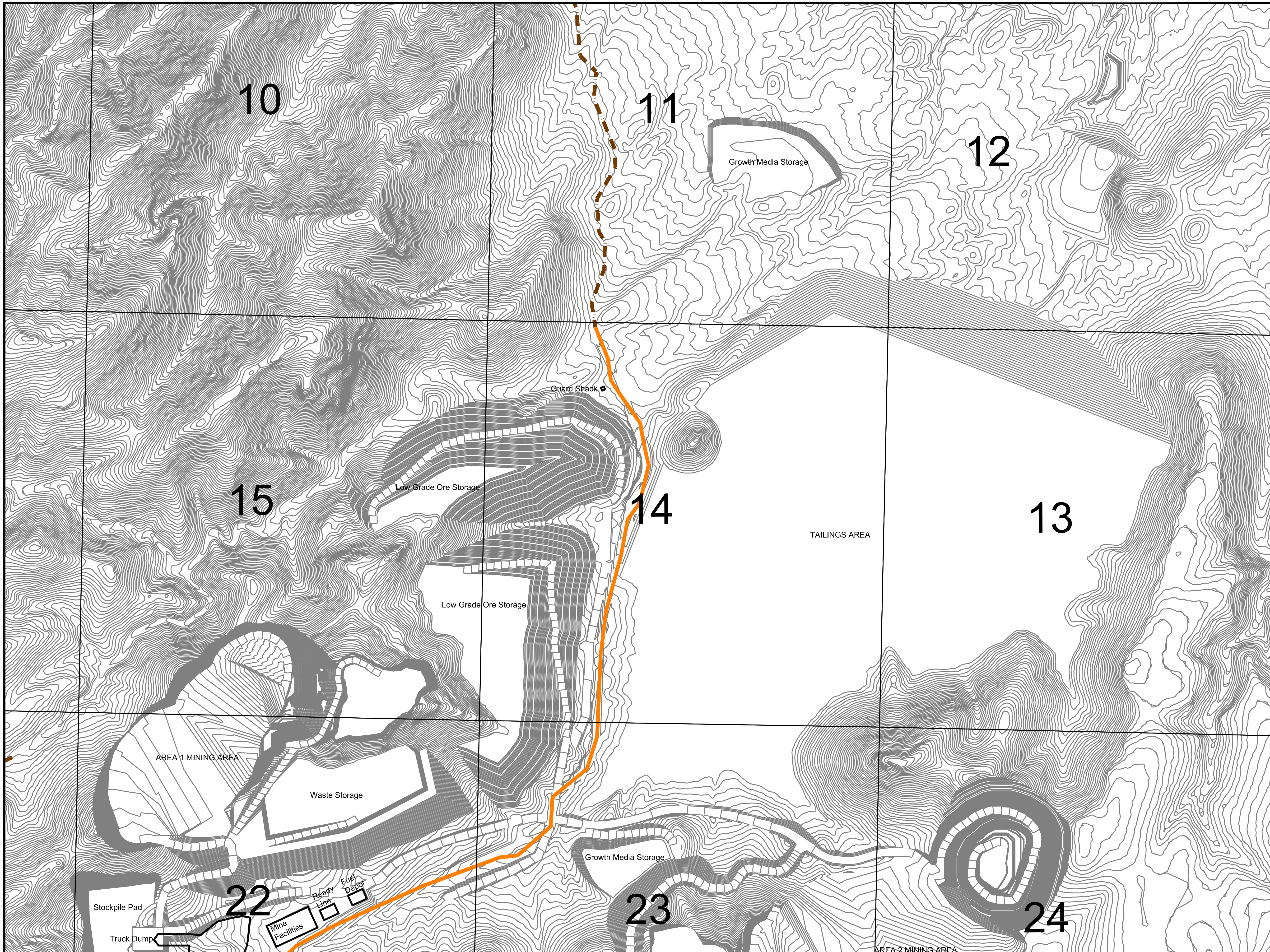


FIGURE 18.7

BLAWN MOUNTAIN PROJECT

CASE I - 2047 TAILINGS

DATE: 10/21/2013

PROJECT: 418-5

NORWEST CORPORATION

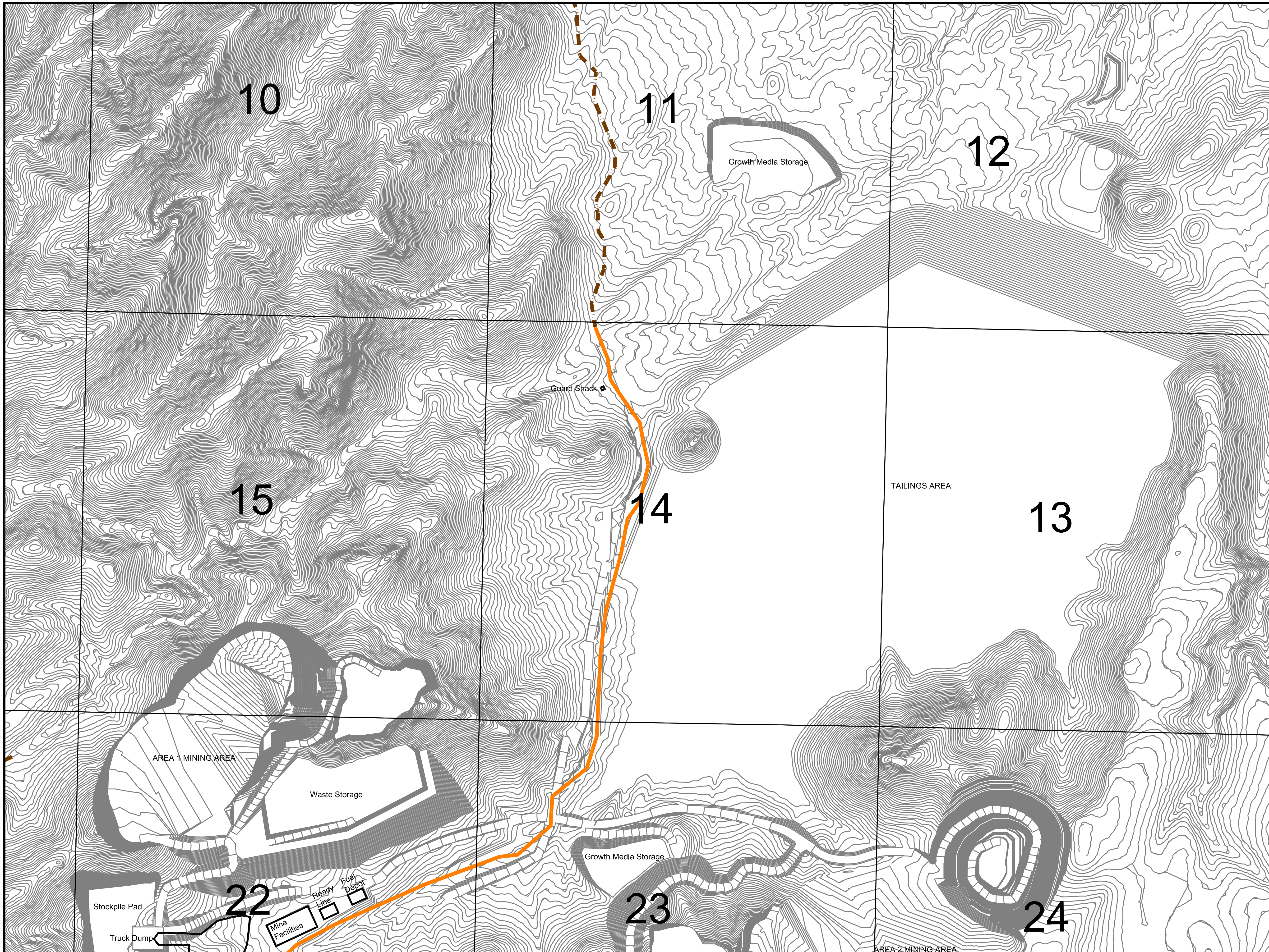


FIGURE 18.8

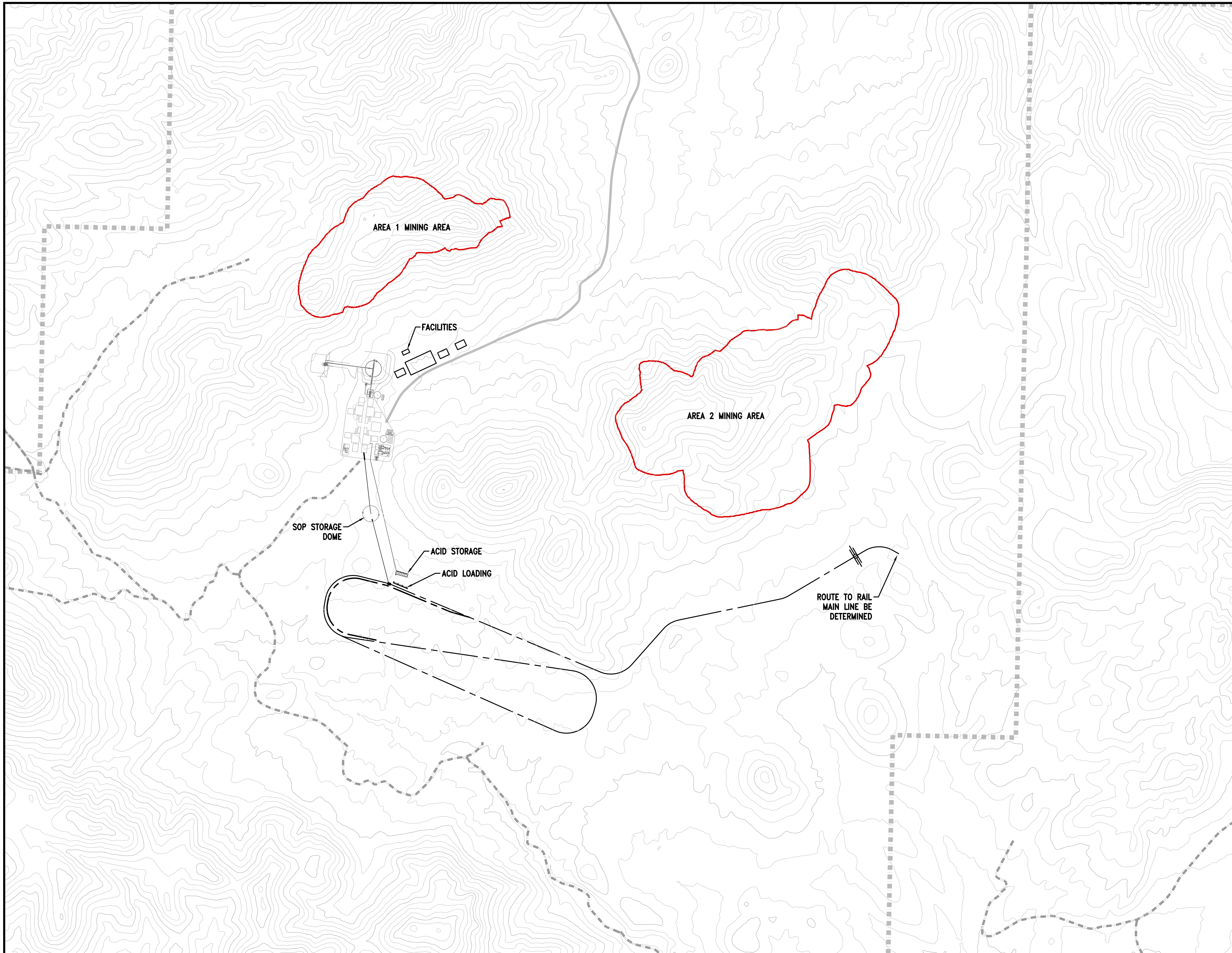
BLAWN MOUNTAIN PROJECT

CASE I - 2057 TAILINGS

DATE: 10/21/2013

PROJECT: 418-5

NORWEST CORPORATION



LEGEND

PROPOSED RAIL LINE
EXISTING ROAD
PROPOSED ROAD

Notes:
Topo shown represents the "Existing Ground Elevation" at 50' intervals. All coordinates are in Utah State Planes; NAD83 datum, Central Zone, US Foot. Elevations given are in "feet above mean sea level".

N

NOT FOR CONSTRUCTION

SCALE: 1" = 1,000'

FIGURE 18.9

BLAWN MOUNTAIN PROJECT
RAIL LINE SITE PLAN

DATE: 10/04/2013
FILE: Railroad Design Option
SCALE: 1:4100
NORWEST CORPORATION

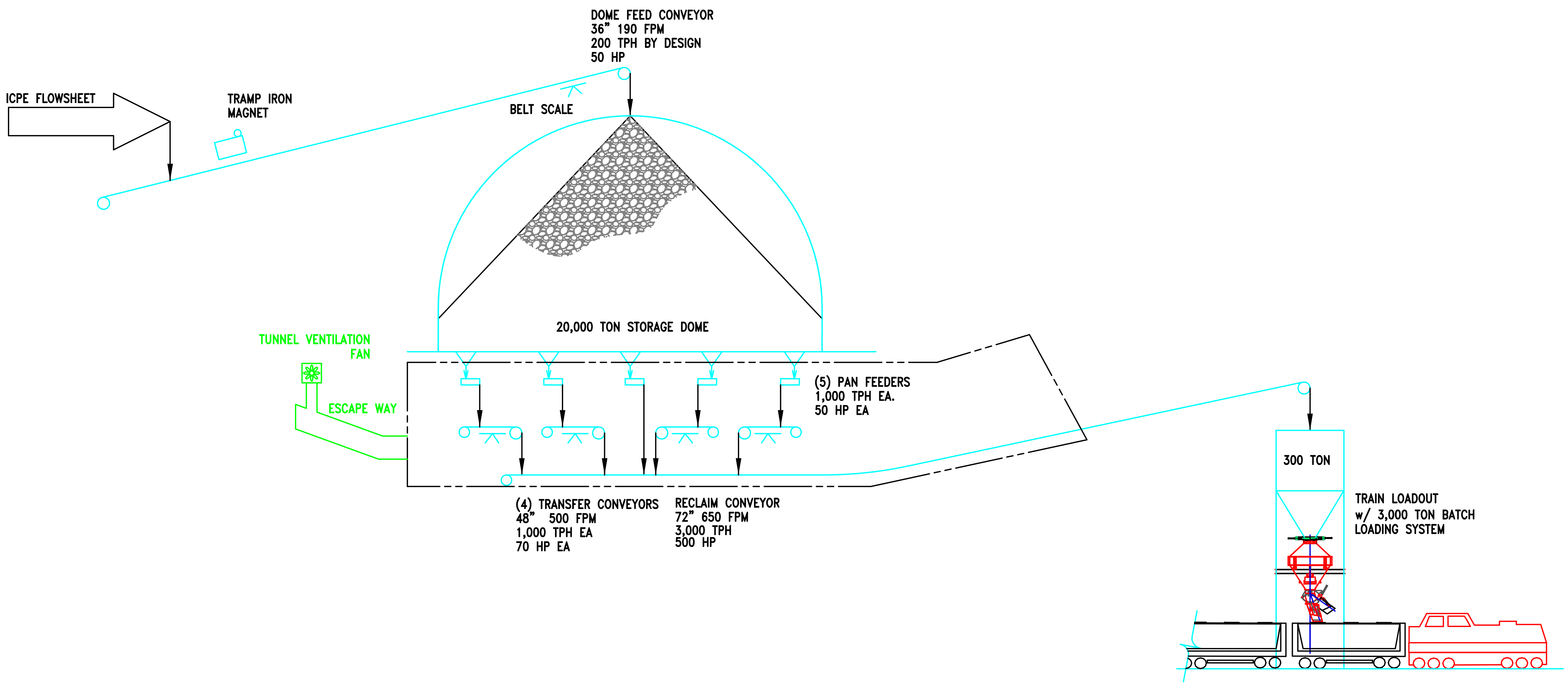


FIGURE 18.10

BLAWN MOUNTAIN PROJECT

SOP FLOWSHEET

Reclaim Conveyor, Storage Dome and Rail Loadout

DATE: 10/18/2013	SCALE:	NORWEST CORPORATION
FILE: 418-5-FAC\MH	NTS	

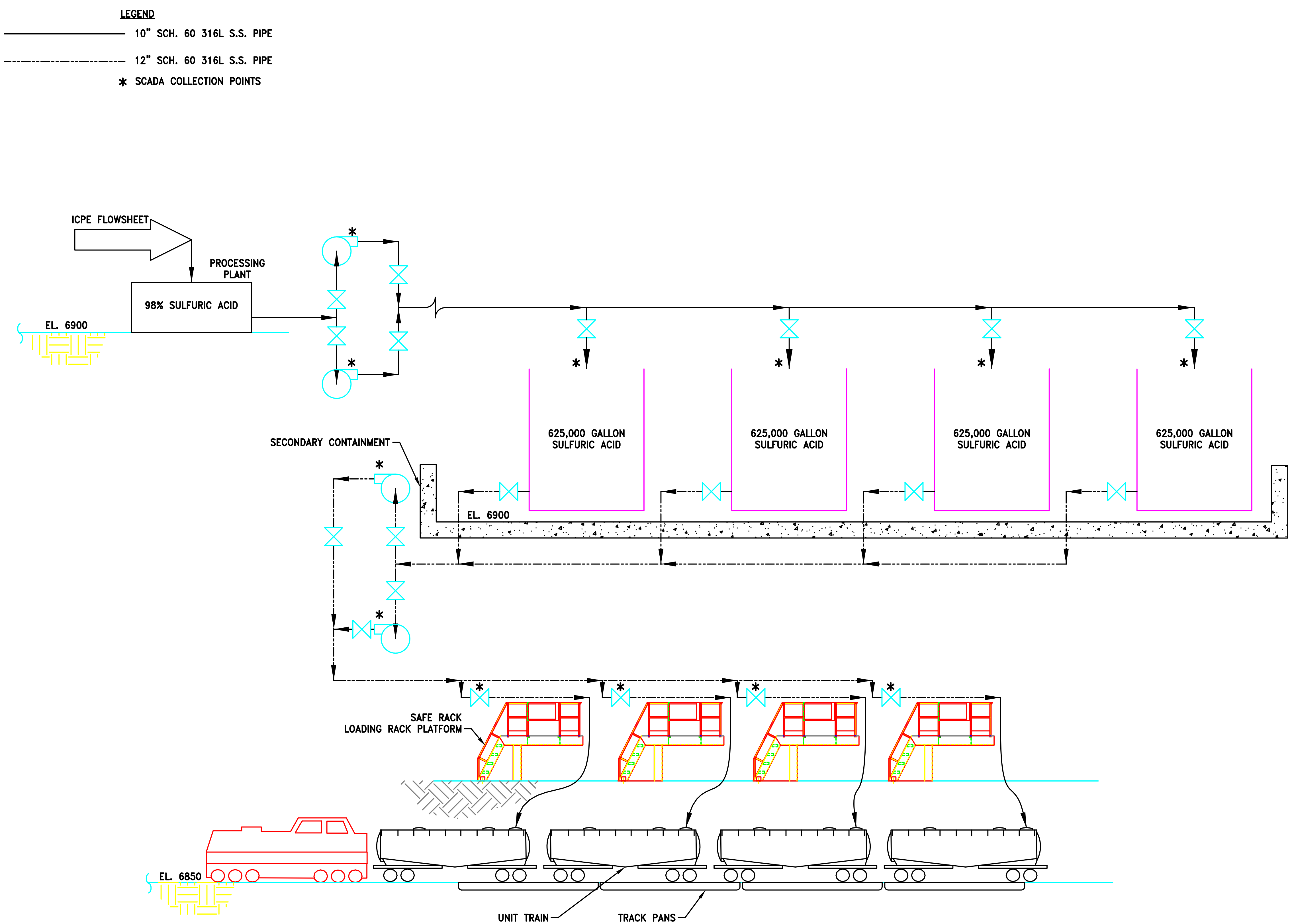


FIGURE 18.11

BLAWN MOUNTAIN PROJECT
ACID FLOWSHEET
Acid Pipeline, Storage and Rail Loadout

DATE: 10/18/2013	SCALE:	NORWEST CORPORATION
FILE: 418-5-FAC\MH	NTS	

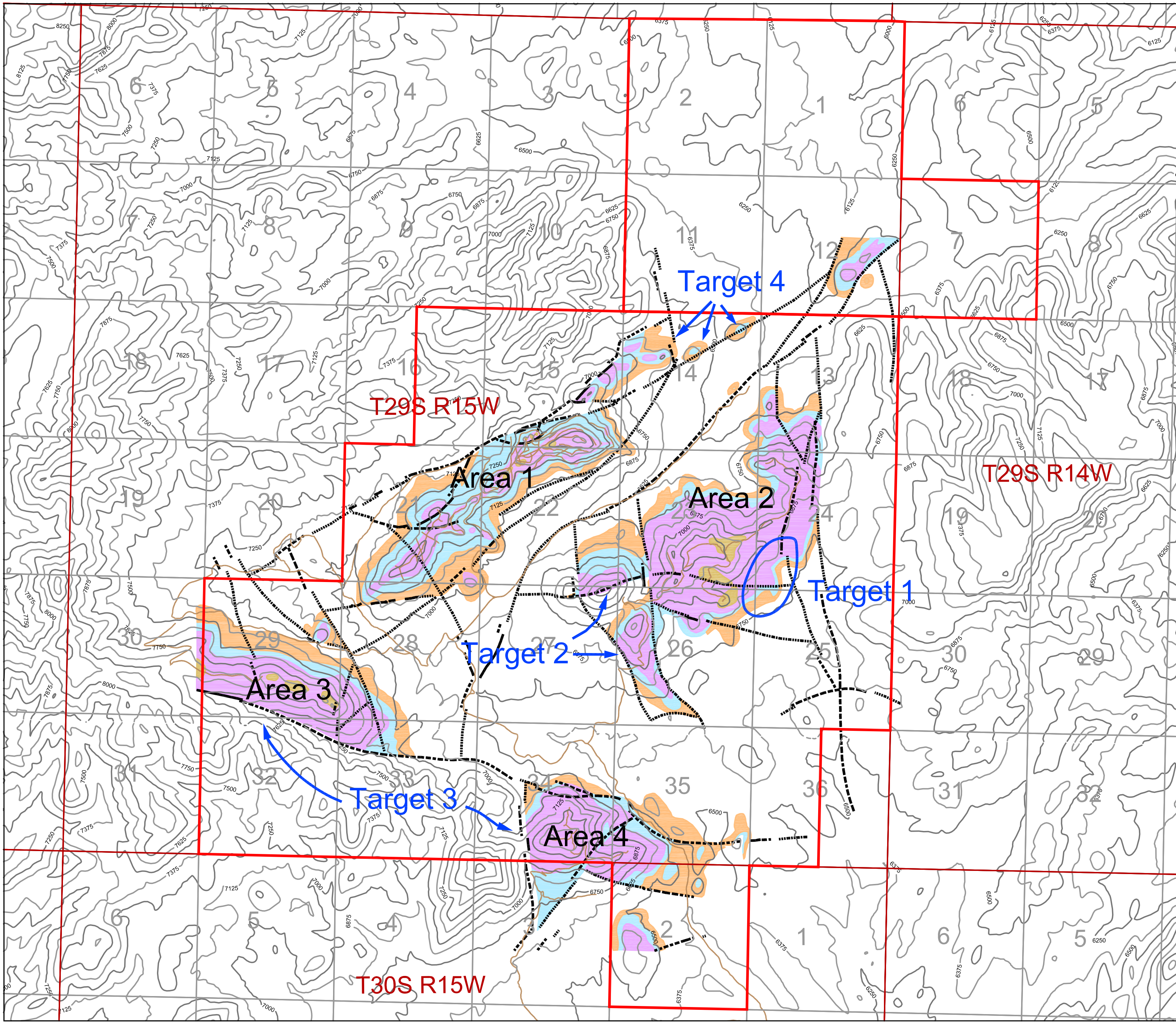


FIGURE 26.1

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

EXPLORATION TARGETS

DATE: 06/25/2013
FILE: 418-5 fig 7.4

NORWEST CORPORATION