Technical Report
Preliminary Economic Assessment
Blawn Mountain Project
Beaver Country, Utah

Report Date:
November 5, 2012
Report Effective Date:
October 30, 2012

ENERGY, MINING, AND ENVIRONMENTAL CONSULTANTS

NORWEST
CORPORATION

TECHNICAL REPORT PRELIMINARY ECONOMIC ASSESSMENT

BLAWN MOUNTAIN PROJECT
BEAVER COUNTY, UTAH

Submitted to:
POTASH RIDGE CORPORATION

Report Date:
November 5, 2012

Report Effective Date:
October 30, 2012

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 B. KERR, CPG
LAWRENCE D. HENCHEL, P.GEO.
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-seven 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 Preliminary Economic Assessment, Blawn Mountain Project, Beaver County, Utah” dated November 5, 2012 (Technical Report) relating to the Blawn Mountain Alunite Property, with an **Effective Date of October 30, 2012**
7. I personally visited and inspected the Blawn Mountain Property on several occasions in 2012. The first visit to property occurred on February 9, 2012 and the most recent visit to the property was on October 30, 2012.
8. Prior to being retained by Potash Ridge Corporation in connection with the preparation of the Technical Report, I have not had prior involvement with the property that is the subject of the Technical Report.
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 parts of the Technical Report for which I am responsible 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 5th Day of November, 2012.

ORIGINAL SIGNED AND SEALED BY “STEVEN B. KERR”

Steven B. Kerr, CPG, PG

CERTIFICATE OF QUALIFICATIONS

I, Lawrence D. Henchel, PG, 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 Geologist in the State of Utah, #6087593-2250 and I am a Registered Member of The Society for Mining, Metallurgy and Exploration, Inc.
4. I have worked as a geologist for a total of twenty-nine 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.
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 Preliminary Economic Assessment, Blawn Mountain Project, Beaver County, Utah" dated November 5, 2012 (Technical Report) relating to the Blawn Mountain Alunite Property, **with an Effective Date of October 30, 2012**.
7. I personally inspected the Blawn Mountain Alunite Property on October 30, 2012.
8. Prior to being retained by Potash Ridge Corporation in connection with the preparation of the Technical Report, I have not had prior involvement with the property that is the subject of the Technical Report.
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 parts of the Technical Report for which I am responsible 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 5th Day of November, 2012.

ORIGINAL SIGNED AND SEALED BY "LAWRENCE D. HENCHEL"

Lawrence D. Henchel, PG
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 fourteen years since my graduation from University.
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 Preliminary Economic Assessment, Blawn Mountain Project, Beaver County, Utah" dated November 5, 2012 (Technical Report) relating to the Blawn Mountain Alunite Property, with an effective date October 30, 2012.
7. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose of which makes the Technical Report misleading
8. I personally inspected the Blawn Mountain Property on March 15, 2012.
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 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 5th Day of November, 2012.

ORIGINAL SIGNED AND SEALED BY "JASON TODD"

Jason N. Todd,

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 am the Principal of Intermountain Consumer Professional Engineers, 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-five years since my graduation from a university, for companies and projects involved with the processing of metals, specialty and precious metals, 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 21of the Technical Report titled "Blawn Mountain Alunite Project, Beaver County, Utah" dated November 5, 2012 (the "Technical Report") relating to the Preliminary Economic Analysis of the Blawn Mountain Alunite property, with an Effective Date of October 30, 2012.

6. I personally inspected the Blawn Mountain Property on March 15, 2012.

7. Prior to being retained by Potash Ridge Corporation in connection with the preparation of the Technical Report, I have not had involvement with the property that is the subject of the Technical Report.

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 scientific and technical information that is required to be disclosed to make the parts of the Technical Report for which I am responsible 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 5th Day of November, 2012.

ORIGINAL SIGNED AND SEALED BY "ROBERT I. NASH"

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 Technical Report titled “Blawn Mountain Alunite Project, Beaver County, Utah” dated November 02, 2012 (the “Technical Report”) relating to the Preliminary Economic Analysis of the Blawn Mountain Alunite property, with an Effective Date of October 30, 2012.
7. I personally inspected the Blawn Mountain Property on March 15, 2012.
8. Prior to being retained by Potash Ridge Corporation in connection with the preparation of the Technical Report, I have not had involvement with the property that is the subject of the Technical Report.
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 scientific and technical information that is required to be disclosed to make the parts of the Technical Report for which I am responsible 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 5th Day of November, 2012.

ORIGINAL SIGNED AND SEALED BY “L. RAVINDRA NATH”

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 private mineral exploration and development company with corporate offices in Toronto, Ontario, Canada. This technical report is a Preliminary Economic Assessment (PEA) and presents estimates of mineral resources from PRC's Blawn Mountain Project in southwestern Utah. This technical report has been prepared in accordance with National Instrument (NI) 43-101 and Form 43-101F1.

1.1 LOCATION AND TENURE

The Blawn Mountain Project consists of 11,549.2 acres of Utah State mineral tracts controlled through a Mining and Exploration With an Option To Lease Agreement (Exploration/Option Agreement) through the state of Utah School and Institutional Trust Lands Administration (SITLA). Alunite is a hydrated aluminium potassium sulfate, $KAl_3(SO_4)_2(OH)_6$ from which both sulfate of potash and aluminum have been extracted.

The property is located approximately 30 air miles southwest of the town of Milford, Utah and 30 air miles from the Nevada 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 Railroad (UP) route, running north-south and connecting Salt Lake City with Las Vegas and farther points on the UP rail system.

The Blawn Mountain Project is comprised of 17 full sections and 2 half sections of SITLA land and potash mineral rights as shown in Figure 4.2. 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 and the mineral lease was assigned by Utah Alunite, LLC to Utah Alunite Corporation. Two small mineral leases occur within the Blawn Mountain Project. One lease is a 40 acre tract located along the western edge of the project area and the second lease is a155 acre tract within the boundaries of the project area. Both mineral leases within the Blawn Mountain Project are for metallic minerals only and do not include potash mineral rights.

The Exploration/Option Agreement is for a 3 year term and required a front-end payment of \$200,000[1], annual payments of \$69,300 (\$6/acre) and a \$1,020,000 bonus for lease issuance. 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.

1.2 HISTORY

Mining operations have been conducted on alunite occurrences in southwest Utah since the early 1900's. Early extraction targeted both potash and aluminum. The Blawn Mountain property 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 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 economics and depressed pricing for alumina and potash in the 1980's.

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 Al_2O_3 content and therefore skew the K_20 estimates since potassium was not optimized. Historic estimates ranged from 142.6 million short tons (Mt) 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%.[2] None of these studies are deemed to be 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

---

[1] Currency in US Dollars
[2] Walker, 1972 and 1973; Chapman 1974; Couzens, 1975.

reporting of resources for Area 1 in accordance with CIM Standards on Mineral Resources and Reserves, effective date April 16, 2012 (Norwest, 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.

1.3 GEOLOGY

The Blawn Mountain Alunite deposit 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 property 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.

Blawn Wash Area 1 has been the most extensively delineated 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 15 reverse circulation holes were completed on Area 1 between October 2011 and February 2012. All 34 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 for Area 2. Under the guidance of NAE, a combination of 3 core holes (HQ size), 44 reverse circulation holes and 3 large diameter core (PQ size) were completed on Area 2 between July 2012 and September 2012. For the infill drilling program conducted on Area 2, Norwest provided an onsite 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.

Infill drilling is currently in progress in Area 1. The 5 core holes (HQ size), 26 reverse circulation holes and 7 large diameter core (PQ size) holes were planned for the purposes of providing additional grade data that is consistent with current laboratory standards as well as providing data bulk sample data for rock strength and material processing simulations. The in progress drill hole locations are illustrated in Figure 9.2.

1.5 MINERAL RESOURCES

A 3DGBM for Area 1 and Area 2 was constructed from data obtained from PRC, who had acquired the historic downhole lithologic and assay base data from a third party. Data validation tasks involved statistical evaluations of base data, site visits conducted by the report authors, a validation drilling program conducted by PRC in Area 1 and PRC infill drilling program for Area 2, and statistical analysis of the PRC twin hole assay values to that of the historic holes that were twinned.

The comparison of the K_2O and Al_2O_3 values from the Area 1 PRC holes with their respective twin ESI holes 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. PRC drilling in Area 2 did not twin any historic holes; however, the spatial distribution of the mineralized zone was confirmed from infill drilling.

Norwest believes the PRC validation and infill drilling program has adequately tested the Area 1 and Area 2 deposits. 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 and infill drilling program. While the ALS results show higher concentrations than previously indicated in the ESI drilling data, the ALS analyses confirm the presence of mineralization and indicate grades determined from the ESI drilling data will be conservative estimations.

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 historic and current drilling database and on a series of interpreted geological cross sections constructed through Area 1 and Area 2.

Resource classification is based on set distances from drilhole sample intervals in 3D space. These distances were based on semi-variogram analysis of K2O and Al2O3 sample data (Table 1.1).

TABLE 1.1 RESOURCE ASSURANCE CRITERIA FROM VARIOGRAPHY

Compound	Measured	Indicated	Inferred
K_2O	<150ft	<350ft	<1,000ft*
Al_2O_3	<150ft	< 250ft	<1,000ft*

Other estimation criteria include assumed density of alunite and waste established at $13ft^3$/short ton (2.077 short tons/yd^3).

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 1.2 shows the estimated classified resource for the Area 1 at increasing incremental K_2O cut-off grades and Table 1.3 shows the estimated classified resource for the Area 2 at increasing incremental K_2O cut-off grades.

TABLE 1.2 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 (%)	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	105,670,000	3.18	5.88	13.94	27.95	37.74	3,359,000	6,212,000	14,730,000	29,531,000	39,878,000
	1	103,838,000	3.22	5.96	13.90	28.32	37.63	3,345,000	6,185,000	14,434,000	29,403,000	39,074,000
	2	84,454,000	3.59	6.64	14.23	31.54	38.52	3,030,000	5,604,000	12,017,000	26,638,000	32,532,000
	3	53,707,000	4.21	7.78	15.34	36.98	41.53	2,259,000	4,178,000	8,238,000	19,863,000	22,302,000
	4	26,949,000	4.93	9.11	17.18	43.30	46.52	1,327,000	2,454,000	4,631,000	11,668,000	12,537,000
	5	10,157,000	5.76	10.64	20.03	50.59	54.23	585,000	1,081,000	2,035,000	5,139,000	5,508,000
INDICATED	0	53,306,000	3.19	5.90	13.98	28.03	37.85	1,699,000	3,143,000	7,453,000	14,939,000	20,176,000
	1	52,447,000	3.23	5.97	13.91	28.37	37.66	1,692,000	3,130,000	7,296,000	14,878,000	19,751,000
	2	42,897,000	3.58	6.61	14.25	31.44	38.59	1,534,000	2,837,000	6,114,000	13,485,000	16,552,000
	3	27,450,000	4.20	7.76	15.36	36.89	41.59	1,152,000	2,130,000	4,217,000	10,125,000	11,417,000
	4	14,351,000	4.86	8.99	16.85	42.72	45.62	697,000	1,290,000	2,419,000	6,130,000	6,548,000
	5	4,443,000	5.85	10.81	20.69	51.40	56.02	260,000	480,000	919,000	2,284,000	2,489,000
MEASURED AND INDICATED	0	158,977,000	3.18	5.88	13.95	27.97	37.78	5,059,000	9,355,000	22,183,000	44,471,000	60,054,000
	1	156,285,000	3.22	5.96	13.90	28.33	37.64	5,037,000	9,315,000	21,729,000	44,281,000	58,826,000
	2	127,350,000	3.58	6.63	14.24	31.51	38.54	4,564,000	8,441,000	18,131,000	40,124,000	49,084,000
	3	81,156,000	4.20	7.77	15.35	36.95	41.55	3,411,000	6,308,000	12,455,000	29,988,000	33,719,000
	4	41,300,000	4.90	9.07	17.07	43.09	46.21	2,025,000	3,744,000	7,050,000	17,798,000	19,084,000
	5	14,600,000	5.78	10.69	20.23	50.84	54.77	844,000	1,561,000	2,954,000	7,422,000	7,997,000
INFERRED	0	392,000	3.29	6.08	17.17	28.92	46.47	13,000	24,000	67,000	113,000	182,000
	1	392,000	3.29	6.08	17.17	28.92	46.47	13,000	24,000	67,000	113,000	182,000
	2	329,000	3.58	6.62	17.96	31.48	48.62	12,000	22,000	59,000	104,000	160,000
	3	175,000	4.54	8.39	22.43	39.87	60.71	8,000	15,000	39,000	70,000	107,000
	4	92,000	5.64	10.42	29.23	49.55	79.12	5,000	10,000	27,000	46,000	73,000
	5	76,000	5.90	10.91	30.28	51.88	81.96	4,000	8,000	23,000	40,000	62,000

TABLE 1.3 CLASSIFIED RESOURCE ESTIMATE FOR THE BLAWN MOUNTAIN AREA 2 ALUNITE DEPOSIT

RESOURCE CLASSIFICATION	CUTOFF GRADE K_2O (%)	IN SITU (TONS)	IN SITU GRADES					CONTAINED RESOURCES				
			K_2O (%)	K_2SO_4 (%)	AL_2O_3 (%)	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	110,164,000	3.05	5.64	13.02	26.79	35.26	3,358,000	6,210,000	14,347,000	29,518,000	38,839,000
	1	106,224,000	3.14	5.81	13.24	27.61	35.84	3,336,000	6,170,000	14,064,000	29,331,000	38,074,000
	2	92,222,000	3.37	6.24	13.49	29.66	36.52	3,112,000	5,754,000	12,441,000	27,354,000	33,680,000
	3	62,137,000	3.77	6.96	13.99	33.10	37.86	2,339,000	4,326,000	8,690,000	20,566,000	23,527,000
	4	19,232,000	4.50	8.33	15.36	39.59	41.59	866,000	1,602,000	2,954,000	7,615,000	7,998,000
	5	2,605,000	5.49	10.15	17.64	48.24	47.77	143,000	264,000	460,000	1,257,000	1,244,000
INDICATED	0	368,380,000	2.99	5.52	13.01	26.24	35.21	10,996,000	20,335,000	47,908,000	96,667,000	129,696,000
	1	358,218,000	3.05	5.65	13.14	26.84	35.58	10,936,000	20,225,000	47,073,000	96,142,000	127,437,000
	2	301,900,000	3.32	6.15	13.40	29.21	36.28	10,032,000	18,552,000	40,464,000	88,192,000	109,543,000
	3	197,469,000	3.73	6.90	13.94	32.79	37.72	7,366,000	13,621,000	27,517,000	64,751,000	74,495,000
	4	58,500,000	4.47	8.26	15.25	39.25	41.27	2,612,000	4,830,000	8,919,000	22,962,000	24,145,000
	5	6,417,000	5.46	10.09	17.89	47.96	48.43	350,000	647,000	1,148,000	3,078,000	3,107,000
MEASURED AND INDICATED	0	478,545,000	3.00	5.55	13.01	26.37	35.22	14,354,000	26,545,000	62,255,000	126,186,000	168,535,000
	1	464,442,000	3.07	5.68	13.16	27.02	35.64	14,273,000	26,395,000	61,137,000	125,473,000	165,511,000
	2	394,122,000	3.33	6.17	13.42	29.32	36.34	13,144,000	24,307,000	52,904,000	115,546,000	143,222,000
	3	259,606,000	3.74	6.91	13.95	32.86	37.76	9,705,000	17,947,000	36,208,000	85,317,000	98,021,000
	4	77,733,000	4.47	8.27	15.27	39.34	41.35	3,478,000	6,432,000	11,873,000	30,577,000	32,143,000
	5	9,022,000	5.47	10.11	17.82	48.04	48.24	493,000	912,000	1,607,000	4,335,000	4,352,000
INFERRED	0	259,034,000	2.84	5.25	12.77	24.96	34.57	7,354,000	13,600,000	33,079,000	64,649,000	89,550,000
	1	250,769,000	2.91	5.37	12.83	25.55	34.73	7,287,000	13,476,000	32,166,000	64,063,000	87,080,000
	2	210,440,000	3.15	5.83	13.12	27.69	35.53	6,629,000	12,259,000	27,616,000	58,274,000	74,762,000
	3	123,406,000	3.63	6.71	13.80	31.88	37.36	4,475,000	8,275,000	17,029,000	39,337,000	46,100,000
	4	31,293,000	4.43	8.19	15.19	38.95	41.11	1,387,000	2,564,000	4,752,000	12,190,000	12,865,000
	5	3,013,000	5.35	9.89	18.36	47.02	49.69	161,000	298,000	553,000	1,417,000	1,497,000

The resources for Area 1 are very similar from Norwest's 2012 reported resource. The only material change from previous resource estimates is the removal of 2 feet (ft) of material on the surface to account for potential weathering on near-surface mineralization. At a 1% K_2O cut-off grade, the Area 1 combined measured plus indicated resource of 156Mt carrying an average grade of 3.22% K_2O and 13.90% Al_2O_3. The calculated potassium sulfate grade (K_2SO_4) at a 1% K_2O cut-off grade is 5.96%. Increasing the cut-off grade to 3% K_2O reduces the combined in situ tons of material to 81Mt. Average grade at a 3% K_2O cut-off is 4.20% K_2O and 15.35% Al_2O_3 with a calculated equivalent grade of 7.77% K_2SO_4. Approximately 66% of the identified resources are classified as measured and 34% as indicated resource.

At a 1% K_2O cut-off grade, the Area 2 combined measured plus indicated resource of 464Mt carrying an average grade of 3.07% K_2O and 13.16% Al_2O_3. The calculated potassium sulfate grade (K_2SO_4) at a 1% K_2O cut-off grade is 5.68%. Increasing the cut-off grade to 3% K_2O reduces the combined in situ tons of material to 260Mt. Average grade at a 3% K_2O cut-off is 3.74% K_2O and 13.95% Al_2O_3 with a calculated equivalent grade of 6.92% K_2SO_4.

Approximately 15% of the identified resources are classified as measured, 50% as indicated resource and 35% as inferred resource.

There are no mineral reserve estimates associated with 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

1.6 MINING METHODS

Mining operations at Blawn Mountain will utilize conventional truck/shovel mining techniques employing a large hydraulic backhoe as well as a front-end-loader to load end-dump mining trucks to remove alunite and waste material from mining areas. The mining areas developed for this PEA were defined mainly by the geologic resources and extents of the alunite body as well as the Potassium oxide grade. The targeted mining production levels are defined by the amount of K_2SO_4 required to meet project plant production levels. On a steady-state basis, K_2SO_4 requirements are 750,000 tons per year (tpy) which equates to roughly 17.2Mt of Run-of-Mine (ROM) alunite. ROM production requirements vary depending on the grade of K_2O being released from mining operations.

Future operations at Blawn Mountain will begin mining in Area 1 and transition to Area 2 once the targeted alunite has been exhausted from Area 1. Alunite and waste will be removed using area and bench mining techniques utilizing multiple lifts and mining faces to assist with in-pit material blending efforts.

At this stage of development, the anticipated mine life is 30 years, with a two year ramp-up period. This first two years of mining (Year -1 and Year 1) are considered a ramp-up period with full production occurring in Year 2. The material removed during Year -1 is considered part of the 30 year mine life. The strip ratio over the mine life averages approximately 0.12:1 (cubic yards of waste: ROM alunite tons). Waste material consists primarily of overburden and mineralized zones of sufficient quality to process. Table 1.4 presents waste volumes, ROM alunite tons, ROM alunite qualities and stripping ratios associated with each mining area. Additionally, ROM alunite tons are further delineated into the assurance categories of measured, indicated and inferred. A mining recovery of 98% was utilized and reflects that minimal material loss is anticipated as the majority of the mining will occur in the alunite zone.

TABLE 1.4 MINE QUANTITIES & QUALITIES

Area	*Alunite (Mt)	Al_2O_3 (%)	K_2O (%)	NA_2O (%)	Waste (Myd^3)	Strip Ratio (yd^3/ton)	*Alunite (Mt) Measured	*Alunite (Mt) Indicated	*Alunite (Mt) Inferred
1	121.6	13.77	3.40	0.51	8.1	0.07	81.1	40.4	0.07
2	387.9	13.14	3.22	0.33	52.7	0.14	63.7	202.8	121.4
Total	**509.5**	**13.30**	**3.26**	**0.37**	**60.8**	**0.12**	**144.8**	**243.2**	**121.5**

*ROM

As Alunite is mined, it will be transported to either the stockpile area or directly to the primary crusher. Waste that is generated from mining will initially be placed in out-of-pit piles. As mining progresses and sufficient area becomes available, it is anticipated that waste material will be placed back into prior excavations. In a general sense, the majority of waste from Area 1 will be placed out-of-pit and waste from Area 2 will be hauled and placed into the Area 1 mine void.

The Mine schedule is summarized in Table 1.5. As previously discussed, mining initiates in Area 1 and when this area is depleted of targeted alunite, operations will shift to Area 2. Full Production is anticipated in Year 2.

TABLE 1.5 MINE SCHEDULE

Year	-1	1	2	3	4	5	6-10	11-15	16-20	21-30
Waste (Myd^3)	0.4	0.7	0.9	1.2	1.2	1.2	3.6	14.4	14.0	23.2
Alunite (Mt)	4.5	9.2	15.1	17.5	17.9	17.6	83.6	86.3	86.0	172.0
Al_2O_3 (%)	14.28	14.77	14.10	12.99	12.88	13.07	13.56	12.99	13.18	13.29
K_2O (%)	4.12	4.02	3.67	3.17	3.10	3.14	3.31	3.21	3.22	3.22
NA_2O (%)	0.20	0.34	0.42	0.47	0.49	0.54	0.42	0.30	0.36	0.35

1.7 METALLURGICAL TESTING AND MINERAL PROCESSING

Alunite is a naturally occurring semi-soluble salt mineral which occurs as a hydrothermal-alteration product in feldspathic (orthoclase feldspar-rich) igneous rocks. It contains potassium, sulfur, and alumina. The Potash Ridge Alunite Project will mine alunite, or hydrous potassium aluminum sulfate, $KAl_3(SO_4)_2(OH)_6$, from the Blawn Mountain deposits in Utah. The alunite will be processed for the production of 750,000tpy of fertilizer-grade (92.5%) potassium sulfate (K_2SO_4), also called sulfate of potash (SOP), approximately 1,800,000tpy of co-product concentrated (98wt.%) sulfuric acid (H_2SO_4), and approximately 3,620,000tpy of an alumina/silicate [$(AlO)_2SiO_3$] product consisting of a mixture of alumina, silica, and oxygen containing approximately 50% Al_2O_3. It is anticipated that the stockpiled alumina/silicate product will eventually be re-processed into an alumina (Al_2O_3) product.

The recovery methods for extracting potassium as SOP from the alunite, as envisioned, consists of the following major unit operations:

- Crushing and grinding of run-of-mine alunite.
- Conditioning with flotation reagents the finely-ground alunite pulped with water.
- Multi-stage flotation to recover alunite concentrate and tailing as slurry.
- Thickening and clarification of alunite concentrate and tailings slurry.
- Filtration of concentrate thickener underflow to obtain alunite filter cake.
- Pumping tailings to impoundment.
- Reuse of thickener overflow and tailings pond water for water conservation.
- Drying and Calcining alunite concentrate obtained as filter cake.
- Recovery of alunite -bearing dust in calciner off-gases.
- Routing sulfur oxides-rich calciner off-gases as feed to sulfuric acid plant.
- Production of concentrated sulfuric acid as by-product.
- Leaching calcines with hot water to extract SOP.
- Filtration of water-leached slurry to recover SOP-rich solution.
- Stockpiling alumina/silicate-rich filter cake for alumina recovery at a later date.
- Evaporation-Crystallization of SOP crystals from SOP-rich solution.
- Compacting for densification of SOP product crystals.
- Packaging, storage and load out of product SOP.
- Bleed stream treatment for controlling impurity buildup.

1.8 ENVIRONMENTAL AND PERMITTING

Mining and processing operations in the United States must comply with all applicable federal and state regulations. Operations located in Utah require compliance with federal as well as state mining and environmental regulations.

1.8.1 State Regulations

Utah has primacy over major environmental disciplines including mining, air and water permitting. The project is located in southwestern Utah on State controlled land therefore most permits that will need to obtain will be issued by State agencies.

1.8.2 Federal Regulations

Vast amounts of land in Utah are managed by the Bureau of Land Management (BLM) due to the presence of either federally-controlled surface land or federal mineral ownership. In Utah the control of large parcels of land and minerals were granted to the state to provide a source of revenue from the management of surface use or mineral development. These “state sections” and other lands obtained through additional grants or exchanges from the federal government are managed by the state School & Institutional Trust Lands Administration (SITLA).

When federal lands (minerals or surface) are impacted, federal approvals are required by the applicable land management agency, most commonly the BLM. Federal actions requiring permits or approvals trigger an environmental review under the National Environmental Policy Act (NEPA). The level of scrutiny a project receives is based upon the land management agencies' discretion and the significance of impacts to the environment. When impacts are potentially more significant, an Environmental Impact Statement (EIS) may be required. This process can take several years and involves multiple agencies and extensive public input. When impacts are less significant, the federal permit or approval can be substantiated with an Environmental Assessment (EA), which is an abbreviated process that generally can be accomplished within 6-12 months. Regardless of the process that applies, environmental impacts of the project must be evaluated prior to receiving federal authorization to proceed.

The PRC project is located on SITLA controlled mineral and surface land. Because SITLA controls both surface and mineral resources on the parcel, the potential for federal involvement under NEPA is minimal.

1.8.3 Major Operating Permit and Authorizations

The following discussion and Table 1.6 identifies the major permits and authorizations that need to be obtained prior to the construction and start-up of the Blawn Mountain project.

TABLE 1.6 MAJOR REQUIRED PERMITS

Major Permits or Approvals	Issuing Agency
Mining Permit	Division of Oil, Gas and Mining
Water Appropriations	Utah Office of State Engineer
Groundwater Discharge Permit	Utah Division of Water Quality
Air Quality Permit	Utah Division of Air Quality
Storm Water and Discharge Permit	Utah Division of Water Quality
Dredge and Fill Permit	U.S. Army Corps of Engineers
County Conditional Use Permit	Beaver County
Federal Right-of-Way	U.S. Bureau of Land Management

This report covers only the major permits required. In addition to the permits discussed in this report, other ancillary environmental permits and authorizations include spill control and response plans, hazardous waste management/transportation authorization, authorization for use of low level radioactive sources, and hazardous chemical training for employees.

Permits and authorizations required for the Blawn Mountain Alunite Project are common to most major mining and processing operations in Utah and throughout the U.S. The schedule to obtain the permits listed above may vary somewhat. However, integration of sound design principles and some flexibility with respect to the project footprint and environmental impacts as well as the implementation of mitigation measures should result in issuance of all required permits for the project. The Federal right-of-way (ROW) approval is required to improve the access road. That approval is being pursued by Beaver County under a separate action that is not connected directly to the project.

1.9 MARKETS

The products produced from the Blawn Mountain project will be marketed to the global fertilizers industry, which is a large component of the global chemicals industry. Fertilizers consist of essential plant nutrients which 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. The key components of agricultural fertilizers are nitrogen (anhydrous ammonia and urea), phosphates (ammonium phosphates and superphosphates derived from phosphate rock), and potassium (potash). Sulphur has gained increased attention in the fertilizer industry over the past several years due to the realization that crops were becoming sulphur deficient. Sulphur is necessary for the production of protein, fostering activity and the development of enzymes and vitamins. 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.

1.10 ECONOMIC ANALYSIS

Pre-production cash outflows total $1.3B over the two year construction period. Cash flow is positive beginning in Year 1. Payback occurs at the end of Year 5 which is 7 years after the initial investment. Cash flow after payback averages $288M per year for a total net cash flow of $7.2B over the life of the project. The internal rate of return for the project is 21.3%. The project economics are more sensitive to selling price than changes in capital or operating costs.

1.11 CONCLUSIONS AND RECOMMENDATIONS

The Blawn Mountain Alunite Project has established control of 10,394.2 acres of mineral rights through an Exploration/Option Agreement with SITLA. Past exploration and current validation and analysis has established a mineral resource of enriched alunite. At a 1% K_2O cut-off grade, there is a combined Area 1 and Area 2 measured plus indicated resource of 621Mt carrying an average grade of 3.11%K_2O and 13.35%Al_2O_3. This technical report has been authored by independent qualified persons and certificates attesting to this are included in the report.

Resource, mining and processing evaluations have shown this deposit to be a property of merit. To develop the project further towards providing the necessary information for prefeasibility level mine planning and higher-level engineering the current Area 1 geologic model would need to be updated with the results of the currently in progress infill drilling program. In addition further drilling is required for Area 1 to define additional resources at greater depths. The recommended 2013 Exploration Program for Area 2 includes 30 drill holes for a total of 21,000ft. Total cost of the proposed exploration is $1.6M.

The PEA is preliminary in nature, and includes inferred mineral resources. Inferred mineral resources are considered too speculative geologically to have technical and economic considerations applied to them that would enable them to be categorized as mineral reserves. Mineral resources that are not mineral reserves do not have demonstrated economic viability. Accordingly, there can be no certainty that the results estimated in the PEA will be realized.

2 INTRODUCTION

Norwest Corporation (Norwest) has prepared this Technical Report on the Blawn Mountain project at the request of Potash Ridge Corporation (PRC). The purpose of the report is to evaluate the potential economic viability of mining resources 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 School and Institutional Trust Lands Administration (SITLA). The property is located approximately 30 miles southwest of the town of Milford, Utah and 30 miles east of the Nevada border.

This Preliminary Economic Analysis (PEA) Technical Report has been prepared in accordance with the current requirements of National Instrument 43-101. Several data sources have been utilized during the preparation of this PEA including:

- Results of exploration documented in various public reports and on recent drilling data supplied by PRC
- 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.
- Certain processing plant operating costs provided by 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 U.S. Federal Tax regulations as they relate to the percentage depletion calculation used in the economic analysis.
- Mine operating and portions of the plant operating costs were derived from the 2012 Western Mine and Mill Cost Estimating guide.

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 Exploration/Option Agreement,

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 Blawn Mountain primarily for the manufacture of potash (potassium sulfate). However, following initial development PRC anticipates multiple products from the alunite including alumina, and sulfur products.

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 (Figure 4.1) 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, 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 United States. Alunite has long been known to have value for alumina, potassium, and sulfur, though three obstacles have often limited development: (1) adequate size of deposit, (2) concentrations of commercial components, and (3) 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 U.S. 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 U.S. whereas all of the bauxite used in the U.S. is imported. Disruption of the bauxite supply can be a concern. 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, alumina and potash. Also, production of potash from alunite is in the form of potassium sulfate which sells for about twice as much as 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 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.

Authors Mr. Kerr and Mr. Henchel visited the Blawn Mountain Alunite Project property on October 30, 2012. Mr. Kerr made his first site visit on February 9, 2012 and has made several site visits since that time. Authors Mr. Todd, Mr. Nash and Mr. Nath visited the property on March 15, 2012. 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 available at the time of preparation and data supplied by outside sources. Norwest staff has not conducted any independent field work for the preparation of this report and have relied on the results of exploration documented in various public reports and on recent drilling data supplied by PRC.

PRC has supplied the appropriate documentation that supports the Exploration/Option Agreement it holds with the State of Utah to be in good standing. The existence of encumbrances to the agreement has not been investigated. Other Norwest personnel assisted in the compilation and digitization of the historical data and documents and the information contained within, in developing a generalized mining layout and methodology, in developing preliminary concepts for mine support facilities, and addressing current metallurgical testing results and developing a conceptual approach to alunite processing. All this work was reviewed and deemed reasonable for this level of study by the authors.

Norwest relied on Intermountain Consumer Professional Engineers, Inc. (ICPE) to prepare Sections 13, 17 and the portion of Section 21 that relates to the processing plant. Mr. Robert Nash and Mr. Ravi Nath are acting as Qualified Persons with respect to those Sections.

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

Norwest relied on certain processing plant operating costs provided by 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. Our reliance on this information applies to Sections 21 and 22.

Norwest relied on PRC’s interpretation of the U.S. Federal Tax regulations as they relate to the percentage depletion calculation used in the economic analysis. Our reliance on this information applies to Section 22.

4 PROPERTY DESCRIPTION AND LOCATION

The PRC Blawn Mountain property 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 T.29S., R.15W., sec. 13-16, 21-29, 32-36 and T.30S., R.15W., sec 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) administered through SITLA. The agreement consists of a tract of land that covers 11,549.2 acres (Figure 4.2). Table 4.1 provides a legal description of the controlled area. The Exploration/Option Agreement is issued to Utah Alunite, LLC, a 100% owned entity of PRC. In April 2012, Utah Alunite, LLC was merged into Utah Alunite Corporation, a 100% owned entity of PRC and the mineral lease was assigned by Utah Alunite, LLC to Utah Alunite Corporation. There are two pre-existing mineral tracts consisting of a 40 acre tract (ML48699.0MC) along the western edge of the project area and a 155 acre tract (ML 48698.0 MC) within the PRC land package designated under the exploration agreement. . Another mineral tract of 640 acres is located approximately one mile east of the PRC property. Remaining lands surrounding the PRC property are predominantly a mix of federal lands administered by the U.S. Bureau of Land Management (BLM) and state lands administered by SITLA.

The SITLA agreement is a combined metalliferrous minerals and potash exploration and an option to a mining lease agreement with the following stipulations: 1) three year lease; 2) bonus payment of $200,000; 3) $6/acre each year ($69,300/annum). At the end of the agreement 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 at 5 year intervals after primary term provided 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/acre as required by statute; in addition $4/acre advanced minimum royalty which would be increased at $1/acre 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.

TABLE 4.1 LEGAL DESCRIPTION OF SITLA PROPERTY

T29S,R15W, SLB&M		**ACRES**
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. Area 1 is the primary focus of this report and has been the primary focus of past and current exploration efforts.

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 currently working to secure additional mineral control of the 155 acre lease that extends across a portion of 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 property is located about 20 miles west of the Union Pacific 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 1,420,000ft 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 that were eligible for the National Register (Perry, 1977). U.S. BLM (1977) noted a high density of archeological sites south and southwest of the ESI-proposed mine site that were thought to be potentially important.

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, U.S. 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 alunite mine and plant would be cooler throughout the year than at Milford because Blawn Mountain is at higher elevation. Average annual precipitation at Milford is 8.4 inches 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 (U.S. BLM, 1977).

5.4 ENERGY CORRIDORS

Two energy corridors pass to the east of the Blawn Mountain tract 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 (U.S. 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 (U.S. BLM, 2011).

5.5 GRAZING

A grazing allotment map (U.S. BLM, 2011) 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 property. The entire Blawn Mountain Alunite tract is covered by grazing allotments administered by SITLA. The alunite tract is not within a HMA but the Four Mile HMA adjoins the south boundary of the Blawn Mountain tract 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 to the 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 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 area 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 toward Escalante Valley to the southeast and Willow Creek drainage carries runoff into Wah Wah Valley to the northeast.

5.8 SEISMOLOGY

Blawn Mountain 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 Exploration/Option Agreement tract is entirely composed of Utah State-owned land surface 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 Alunite property 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 Union Pacific Railroad route connecting Salt Lake City, Utah to Las Vegas, Nevada passes approximately 20 miles to the east of the Blawn Mountain property.

5.11 VEGETATION

The Blawn Mountain area 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. An inventory by the BLM revealed no threatened or endangered species of vegetation (U.S. BLM, 1977).

5.12 WATER RESOURCES

The Blawn Mountain area has no perennial streams. Water to support mining and milling will need to be sourced from ground water. USGS studies indicate substantial groundwater resources are present in the Wah Wah and nearby Pine Valley drainages. PRC is in the process of securing water rights and resources to support mining and milling.

5.13 WILDERNESS DESIGNATION

The Blawn Mountain area has not been designated for study or inclusion for wilderness. In 1999 the BLM re-inventoried its lands for suitability for classification of U.S wilderness designation. Part of the Wah Wah Range north of the Blawn Mountain tract met the wilderness re-inventory criteria. The southern boundary of the re-inventoried Central Wah Wah wilderness area is about five miles north of the northern border of the Blawn Mountain tract.

5.14 WILDLIFE

Deer, wild horse, antelope, cougar, raptors and other birds, coyote, bobcat, and fox all are common animals in the area. No endangered species are known to inhabit the Blawn Mountain area (U.S. BLM, 1977). A BLM map of wildlife management areas for the Cedar City Field Office (U.S. BLM, 2011) shows no special management areas within the property.

5.15 CONFLICTING DEVELOPMENT

The Blawn Mountain 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 area. The Cedar City Field Office of the BLM compiled a draft map for the Resource Management Plan (USBLM, 2011). This draft map indicates that the tracts with geothermal, wind power, and solar energy potential would not conflict with Blawn Mountain 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 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 the Blawn Mountain Alunite 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 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 3,000 ton (t) (Krahulec, 2007) sample from a pit identified as number 5. The Golden pilot plant (designed by Hazen) had the capacity to process 12 to 18 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 500,000tpy of alumina with by-products of 450,000tpy of sulfuric acid, 250,000tpy of potash (sulfate of potash), 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 Blawn Mountain 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 northwest in central Utah to provide fuel for the alunite plant. Local water rights were acquired and water wells have been 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 challenging economics 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 Alumet 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. Figure 9.2 shows the drilling completed by PRC.

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, 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.

7 GEOLOGICAL SETTING AND MINERALIZATION

The Blawn Mountain Alunite deposit 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, 1) thrust faulted during the Sevier orogeny, 2) buried under a thick layer of regionally distributed Oligocene volcanic rocks and locally derived volcanic rocks, 3) extended to the west by the Basin and Range event, 4) 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), and 5) 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 below.

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).

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 Quarzite	
Proterozoic	Precambrian			Mutual	

7.3 STRUCTURAL GEOLOGY

The Blawn Mountain deposit 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 deposit. 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.

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	Bald Hills Tuff
					Bullion Canyon Volcanics	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, three of which occur within PRC's exploration tracts. 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 (>0.5 inch), 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 Alunite 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 ridge now referred to as Area 1. Figure 9.1 shows the rotary drill locations completed by ESI at Blawn Mountain.

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 NAE, a combination of 19 core holes and 15 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 34 drill holes were twinned to locations of previous drill holes completed by ESI. The Area 1 twin hole program was followed up by infill drilling program in Area 2. Under the guidance of NAE, a combination of 3 slim-core holes, 3 large diameter cores and 44 reverse circulation holes were completed on Area 2 between July 11, 2012 and September 17, 2012. Figures 9.2 and 9.2 show the drilling completed by PRC.

Infill drilling and analysis is currently in progress in Area 1. The 5 core holes (HQ size), 26 reverse circulation holes and 7 large diameter core (PQ size) holes are planned for the purposes of providing additional grade data that is consistent with current laboratory standards as well as providing material for rock strength characterization and collecting bulk samples for alunite processing simulations. The in progress drill hole locations are illustrated in Figure 9.2.

10 DRILLING

ESI company records indicate a total of 320 drill holes were completed on their NG Alunite deposit. All but three of these holes are located within the current PRC exploration areas. The three holes not located within the PRC tracts are located a short distance outside the property boundary to the southeast. Two-hundred-eighty-seven holes were completed on Area 1, eighteen holes at Area 2, and twelve holes at Area 3. Six of the drill holes located in Area 2 are located within the 155 acre mineral tract (ML 48698.0 MC) not under PRC control.

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 holes were completed at Area 1 for a total of 4,438ft, fourteen holes were completed at Area 2 for a total of 2,865ft, nine holes were completed at Area 3 for a total of 2,590ft, and three holes were completed on a fourth area outside the current PRC lease 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.

Table 10.1 summarizes the drilling completed at Blawn Mountain. 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.

TABLE 10.1 DRILL HOLE SUMMARY

Drill Hole ID	USP-South NAD27 (ft)			Total Depth (ft)	Lithology	Assays	Model
	Easting	Northing	Elevation				
ESI Drilling							
A1	1062660	10421295	7120	255	X	X	X
A2	1063582	10420152	6920	290	X	X	X
A3	1062021	10420051	6940	455	X	X	X
A4	1064407	10420741	6820	140	X	X	X
A5	1062701	10420364	7020	100	X	X	X
A6	1062001	10420580	7200	410	X	X	X
A7	1061837	10421134	7380	630	X	X	X
A8	1061617	10421159	7340	100	X	X	X
A9	1061164	10421159	7360	410	X	X	X
A10	1061824	10421672	7215	100	X	X	X
A11	1061954	10422450	7120	300	X	X	X
A12	1061907	10422082	7180	400	X	X	X
B1	1064970	10429344	7140	315	X	X	X
B2	1067658	10432547	7340	260	X	X	X
B3	1066478	10432175	7350	425	X	X	X
B4	1065866	10430843	7140	150	X	X	X
B5	1064895	10427911	6980	100	X	X	X
B6	1068044	10432016	7250	200	X	X	X
B7	1068644	10432643	7140	135	X	X	X
B8	1069421	10435685	7060	170	X	X	X
B9	1070452	10435418	7020	350	X	X	X
B10	1069620	10433459	7020	280	X	X	X
B11	1070614	10434708	6940	230	X	X	X
B12	1067981	10430389	7030	60	X	X	X
B12A	1067926	10430294	7000	200	X	X	X
B13	1066710	10431868	7280	300	X	X	X
B14	1067196	10430631	7050	360	X	X	X
B15	1065718	10432131	7120	130	X	X	X
B16	1070162	10433301	6940	240	X	X	X
B17	1067278	10431231	7070	110	X	X	X
C1	1060515	10435797	7446	890	X	X	X
C2	1057378	10433218	7460	40	X	X	X
C3	1056141	10431792	7480	80	X	X	X
C4	1058714	10434443	7330	250	X	X	X
C5	1060261	10436473	7370	270	X	X	X
C6	1060427	10436057	7380	400	X	X	X
C7	1062072	10436503	7595	600	X	X	X
C8	1062548	10436391	7620	450	X	X	X
C8A	1062540	10436391	7620	30		X	

Drill Hole ID	USP-South NAD27 (ft)			Total Depth (ft)	Lithology	Assays	Model
	Easting	Northing	Elevation				
C8B	1062542	10436385	7620	60		X	
C9	1060460	10435503	7340	650	X	X	X
C10			7460	355		X	
C10A	1059312	10435011	7457	620	X	X	X
C11	1061984	10436870	7420	240	X	X	X
C12			7470	50		X	
C12A	1061262	10436403	7470	220	X	X	X
C13	1062091	10435899	7380	270	X	X	X
C14	1059229	10434558	7380	200	X	X	X
C15	1059173	10435230	7320	170	X	X	X
C16	1055326	10431521	7500	43	X	X	X
C17				10	X	X	
C18	1061764	10436294	7404	11	X	X	X
C19				12	X		
C20				15	X		
C21				11	X		
C22				11	X		
C23A				10	X		
C23B				23	X		
C24				10	X		
C25				23	X		
C26				11	X		
C27				11	X		
C28				11	X		
C29				11	X		
C30				11	X		
C31				23	X		
C32				23	X		
C33				23	X		
C34				23	X		
C34A	1059223	10434453	7455	23	X	X	X
C35				23	X	X	
C36				23	X	X	
C37				11	X	X	
C38	1061999	10435961	7408	47	X	X	X
C39	1061474	10436083	7435	47	X	X	X
C40				30	X		
C41				23	X	X	
C42				12	X	X	
C43				35	X	X	
C44	1060409	10436549	7380	60	X	X	X

Drill Hole ID	USP-South NAD27 (ft)			Total Depth (ft)	Lithology	Assays	Model
	Easting	Northing	Elevation				
C45				30	X	X	
C46				20	X	X	
C47				15	X		
C48				12	X		
C49				10	X		
C50				12	X	X	
C51				30	X	X	
C52				10	X		
C53				12	X	X	
C54							
C55	1062549	10435630	7189	140	X	X	X
C56				8	X		
C57				7	X		
C58				30	X		
C59				12	X	X	
C60				12	X		
C61				20	X		
C62	1061403	10436915	7280	20	X	X	X
C63				24	X		
C64	1062490	10437351	7251	71	X	X	X
C65	1062671	10437218	7252	20	X	X	X
C66	1062898	10437072	7213	40	X	X	X
C67	1063191	10436935	7190	12	X	X	X
C68				30	X		
C69	1063567	10436482	7161	40	X	X	X
C70				12	X		
C71				20	X		
C72	1062912	10435760	7176	40	X	X	X
C73				40	X		
C74				12	X		
C75				12	X		
C76				20	X	X	
C76A				10	X		
C76B				20	X		
C77	1061745	10435743	7275	70	X	X	X
C78				30	X		
C79	1061452	10435850	7315	50	X	X	X
C80				60	X	X	
C81				10	X		
C82	1061045	10435867	7360	60	X	X	X
C83				12	X		

Drill Hole ID	USP-South NAD27 (ft)			Total Depth (ft)	Lithology	Assays	Model
	Easting	Northing	Elevation				
C84				10	X		
C85				60	X	X	
C86				35	X	X	
C87				36	X	X	
C87A				60	X	X	
C88	1060317	10435801	7453	60	X	X	X
C89	1060668	10435901	7460	60	X	X	X
C90				60	X	X	
C91				15	X		
C91A				20	X	X	
C91B				18	X		
C91C	1062272	10436468	7645	80	X	X	X
C92				20	X		
C92A	1062387	10436455	7650	84	X	X	X
C93				16	X		
C93A	1062663	10436319	7600	70	X	X	X
C94	1061965	10436161	7475	140	X	X	X
C95	1061893	10436108	7450	53	X	X	X
C96	1062126	10436083	7457	60	X	X	X
C97	1059483	10435324	7335	60	X	X	X
C98	1058901	10435101	7343	140	X	X	X
C99	1058725	10434880	7350	140	X	X	X
C100	1058963	10434504	7371	140	X	X	X
C101	1059050	10434285	7259	140	X	X	X
C102	1059362	10434283	7245	136	X	X	X
C103	1059603	10434455	7255	140	X	X	X
C104	1062073	10435541	7223	79	X	X	X
C105	1063426	10436061	7185	118	X	X	X
C106	1063693	10436223	7197	99	X	X	X
C107	1063624	10436403	7167	118	X	X	X
C108			7167	65	X	X	
C109	1062529	10437332	7249	24	X	X	X
C110	1062031	10437260	7278	20	X	X	X
C111				35	X	X	
C112				26	X		
C113A				16	X		
C113B				12	X		
C113C				9	X		
C113D				5	X		
C113E				13	X		
C114				10			

Drill Hole ID	USP-South NAD27 (ft)			Total Depth (ft)	Lithology	Assays	Model
	Easting	Northing	Elevation				
C115	1060202	10435977	7410	57	X	X	X
C116	1061510	10436093	7440	110	X	X	X
C117	1060683	10436692	7362	315	X	X	X
C118	1060794	10436286	7404	180	X	X	X
C119	1061999	10435961	7408	150	X	X	X
C120	1062517	10436056	7428	100	X	X	X
C121	1059764	10435709	7325	68	X	X	X
C121A				6			
C121B				6			
C122				140	X	X	
C123				140	X	X	
C123A				5	X		
C123B				11	X		
C124				150	X	X	
C125	1060472	10435687	7447	400	X	X	X
C126				150	X	X	
C127				130	X	X	
C128				24	X		
C129A				10	X		
C129B				11	X		
C129C				18	X		
C129D				10	X		
C129E				10	X		
C130	1060464	10435949	7446	650	X	X	X
C131				150	X	X	
C132				150	X	X	
C132A				22	X		
C132B				10	X		
C133A				28	X		
C133B				30	X		
C134				77	X	X	
C135	1063021	10436265	7451	210	X	X	X
C136				48	X	X	
C137				100	X	X	
C138				50	X	X	
C139				125	X	X	
C140	1062882	10436468	7451	400	X	X	X
C141				107	X	X	
C142				90	X	X	
C143				150	X	X	
C144	1063014	10435790	7177	95	X	X	X

Drill Hole ID	USP-South NAD27 (ft)			Total Depth (ft)	Lithology	Assays	Model
	Easting	Northing	Elevation				
C145	1060341	10436276	7382	545	X	X	X
C146				140	X	X	
C146A				10	X		
C147				144	X	X	
C148	1059982	10435999	7360	48	X	X	X
C148A				10	X		
C149				20	X		
C149B				10	X		
C149C				5	X		
C150	1062529	10436748	7470	300	X	X	X
C151	1059742	10436598	7191	60	X	X	X
C152	1059749	10436545	7189	47	X	X	X
C153	1062918	10437059	7210	200	X	X	X
C154	1063372	10436832	7167	150	X	X	X
C155	1061673	10436720	7435	250	X	X	X
C156	1061701	10436569	7511	450	X	X	X
C157	1060821	10436105	7439	490	X	X	X
C158	1060737	10436459	7359	330	X	X	X
C159	1061211	10436698	7332	300	X	X	X
C160	1061609	10437045	7289	220	X	X	X
C161	1062737	10436005	7400	300	X	X	X
C162	1062517	10436056	7428	370	X	X	X
C163	1061782	10436042	7411	240	X	X	X
C164	1061345	10436095	7427	420	X	X	X
C165	1061794	10435681	7270	150	X	X	X
C166	1061383	10435846	7197	200	X	X	X
C167	1060690	10435445	7272	450	X	X	X
C168	1060370	10435240	7206	420	X	X	X
C169	1059930	10435211	7237	550	X	X	X
C170A	1060134	10435232	7222	810	X	X	X
C171	1059882	10435569	7357	700	X	X	X
C172	1060164	10435511	7353	635	X	X	X
C173	1060721	10435648	7380	620	X	X	X
C174	1060918	10435801	7338	435	X	X	X
C175	1059883	10434952	7233	900	X	X	X
C176	1059789	10434732	7233	540	X	X	X
C177	1060761	10436266	7403	370	X	X	X
C178	1060260	10436000	7407	465	X	X	X
C178A				15	X		
C179	1060062	10435773	7408	750	X	X	X
C180	1059726	10435417	7346	700	X	X	X

Drill Hole ID	USP-South NAD27 (ft)			Total Depth (ft)	Lithology	Assays	Model
	Easting	Northing	Elevation				
C181	1059579	10435219	7389	810	X	X	X
C182	1062093	10436224	7524	400	X	X	X
C183	1059699	10434971	7336	700	X	X	X
C184	1061761	10436301	7519	450	X	X	X
C185			7313	10			
C185A	1059786	10435220	7313	540	X	X	X
C186	1061295	10436416	7471	400	X	X	X
C187	1059484	10435324	7335	536	X	X	X
C188	1059759	10435850	7304	285	X	X	X
C189	1059418	10434759	7426	540	X	X	X
C190	1059932	10436146	7308	280	X	X	X
C191	1060207	10436705	7312	248	X	X	X
C192	1060607	10436826	7295	250	X	X	X
C193	1061122	10436770	7275	710	X	X	X
C194	1059007	10434776	7458	555	X	X	X
C195	1058920	10434628	7436	580	X	X	X
C196	1059189	10434581	7408	240	X	X	X
C197	1059995	10435897	7371	780	X	X	X
C198	1063224	10436296	7350	255	X	X	X
C199	1062877	10436730	7359	200	X	X	X
C200	1060384	10436168	7392	120	X	X	X
C201				42	X		
C201A	1060327	10436417	7383	79	X	X	X
C202	1063224	10436296	7350	50	X	X	X
C202A				10	X		
C203	1063101	10436492	7370	30	X	X	X
C203A				30	X		
C204	1062459	10436967	7373	79	X	X	X
C205	1062000	10437001	7375	133	X	X	X
C206	1061643	10436871	7373	32	X	X	X
C206A				10	X		
C207	1059603	10434455	7255	154	X	X	X
C208				80	X	X	
C208-45	1060370	10435240	7206	120	X	X	X
C208-60	1060370	10435240	7206	80	X	X	X
C209	1059958	10436054	7344	154	X	X	X
C209A				154	X	X	
C210	1060885	10435719	7316	102	X	X	X
C211				32	X	X	
C211A	1059196	10435168	7367	42	X	X	X
C211B				32	X		

Drill Hole ID	USP-South NAD27 (ft)			Total Depth (ft)	Lithology	Assays	Model
	Easting	Northing	Elevation				
C212				35	X	X	
C212A	1058905	10435015	7373	112	X	X	X
C213A				10	X		
C213B				10	X	X	
C213C				20	X		
C213D	1058798	10434795	7381	134	X	X	X
C214	1061063	10436835	7230	134	X	X	X
C215	1061140	10435871	7330	340	X	X	X
C216				20	X		
C216A	1060603	10436924	7265	92	X	X	X
C217	1060214	10436623	7345	94	X	X	X
C218				134	X	X	
C219				134	X	X	
C220	1061245	10436551	7409	500	X	X	X
C221A				10	X		
C221B				10	X		
C221C				30	X		
C221D				20	X		
C222				60	X	X	
C223				60	X	X	
C224A				10	X		
C224B				10	X		
C224C				20	X	X	
C225				50	X	X	
C225A				60	X	X	
C226				80	X	X	
C226A				72	X	X	
C227				45	X	X	
C227A				120	X	X	
C228	1062459	10436967	7373	200	X	X	X
C229	1062000	10437001	7375	200	X	X	X
C230	1061643	10436871	7373	200	X	X	X
C230A				200	X		
PRC Drill Holes							
BM1	1421006	594151.7	7332	229	X	X	
BM2	1419398	591908.7	7255	146	X	X	
BM3	1421057	593856.7	7470	280	X	X	
BM3A	1421057	593856.7	7470	200	X	X	
BM4	1420055	593454.7	7407	368	X	X	
BM5	1420136	593730.7	7382	480	X	X	
BM6	1418984	592034.7	7408	301	X	X	

Drill Hole ID	USP-South NAD27 (ft)			Total Depth (ft)	Lithology	Assays	Model
	Easting	Northing	Elevation				
BM7	1421779	594323.7	7420	133	X	X	
BM8	1419790	593350.7	7371	790	X	X	
BM8A	1419790	593350.7	7371	800	X	X	
BM9	1420267	593140.7	7447	477	X	X	
BM10	1418803	592230.7	7458	536	X	X	
BM11	1419521	592869.7	7346	550	X	X	
BM11A	1419521	592869.7	7346	457	X	X	
BM12	1422312	593509.7	7428	300	X	X	
BM13	1421247	593303.7	7315	200	X	X	
BM14	1420255	592956.7	7340	370	X	X	
BM14A	1420255	592956.7	7340	273	X	X	
BM14B	1420255	592956.7	7340	650	X	X	
BM15	1420165	592693.7	7206	380	X	X	
BM16	1421886	593352.7	7380	228	X	X	
BM16A	1421886	593352.7	7380	200	X	X	
BM17	1419929	592685.7	7222	410	X	X	
BM17A	1419929	592685.7	7222	820	X	X	
BM18	1421577	593495.7	7411	198	X	X	
BM19	1419960	592964.7	7353	651	X	X	
BM20	1421866	593956.7	7595	400	X	X	
BM21	1419678	592405.7	7233	810	X	X	
BM22	1421141	593548.7	7427	338	X	X	
BM23	1420112	593254.7	7453	198	X	X	
BM24	1420616	593558.7	7436	431	X	X	
BM25	1419677	593022.7	7357	790	X	X	
BM26	1421495	594023.7	7511	420	X	X	
BM27	1421888	593677.7	7524	300	X	X	
BM28	1420259	593402.7	7446	700	X	X	
Area 2 Infill Drilling							
PDH-2-01	1430669	593572	6880	200	x	x	x
PDH-2-03	1430244	592820	7020	220	x	x	x
PDH-2-04	1430542	593194	6906	160	x	x	x
PDH-2-05	1430413	592033	6929	280	x	x	x
PDH-2-06	1430583	591784	6879	620	x	x	x
PDH-2-07	1430230	591754	6940	360	x	x	x
PDH-2-08	1430254	591305	6894	320	x	x	x
PDH-2-09	1429508	590855	7020	400	x	x	x
PDH-2-10	1429949	590657	6936	400	x	x	x
PDH-2-11	1429049	590148	7001	330	x	x	x
PDH-2-12	1429733	590237	6926	350	x	x	x
PDH-2-13	1431117	592112	6924	220	x	x	x

Drill Hole ID	USP-South NAD27 (ft)			Total Depth (ft)	Lithology	Assays	Model
	Easting	Northing	Elevation				
PDH-2-14	1427455	591105	6951	220	x	x	x
PDH-2-15	1428697	590245	7098	320	x	x	x
PDH-2-16	1429055	590510	7013	414	x	x	x
PDH-2-17	1429407	589929	6989	300	x	x	x
PDH-2-18	1427946	590701	6933	100	x	x	x
PDH-2-19	1428546	589471	7070	500	x	x	x
PDH-2-20	1429045	589453	6939	360	x	x	x
PDH-2-22	1427475	589948	7341	220	x	x	x
PDH-2-23	1428630	588951	6947	500	x	x	x
PDH-2-24	1426664	590952	6981	270	x	x	x
PDH-2-26	1427043	589986	7360	370	x	x	x
PDH-2-27	1427678	589588	7281	400	x	x	x
PDH-2-28	1427961	589319	7223	590	x	x	x
PDH-2-29	1426241	590532	7001	200	x	x	x
PDH-2-30	1426534	589831	7368	420	x	x	x
PDH-2-31	1427039	589371	7193	160	x	x	x
PDH-2-32	1427518	589123	7125	570	x	x	x
PDH-2-33	1428375	588669	6938	680	x	x	x
PDH-2-34	1425508	590270	7012	140	x	x	x
PDH-2-36	1426496	589216	7283	504	x	x	x
PDH-2-37	1426722	588683	7112	360	x	x	x
PDH-2-38	1427613	588413	6975	440	x	x	x
PDH-2-39	1425161	589587	6968	200	x	x	x
PDH-2-40	1425759	589118	7157	220	x	x	x
PDH-2-42	1426996	588098	7080	360	x	x	x
PDH-2-43	1425901	587728	6974	300	x	x	x
PDH-2-44	1426504	587242	6904	520	x	x	x
PDH-2-45	1425318	588184	7118	300	x	x	x
PDH-2-46	1429292	593031	7060	355	x	x	x
PDH-2-47	1429927	592985	6990	370	x	x	x
PDH-2-48	1425573	589515	7166	120	x	x	x
PDH-2-49	1426215	589059	7292	300	x	x	x
PDH-2-50	1428083	588387	6932	620	x	x	x
PDH-2-51	1427754	587888	7040	380	x	x	x
PDH-2-52	1424980	587746	7068	200	x	x	x
PDH-2-53	1426155	588700	7247	329	x		x
PDH-2-55	1427983	590023	7210	245	x		x
PDH-2-56	1429960	590973	6956	329	x		x

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 (2.5 inch diameter) 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 reverse-circulation (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. 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 Blawn Mountain. 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.

Following the validation drilling program in Area 1, PRC completed an infill drilling program in Area 2 between July and September, 2012. All drill sites were oriented to provide adequate spatial representation of the deposit. Three (3) of the PRC holes were drilled using wire-line slim coring methods, continuously collecting HQ (2.5 inch diameter) core. A total 1,078feet of drilling was accomplished through slim core drilling with an average recovery of 92%. Three (3) of the PRC holes were drilled using wire-line large diameter coring methods, continuously collecting PQ (3.4 inch diameter) core. A total 903.5feet of drilling was accomplished through large diameter core drilling with an average recovery of 94%. The remaining 44 drill holes were completed using reverse-circulation (RC) drilling equipped with either a down-hole hammer or deep-hole bit. A total of 15,085feet were completed with RC drilling. 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.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 SAMPLING METHOD AND APPROACH

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.

For PRC's validation drilling program logistics, logging, and initial sample preparation has been managed by NAE following recommendations made by Norwest. NAE has 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 are collected on 5 ft 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 are collected continuously at 5ft intervals. Each 5ft sample weighs approximately 18 to 24lbs.

11.2 SAMPLE PREPARATION, ANALYSES AND SECURITY

ESI determined both the elemental and mineralogical content of a large number of samples. Some of the mineralogy was done by X-ray diffraction. The most critical analytical number for ESI was the Al_2O_3 content of the alunite and was determined by three methods simultaneously, 1) indirectly by measuring the SO_3 content through a LECO furnace determination of the sulfur content, 2) by determining the soluble Al_2O_3 content, presumably by wet chemical methods, and 3) 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, Hazen Research, Skyline

Labs, and NSA. 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.

Slim 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. NAE delivered 760 slim core samples and 2,386 RC samples. This includes 194 blind duplicate samples to evaluate analytical precision.

At the ALS 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 g representative samples
- The 1000 g 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.

PRC selected two analytical packages to use on all samples from the validation drilling program. 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_2O_3, Fe_2O_3, CaO, MgO, K_2O, $Cr2O_3$, TiO_2, MnO, P_2O5, SrO, BaO, 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 ppm concentrations.

For the infill drilling program on Area 2, the ICP-AES whole rock analytical package is being used on all samples from the HQ core drilling and the reverse-circulation drilling. The HQ core samples are also tested for minor and trace elements using the four-acid ICP-AES procedure and will undergo mineral analysis using X-Ray Diffraction (XRD).

Selected core samples from the HQ core holes being drilled in Area 1 will also undergo geotechnical testing. Large diameter core samples (PQ size) have been retained for later metallurgical testing.

12 DATA VERIFICATION

Norwest has conducted four site visits to the Blawn Mountain property. Mr. Kerr and Mr. Henchel visited the Blawn Mountain Alunite Project property on October 30, 2012. Mr. Kerr first performed a site visit on February 9, 2012 and has made several site visits to the property since that time. 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 a few of the major structures that bound the Area 1 deposit. Norwest has maintained an onsite presence throughout the infill drilling program on Area 2 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 Blawn Mountain. PRC has valid mineral control through the Exploration/Option Agreement for 10,394.2 acres.

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. The PRC drill hole locations are 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 100feet (ft) 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	
			12.1 From	12.2 To	12.3 K_2O (%)	2.4 Al_2O_3 (%)	K_2O (%)	Al_2O_3 (%)
PDH-2-42	12.5 B14	12.6 30	12.7 10	12.8 360	12.9 2.67	12.10 10.63	12.11 2.73	12.12 12.66
12.13 PDH2-03	12.14 B9	12.15 100	12.16 Various		12.17 2.76	12.18 12.49	12.19 3.96	12.20 14.64

The samples sent to ALS Minerals from both Area 1 and Area 2 included 194 blind duplicates. Comparison of K_2O and Al_2O_3 concentrations between the original and duplicate samples shows good correlation with mean differences of 4.61% and 0.04% respectively. Figure 12.1 shows the correlation for K_2O and Al_2O_3 for the original and duplicate samples for Area 1 and Figure 12.2 for Area 2.

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 set of 12 sample pulps was forwarded to ACT Labs for comparative analysis (Table 12.3) 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%.

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.

Norwest believes the PRC validation and infill drilling programs for Area 1 and Area 2 respectively have adequately tested the deposit both spatially and in number of twinned drilling locations.

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 results show higher concentrations than previously indicated in the ESI drilling data, the ALS 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

Alunite is a naturally occurring volcanic mineral containing potassium, sulfur, and alumina. The Blawn Mountain Alunite Project will mine alunite, or hydrous potassium aluminum sulfate, KAl3(SO4)2(OH)6, from the Blawn Mountain deposits in Utah. The alunite will be processed for the production of 750,000tpy of fertilizer-grade (92.5%) potassium sulfate (K_2SO_4), also called sulfate of potash (SOP), approximately 1,800,000tpy of co-product concentrated (98wt.%) sulfuric acid (H_2SO_4), and approximately 3,620,000tpy of an alumina/silicate [$(AlO)_2SiO_3$] product consisting of a mixture of alumina, silica, and oxygen containing approximately 50% Al_2O_3. It is anticipated that the stockpiled alumina/silicate product will eventually be re-processed into an alumina (Al_2O_3) product (R. J. Mellon, May 21, 2012)*.

13.1.1 Product Specifications

Table 13.1 summarizes the three main parameters that determine product quality and grades. Those in parentheses include purity, particle size and chloride content.

TABLE 13.1 TYPICAL MARKET PRODUCT GRADES

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

Block Flow Diagrams (BFDs) and description of major unit operations in the proposed methods of processing alunite for the production of sulfate of potash product are given in Section 17.

13.2 HISTORICAL AND RECENT METALLURGICAL TESTING

In 1970, ESI, began to explore for alunite in what it referred to as “the NG Alunite Property” in Blawn Mountain. The primary objective of ESI was to develop its NG Alunite property as a domestic source of alumina.

On April 27, 2011, Potash Ridge Corp. (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, a feasibility study and engineering work performed and/or commissioned by ESI. This historical information on the development of an alumina mine with a SOP by-product is being supplemented and refined by PRC from additional exploration, mine planning, and metallurgical testwork currently underway in support of the Blawn Mountain Alunite Project, References appear in Section 27.

In the early and mid-1970s, ESI commissioned Hazen Research, Inc. (Hazen) to develop and perform an extensive metallurgical testing program on composite samples from the Blawn Mountain deposit. Bench scale testwork was performed on composites of drill core and rotary drill cuttings from the exploration drilling program. Testwork was performed on all aspects of the process including, comminution, beneficiation, flotation, calcination, leaching, crystallization, and solid/liquid separation studies. A large bulk sample was collected from test pits on Blawn Mountain and processed through a pilot plant at the Hazen facilities in Golden, Colorado. The testing program was successful and design criteria were established for design of the full scale process facilities.

In 2011, PRC commissioned Hazen to perform confirmatory testwork on a bulk sample collected from Test Pit No. 5 on 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 this preliminary test program was to confirm the results of the original Hazen testwork. Since that time, modified block flow sheets have been suggested as improvements.

The test program consisted of the following tests:

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

Hazen, in addition to two groups of batch kiln calcining of alunite and subsequent water leaching of calcine experiments, has performed (R. J. Mellon, May 21, 2012) other hydrometallurgical experiments, which included:

- A kinetics study to determine the leach cycle time to maximize the potassium extraction.
- A rudimentary crystallization experiment in which the leachate was heated to 90°C under vacuum to determine the nature and concentration of impurities as crystals are formed.

- A recycle leach experiment for evaluating the buildup of ion concentrations by repeatedly contacting the leachate with fresh calcine.
- A leach to determine the effect of ambient temperature on potassium extraction.
- An experiment on the leached tails to determine the viability of leaching aluminum.
- Settling and filtration experiments to assist selection of solid/liquid separation equipment.

Metallurgical test results from the most recent test program at Hazen (R. J. Mellon, May 21, 2012) and from the historical laboratory program conducted during the months of March through May of 1972 for ESI also by Hazen (F. J. Bowen, et al. April 12, 1973) are presented in the following sections.

PRC also initiated a drilling program under the guidance of North American Exploration Company (NAE). Norwest, as a follow-up to the site visit in February 2012, made additional recommendations (Norwest Corp. April 16, 2012) to the validation drilling program that included the two final reverse circulation holes and adjustments to the sample preparation procedures.

- The drilling program between October 2011 and February 2012 consisted of drilling a combination of 19 holes to a total of 2,062 meters using wire-line coring methods and collecting HQ (63.5 mm diameter) core with an average recovery of 91%, and 15 reverse circulation holes to a total of 2,454 meters to validate the previous exploration efforts of ESI.
- From 1969 through 1974, ESI collected extensive outcrop and trench samples as well as samples from rotary drilling on 3 meter intervals. Laboratory analysis was only performed in some drill holes on some samples at every 9.1 to 15.2 meters or on composite samples.
- During PRC's validation program, NAE collected samples on 3 meter intervals for core holes and on 1.5 meter intervals for reverse circulation holes. Core samples submitted for analyses consisted of 10ft (3 meter) quarter-core sections and each sample weighing approximately 4.5 to 5kg. For reverse circulation drilling, samples were collected on 1.5 meter intervals. The cuttings were passed through a cyclone and then through a Jones Splitter set to a 50/50 split, and each sample weighing approximately 8 to 11kg.

Exploration results pertaining to mineralization of Blawn Mountain Alunite deposits are as follows:

- Alunite mineralization is found on four ridges which are erosional remnants that occur within PRC's exploration tracts.
- Erosion-resistant silica cap forms the tops of peaks. The underlying erosion-resistant quartz- alunite facies (body of rock with specified characteristics) forms the steeper parts of the ridges.

- The silica cap is typically massive but porous material resembling a siliceous sinter.
- On the surface, the quartz- alunite zones are composed of intermixed alunite and silica with minor impurities, mainly iron.

NAE delivered ALS Minerals samples preparation facility in Winnemucca, Nevada, 651 core samples and 735 reverse circulation samples including 59 blind duplicate samples to evaluate analytical precision.

PRC selected two analytical methods which use Inductively Coupled Plasma – Atomic Emissions Spectroscopy (ICP – AES) on all samples from the validation drilling program:

- The first analytical procedure with reporting levels of 0.01% is a whole rock analysis for major oxides using ICP – AES following a lithium metaborate fusion for determining SiO_2, Al_2O_3, Fe_2O_3, CaO, MgO, K_2O, Cr_2O_3, TiO_2, MnO, P_2O_5, SrO, BaO, and loss on ignition (LOI).
- The second analytical procedure for major, minor and trace elements using ICP- AES following four-acid digestion is for determining Al, Ca, Fe, K, Mg, Na, S, and Ti with reporting levels of 0.01%. The reporting levels for Ag, As, Ba, Be, Bi, Cd, Co, Cr, Cu, Ga, La, Mn, Mo, Ni, P, Pb, Sb, Sc, Sr, Th, Ti, U, V, and Zn are in ppm concentrations.
- After receiving the analytical results from ALS Minerals, a set of 12 sample pulps was sent to Activation Laboratories limited (Actlabs) for comparative analysis. Actlabs' 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 exceeds 98%.

13.3 Historical Metallurgical Test Results

13.3.1 Alunite Characterization

In January 2012, Hazen received 4,800 lb of alunite 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. The mineralogical assessment indicated the presence of three major components in the alunite:

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

Additionally, samples of alunite from the Blawn Mountain deposit were analyzed by Hazen were found to contain about 4.6wt.% K and 10.6wt.% Al.

Crushability index (CWi), Bond abrasion index (Ai), and Bond ball mill work index (BWi) tests were also performed to determine the comminution requirements for selection of crushing and grinding equipment.

The 1972 metallurgical studies on Blawn Mountain samples (F. J. Bowen, et al. April 12, 1973) demonstrated:

- Potassium (K) content of alunite: 4.51wt.%.
- Sodium (Na) content of alunite: 0.16wt.%.
- Na_2SO_4:K_2SO_4 ratio: 0.05.
- Water requirement: 3.1 x 10^6tpy.
- To assure minimum product purity of 50wt.% K_2O, the Na_2SO_4:K_2SO_4 ratio should be maintained at 0.08.
- Sodium leaves the process in the form of glaserite, a double salt of potassium and sodium with empirical formula $3K_2SO_4$. Na_2SO_4, which limits the recovery of high-purity product.
- Blending of the run-of-mine (ROM) alunite to meet target Na_2SO_4:K_2SO_4 ratio is a method of minimizing the effects of the variability of ROM feed to the SOP Process Plant.

13.3.2 Sample Preparation

In 2012, Hazen prepared the as received samples of alunite from Pit No. 5 for additional experimental work as follows:

- First, set aside 20 pieces of samples sized 2 by 3in size for comminution testing.
- Stage-crushed the as received material to passing 1 inch, cone-and-quartered the crushed material to obtain three representative lots of 200lb.
- Further stage-crushed each of the 200lb lots to obtain ¾ by ½-ich., minus ½-inch, and minus 6-mesh lots for studying the effect of particle size in the calcination step.

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

13.3.3 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.

Table 13.2 presents the results of tests performed in 2012 by Hazen (D. W. Gillespie, March 1, 2012) on sample from Pit No. 5, which was assigned Hazen number 53021, to evaluate Bond ball mill work index (BWi) and Bond crusher impact work index (CWi) in kilowatt hours per ton (kWh/t), and Bond abrasion work index (Ai) in grams (g).

TABLE 13.2 WORK INDICES BW$_I$, CW$_I$ AND A$_I$ FROM HAZEN TEST RESULTS[*]

Parameter	BW$_i$, kWh/t	CW$_i$, kWh/t	A$_i$, g
HRI Sample # 53021	5.9	7.06	0.2391

*D. W. Gillespie, March 1, 2012.

- Twenty pieces of alunite ranging from 2 to 3 inch set aside were used to determine CW$_i$.
- Two kilograms of the ¾ by ½ inch material saved were used to determine the A$_i$.
- Twelve kilograms of the 6-mesh material was used to determine BW$_i$.

Hazen test results suggest that the Blawn Mountain Alunite is relatively soft and less power will be expended in size reduction operations.

In 1972, Hazen investigated (F. J. Bowen, et al. April 12, 1973) crushing and grinding indices of two samples of 12in size rocks from outcrops representing average alunite (Sample HAZEN-4550-1) and hard, tough alunite (Sample Hazen 4550-2). The results of these tests from a historical perspective were as follows:

- Ore in the natural state is quite hard and tough. The work indices range from 16.2kWh/t to 20.4kWh/t at 270-mesh of grind.
- The work index of the hard alunite was 17.9kWh/t when ground through 100-mesh.
- Roasting the alunite followed by air quench decreased the work index to 12.5kWh/t at 100-mesh.
- Roasting, air quench followed by water quench further weakened the rock, resulting in a decrease in work index to 10.9kWh/t at 100-mesh.
- In the Bond ball mill grindability test, the roasting-quenching technique decreased the work input to 5.5kWh/t, which is equivalent to 6.8 kWh/t savings when calculated on the basis of unroasted alunite.

13.3.4 Screen Analysis

Hazen has performed tests on Composite NGC-101 sample prepared by coning, quartering, and splitting 110 bags (2,750 pounds) of alunite from core holes, which varied in depth from 10ft to 400ft. Assay heads were prepared on the as-received and minus 65-mesh composite. Screen analyses for as-received and crushed portions of Composite NGC-101 alunite are given in Table 13.3.

TABLE 13.3 SCREEN ANALYSES*

% Retained					
	As-Received*		Stage Crushed Through		
Mesh	A	B	8-mesh	28-mesh	65-mesh
+4	-	3.0			
8	-	7.9			
10	15.7	-	5.9		
14	-	11.6	-		
20	13.7	-	17.3		
28	-	13.2	-	0.1	
35	13.5	-	15.0	13.9	
48	-	11.9	-	13.7	
65	10.6	4.3	11.0	8.7	
100	4.1	4.2	4.5	7.6	10.7
150	3.7	3.5	3.8	5.6	11.6
200	3.0	2.9	3.1	4.6	9.7
325	3.1	3.0	3.0	4.4	7.7
-325	32.6	34.5	36.4	41.4	60.3

**Note:* The “as-received” alunite was 33 to 35% minus 325-mesh. (F. J. Bowen, et al. April 12, 1973)

13.3.5 Mineralogical Analysis

Hazen 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), which shows:

- The particle size of alunite varies from coarse-grained to fine-grained aggregates in a matrix of microcrystalline quartz. alunite particles vary in size from 200 x 50 microns to 50 x 10 microns.
- 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 1 mm diameter and filled with opal-like substance with minor amounts of calcite ($CaCO_3$) on microfractures.
- X-ray diffraction 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.

13.3.6 Chemical Analysis of Alunite

Analytical results of 2012 of four hundred grams of the 6-mesh material, pulverized to produce two head samples (Sample A and Sample B), including repeat analyses for Sample B, are given in Table 13.4.

TABLE 13.4 ALUNITE 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

Hazen also determined particle size distribution (PSD) for the 1-inch 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% (100 ppm) 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 13.5 and Table 13.6 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 13.5 XRF Results of Major Constituents by Particle Size Distribution*

Analyte	Analysis, wt.%					
	¾ x ½ in.	½in. 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 13.6 XRF Results of Minor Constituents by Particle Size Distribution*

Analyte	Analysis, wt.%					
	¾ x ½in.	½in. 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.

Table 13.7 summarizes the 1973 chemical analyses of Composite NGC-101 and bulk samples taken from Test Pit C5.

TABLE 13.7 HISTORICAL INTEREST CHEMICAL ANALYSES OF COMPOSITE ALUNITE*

	Assay, wt. %, (Dried at 100°C)							
	K	SO_4**	Fe	Al_2O_3	SiO_2	Ti	Na	P
Historical								
As-received								
Head A	3.65	20.6	1.38	17.07	-	0.45	-	-
Head B	-	20.7	-	-	-	0.45	-	-
Head C	3.60	20.1	1.38	16.85	-	-	-	-
Other	-	-	-	-	51.8	-	0.27	0.05
Minus 65-mesh								
Head I	3.67	20.6	-	-	-	-	-	-
Head II	3.66	20.7	-	-	-	-	-	-
Average	**3.65**	**20.5**	**1.38**	**16.96**	**51.8**	**0.45**	**0.27**	**0.05**
Bulk Samples	**4.51**	**5.85**	-	**-**	**46.2**	-	**0.16**	-

* F. J. Bowen, et al. April 12, 1973.
**Leco analysis and assumes all sulfur reported is in the sulfate form.

Based on the formula $KAl_3(SO_4)_2(OH)_6$, the theoretical potassium analysis should be 4.06% based on the SO_4 content, and the Al_2O_3 content should be 16.31%. The new assay results of bulk samples reported by Hazen indicate that the alunite has higher potassium content than that reported for the historical data**.**

The results of a partial semi-quantitative spectrographic analysis of the Composite NGC-101 alunite are given in Table 13.8.

TABLE 13.8 SPECTROGRAPHIC ANALYSIS OF COMPOSITE NGC-101 ALUNITE*

Element (Symbol)	Atomic Number	%	Element (Symbol)	Atomic Number	%
Copper (Cu)	29	0.001	Strontium (Sr)	38	0.061
Zinc (Zn)	30	0.006	Titanium (Ti)	22	0.11
Iron (Fe)	26	0.92	Zirconium (Zr)	40	0.022
Rubidium (Rb)	37	0.003	Manganese (Mn)	25	0.004
Barium (Ba)	56	0.072	Yttrium (Y)	39	0.002

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

13.3.7 Thermal Analysis

Hazen performed (R. J. Mellon, May 21, 2012) Thermal Gravimetric Analysis (TGA) to understand the behavior of the alunite 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 alunite 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%.

Hazen surmises that:

- 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 Hazen, 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 Hazen, the as-received alunite was found to be moist and became air dried during sample preparation. The moisture content of five 20 lbs. bag samples averaged 0.5% H_2O when dried at 100°C before stage-crushing to 65-mesh.

Table 13.9 summarizes observed weight loss during drying tests.

TABLE 13.9 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^{o}C to 900^{o}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.

13.3.8 Calcination Tests

Hazen has conducted eighteen calcination experiments (R. J. Mellon, May 21, 2012) in a batch quartz kiln system. The kiln, 44 inches in overall length, consisted of a 7 inch diam. by 18 inch long quartz reactor section, four rows of lifters pressed into the sidewall at 90^{o} intervals for mixing the charge, a direct current motor for controlling the rotation of the kiln between 1 and 5 rpm, 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 was placed inside an electrically-heated furnace and the thermocouple positioned into the material.
- From 2 to 4 L of 25% NaOH solution were added to the exit gas impingers (bubblers) to remove the SO_2 that evolved from heating the alunite.
- The heating rate was set at 10 to 15^{o}C per minute to achieve the shortest burden heat up time.
- Air was swept into the kiln at a rate of 0.09 scfm (2.5 L/min), which is equivalent to one changeover of kiln atmosphere every 5 minutes (at standard conditions).
- In the first group of 12 experiments, the alunite was ground to minus ½ inch and the retention time (30, 60, 90, and 120 minutes) and temperature (800, 850, and 900^{o}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 alunite crushed to minus 6-mesh and three on the alunite sized to ¾ by ½ inch to study the effect of alunite particle size on potassium extraction.
- The experiments performed on the alunite crushed to ¾ by ½ inch material were planned to specifically assess potassium extraction from larger particles of alunite.
- Sulfur evolution for the 18 calcining experiments ranged from 0.03 to 0.06lb per /lb alunite. It was determined as the difference between the residual sulfur in the calcine versus the starting quantity in the alunite.

- A continuous online analyzer measured the SO_2 evolved and its concentration was as high as 31% in some tests.

Air was used for purging the kiln and oxygen was present (and measured) in the off-gases from the kiln.

- In all experiments, the O_2 concentration in the exhaust gas consistently increased or decreased as the concentration of SO_2 increased or decreased.

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:

$$SO_3 \Leftrightarrow SO_2 + O_2$$

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 alunite. Gas analyses and thermocouple readings were continuously recorded to a digital file on a computer.

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

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

Vycor®, manufactured by Corning, Inc., is a high-temperature and thermal shock-resistant glass with a low thermal coefficient of expansion. Only high-temperature roasting tests were conducted in the Vycor Retort using 200 and 300 gram 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 alunite 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 alunite at a rate of 5 to 10lbs. per hour along the reactor. The externally heated roasting zone measured about 50in in 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 alunite. 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 Hazen have shown that:

- Relatively coarse alunite can be roasted at about 600^oC 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^oC eliminates the sulfate which is combined with the aluminum in the alunite 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).
- SO_2 and SO_3 in the roaster off-gases are a potential feedstock to the sulfuric acid plant.

13.3.9 Extraction of Potassium from the Alunite

Extraction of potassium was investigated using a two-step approach consisting of calcining the alunite followed by leaching the calcine with water:

- The alunite 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^oF (800 to 900^oC) 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.

Hazen 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 200 g 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.

Extraction of potassium by leaching the calcine with water was as follows:

- The calcine from each experiment was dry-ground in a Buhler mill 100% passing 14-mesh (1.41 mm) so that potassium would be leached in solution and recovered rather than lost during wet grinding.
- Two hundred grams of ground calcine were added to a beaker filled with 800mL of deionized (DI) water to produce the 20% solids for the leach experiments.
- The contents of the beaker were transferred to a 2L leach vessel and the mixture was heated and agitated.
- The leaching was performed at 90°C for 1 hour at 20wt.% solids in water as prescribed in U.S. Patent 4,031,182 (K. W. Loost, July 21, 1977).
- The water vapor was condensed and returned to the leach.
- At the end of the leach, the mixture was filtered, the cake washed with DI water three times, and dried overnight at 60°C.
- In each step, the dispositions of potassium, aluminum, sulfate, titanium, phosphorus, silicon, sodium, iron, and magnesium were monitored analytically in the solids and liquids, with the first three analytes being of greatest interest in the project.

The test conditions and results for three sized samples calcined and leached with water in 2012 are summarized in Table 13.10.

TABLE 13.10 SUMMARY OF CALCINING AND LEACHING CONDITIONS AND RESULTS*

Conditions	¾ by ½in.	Minus ½in.	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.

When evaluating the water leach test results, both potassium and aluminum extractions must be taken into consideration. An inspection of test results in Table 13.10 shows:

- Calcine produced at 850°C resulted in one of the highest potassium extractions of 87% and aluminum extraction of 1.3%.
- Calcining the alunite 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 alunite 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.

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 50wt.% and 17wt.% 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% 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.

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 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.

Table 13.11 summarizes the 1973 test results showing percent potassium extraction as a function of calcining temperature and residence time in the calciner.

.

TABLE 13.11 PERCENT POTASSIUM EXTRACTION AS A FUNCTION OF CALCINING TEMPERATURE AND RESIDENCE TIME*

Hazen Calcining Test Results		
Temperature, °C	**Residence Time, minutes**	**Potassium (K) Extraction, %**
850	60	74

The 1973 roast/leach test results are summarized in Table 13.12.

TABLE 13.12 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.

The 1973 historical test results from experiments summarized in Table 13.12 compare favorably with those from the 2012 test results summarized in Table 13.10.

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 13.13.

- 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 13.13 INFLUENCE OF LEACHING TIME ON ALUMINA AND SULFATE DISSOLUTION*

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

Table 13.14 and Table 13.15 summarize the results of leaching tests at Hazen establishing the influence of leaching parameters on percent extraction of potassium.

TABLE 13.14 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 90 100	60 60 60	50 50 17	65.2 80.5 74

TABLE 13.15 INFLUENCE OF LEACHING TIME ON WATER LEACHING OF HIGH TEMPERATURE (800OC) CALCINE*

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

13.3.10 Effect of pH of Leaching Medium

Hazen selected minus 8-mesh calcine (F. J. Bowen, et al. April 12, 1973) produced in the 6 inch Screw Reactor during high temperature Test SR-13 to study the effect of pH of the lixiviant (leaching medium). Table 13.16 summarizes the characteristics of the calcine produced during high temperature Test SR-13.

TABLE 13.16 CHARACTERISTICS OF THE CALCINE FROM TEST SR-13*

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

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

Three parallel leaching tests were performed by Hazen at 85°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 7 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.

13.3.11 Off-Gas Analyses

During the 1973 high temperature roasting tests using the Vycor Retort, nitrogen was used for purging at a rate of 250mL/min. Table 13.17 summarizes the average composition of off-gases evolved at various roasting temperature during the 1.0 hour roasting periods.

Orsat analyses were performed on off-gases evolved during several runs in the 6-inch Screw Reactor in the 1973 roasting tests. During these runs, 0.35 or 0.6ft^3/min of air was used for purging. Table 13.18 summarizes the test results.

TABLE 13.17 OFF-GAS ANALYSES FROM VYCOR RETORT ROASTING TESTS*

Test No.	Roasting Temperature, °C	Average % Analysis		
		SO_2	SO_3	O_2
T-7	800	18.7	1.2	8.8
T-8	850	15.2	1.3	6.6
T-9	900	12.6	1.0	6.1

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

TABLE 13.18 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 alunite = 20.5%.

13.3.12 Sulfate in Potassium Water Leach Residues

Hazen performed in 1973 high (≈800°C) and low (≈600°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 6-inch 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.

13.3.13 Crystallization Testwork

Hazen observed in the water-leach tests (Mellon, Robert J. May 21, 2012) the concentration of potassium in the leach liquor was 12.8 g/L at 5 minutes and 12.4 g/L at the end of 60 minutes leach cycle, whereas the respective aluminum concentration declined from 0.5 g/L to 0.1 g/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 downstream K_2SO_4 crystallizer, thus enhancing the quality of the SOP product.

Hazen 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 25wt.% 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 160 mg 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 1 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, 0.025wt.% Fe, and less than 0.0025wt.% Mg.

Leaching the calcines at higher solids content and/or recycling the leach solution to build up 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.

13.3.14 Recycling Leach Liquor for Enriched Extraction

Hazen conducted an experiment in 2012 to determine whether the concentration of potassium ion (and impurities) in the leachate increases (Mellon, Robert J. May 21, 2012) by re-contacting the leachate produced from leaching a calcine with fresh calcine.

- Four cycles of leachate recycling were conducted on a portion of the calcine generated at 800°C and 90 minutes. residence time.
- During each leach cycle, the mechanically-agitated leach conditions were identical, with temperature controlled at 90°C and pulp density at 20wt.% solids.
- An aliquot of the leach liquor was analyzed for potassium, aluminum, titanium, phosphorus, silicon, sodium, iron, magnesium, and sulfate.

Table 13.19 summarizes the progression of enrichment of potassium and aluminum in the leachate from cycle to cycle and the ratio of potassium to aluminum as an indicator of increases in aluminum concentration as a function of the cycle.

- It was observed that after the fourth cycle, potassium and sulfate concentrations increased from the initial leach by a factor of 3- and 3.5-fold, respectively.

TABLE 13.19 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.

- Based on the mass ratio of potassium to aluminum, aluminum was not increasing with respect to potassium over the four cycles.

13.3.15 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.

13.3.16 Vacuum Filtration and Kynch Settling Tests

Hazen performed in 2012 vacuum filtration and Kynch settling experiments on slurry obtained by water leaching calcines produced at 850°C and 30 min 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 808 (kg/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 13.20.

TABLE 13.20 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.

Hazen used in the Kynch experiments two doses of Hychem AF-303-HH flocculant and Table 13.21 summarizes the Settling test results.

TABLE 13.21 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 12 mg/kg of solids proved sufficient in clearing the supernatant of turbidity.

13.3.17 Flotation of Alunite

The alunite evaluated by Hazen 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. Flotation tests are planned for the recovery of alunite concentrate with low levels of quartz, which could potentially reduce the capital and operating costs, result in energy savings in the calcining step as well as reduced equipment size in calcining and leaching operations.

In 1976, Alumet investigated the flotation of alunite from the alunite 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 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 13wt.% 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.

Table 13.22 summarizes the composition of two alunite samples and one water leach residue from roasted alunite employed in flotation tests.

TABLE 13.22 COMPOSITION OF FLOTATION TEST SAMPLES*

Sample:		Alunite (Lot 1)	Water Leach Residue	Alunite (Lot 2)
Quantity		5 gal.	5 gal.	1 Drum
Size		minus 20-mesh	minus 20-mesh	minus 20-mesh
Analyses:				
Potassium	%K	3.78	0.66	4.20
Sodium	%Na	0.22	0.06	0.21
Alumina	%Al_2O_3	17.0	20.2	18.2
Sulfur	%S	6.98	1.22	7.58
Silica	%SiO_2	51.86	31.1	48.2
Alunite	%	45	-	49

*R. Myertons, May 27, 1976.

The effect of grind size on recovery of alunite in rougher flotation was investigated in two tests after the alunite was ground wet in a rod or ball mill, followed by thickening and conditioning with 2lb/t fatty acid collector. The results from these tests are summarized in Table 13.23.

TABLE 13.23 EFFECT OF GRINDING ON ROUGHER FLOTATION*

Test No.	Grind Size		Total Rougher Concentrate		
	P_{80} microns	% -400-mesh	% Weight	% Alunite	% Recovery
4A	233	14	47	66	70
5A	120	26	57	64	83
5D	110	33	60	61	83
4C	100	28	58	62	83
4D	97	31	58	62	84
4B	83	35	60	60	87
5E	58	68	60	60	87

*R. Myertons, May 27, 1976.

- The initial test at 14% minus 400-mesh (P_{80} = 233 µm) yielded approximately 70% recovery due to incomplete liberation.
- All of the tests at finer grind resulted in 83 to 87% recovery with 60 to 64% alunite grade.
- A test at 68% minus 400-mesh yielded the same recovery as a test at 35% minus 400-mesh.
- These tests indicate that there is no need to grind the samples finer than 80% passing 83 µm.
- Refined oleic acid (Darling Red Oil, RO) and a neutral mineral oil (vapor oil, VO) were used as collectors for alunite in the rougher flotation.

Table 13.24 summarizes the effect of collector on alunite recovery.

TABLE 13.24 EFFECT OF COLLECTOR ON ALUNITE RECOVERY*

Test No.	Red Oil, lb/ton	Vapor Oil, lb/ton	VO:RO Ratio	Total Collector, lb/ton	Alunite Recover, %
4B	2	2	1	4	87
4D	2	2	1	4	83
5A	4	4	1	8	83
5B	6	6	1	12	95
5D	4	8	2	12	83
5C	2	10	5	12	74

*R. Myertons, May 27, 1976.

In 1972, at the request of ESI (F. J. Bowen, et al. April 12, 1973) Hazen investigated flotation of alunite from mined material to obtain a concentrate with 80% alunite at 80% recovery as the minimum target objective.

- Concentrate up to 85% alunite was produced using fatty acid in open circuit flotation.
- Concentrates ranging from 89 to 90% alunite were produced, however, only at 35 to 50% recovery.

Table 13.25 summarizes the open circuit flotation test results.

TABLE 13.25 OPEN CIRCUIT ALUNITE FLOTATION TEST RESULTS*

% Alunite	% Recovery
80	83
81.5	82
83.5	80
85	78
88	45

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

Alunite flotation tests have also been performed to investigate the possibility of floating alunite from the silica and associated iron-bearing gangue minerals.

- Fatty acid flotation was found to be effective, the test results indicating that 85% alunite concentrate may be achieved at 78% recovery, which could increase to 80 to 82% recovery in closed circuit operation.
- Maximum of 90% alunite concentrate grade was achieved, but only at 40 to 50% recovery.
- The alunite is liberated at a size finer than 270-mesh grind, although inclusions of silica within alunite occur at ≈10 µm size.
- Rougher flotation at a coarse size followed by regrinding and cleaner flotation yields better grade and higher recovery than flotation of finely ground alunite without regrinding.
- The best results were obtained in a test (Test 3N) which utilized fatty acid and mineral oil collectors, hot conditioning, rougher flotation at minus 100-mesh grind followed by three stages of regrinding and eight stages of cleaner flotation.
- Over 10% additional recovery was obtained using three stage additions of red oil (fatty acid collector) to the scavenger circuit.

Several alternative collector systems for alunite flotation were investigated (F. J. Bowen, et al. April 12, 1973) as substitutes for red oil. These included Actinol 2122, a crude mixture of fatty acids from tall oil; FA-1, a partially-refined tall oil (also called "liquid rosin"), vapor oil, a neutral partly-refined mineral oil; and Aero 801, used with mineral oil to increase agglomeration and collection of fine alunite. A comparison of such systems is given in Table 13.26.

TABLE 13.26 ALTERNATIVE COLLECTORS FOR ALUNITE FLOTATION*

Test No.	Collector		Rougher Grind, Mesh	Rougher + Scavenger Concentrate			Cleaned Conc. % Alunite
	lb/ton	Reagent		% Weight	% Alunite	% Recovery	
3B	0.8	Red Oil	270	73.6	54.7	93.3	90.5
3H	0.2 0.3	Actinol 2122 Mineral Oil	270	26.7	79.2	47.1	-
3M	2.2	90% Actinol 2122 + 10%DETA**	270	63.7	61.2	89.0	82.1
3I	0.5	FA-1	270	46.7	69.0	71.8	82.6
2A	1.2 2.5	Aero 801 Mineral Oil	270	57.1	62.8	80.8	80.7
2B	1.3 1.0	Aero 801 Mineral Oil	270	38.6	68.5	59.4	70.0

* F. J. Bowen, et al. April 12, 1973; **DETA = Diethylenetriamine

Vapor oil, a neutral and partly refined mineral oil, used as an auxiliary collector in flotation was found to improve the recovery of alunite. Table 13.27 summarizes the effect of vapor oil on alunite flotation.

TABLE 13.27 EFFECT OF VAPOR OIL ON ALUNITE FLOTATION*

Test No.	Collector, lb/ton			Rougher Concentrate		% Recovery at	
	Vapor Oil	Red Oil	Aero-825	% Alunite	% Recovery	80% Alunite	85% Alunite
3D	-	0.75	0.05	59.8	90.4	72	65
3L	-	1.0	0.05	59.3	89.1	76	69
3J	0.24	1.15	0.05	61.4	90.9	81	72
3N	0.24	1.75**	-	56.2	91.9	83	78

* F. J. Bowen, et al. April 12, 1973; ** Staged to multiple regrind steps

- The effect of pH on alunite flotation was evaluated by maintaining the pH in two tests at 8.9 and 8.3 levels with soda ash. There was a strong indication in favor of pH 8.3 with respect to grade and recovery.

To improve selectivity and recovery, hot conditioning of pulp is often used in fatty acid flotation. Table 13.28 compares the metallurgical test results at 90, 65, and 35°C, which indicates some benefit on the order of 5 to 10% in recovery. Further investigations are required to determine the cost of energy expended to the value of incremental recovery.

TABLE 13.28 EFFECT OF TEMPERATURE IN ROUGHER CIRCUIT CONDITIONING*

Test No.	Temperature, Rougher Conditioner	Mesh of Grind	Rougher Concentrate	
			% Alunite	% Recovery
3B	90°C	270	54.7	93.3
3D	65°C	100	59.8	90.4
3C	35°C	270	62.2	85.0

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

13.3.18 Reverse Flotation of Silica from Alunite

Variations of a technique employed at Cleveland-Cliffs Tilden Concentrator for selectively dispersing and decanting silica from flocculated minerals in taconite ores was tested for removing silica (R. Myertons, May 27, 1976) in the cleaner alunite concentrate as well as on ROM alunite

- Sodium silicate as a dispersant, however, was found to be equally effective in dispersing both alunite and quartz.
- Similar tests with tetrasodium pyrophosphate (TSPP) as dispersant also yielded the same result.

Point of Zero Charge (PZC) is the pH value at which the electrical charge density on the surface of a mineral particle submerged in an electrolyte is zero. Silica, which forms silicic acid in water, yields an acidic PZC at pH 2, while magnesia (MgO), which forms magnesium hydroxide, yields a basic PZC at pH 12.

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.

13.4 CONCLUSIONS AND RECOMMENDATIONS

13.4.1 Conclusions

Based on a review of technical reports and results of an extensive metallurgical testing program commissioned by both ESI and PRC at Hazen, on composites of drill core and rotary drill cuttings from the Blawn Mountain deposit exploration drilling program on all aspects of the comminution, beneficiation, flotation, calcination, leaching, crystallization, and solid/liquid separation studies, the conclusions are as follows:

- Annual production rate = 750,000tpy SOP product
- Required product quality = 92.5% K_2SO_4

- Pit recovery = 98%
- Alunite grade in feed = 28.41%
- K_2SO_4 grade in feed = 5.98%
- Flotation concentrate = 83%
- Leach recovery = 85%
- Alunite concentrate grade = 80%
- Quality specifications for typical market-grade Standard SOP product are:
  - Purity:
    50wt.% K_2O (92.5wt.% K_2SO_4)
  - Particle size:
    70 to 10 Tyler mesh
  - Chloride content
    < 1.0%
- Byproduct sulfuric acid:
  - Production rate:
    250tph (6,000 tpd)
  - Product purity:
    98% H_2SO_4 \
- Potential co-product:
  - Alumina (Al_2O_3) from processing stockpiled alumina/silicate

13.4.2 Recommendations

Recommendations on additional metallurgical test work and trade-off studies required for optimized flowsheet 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 alunite, which have a direct bearing on product grade and recovery.
- Conduct tests on dry ROM alunite samples to obtain both crushing and grinding Bond Work Index (kWh/t) measurements and develop quantitative data on liberation of alunite as a function of grain size screen fractions.
- Perform re-grinding and flotation tests on rougher flotation concentrate and scavenger circuit tails to determine the relationship between grinding time, liberation of locked-in alunite /quartz, and size distribution in the re-ground product, as well as percent recovery and grade of the flotation concentrate recovered.
- Estimate grinding media consumption and the amount and composition of fines generated during crushing and grinding which can be sent directly to leaching.
- Perform bench-scale flotation tests to evaluate the effect of particle size on flotation efficiency, selection and dosage of reagents, conditioning and residence time, reverse

flotation of silica from alunite, and effect of recycling process effluents with residual reagents on flotation recovery and reagent consumption.

- Conduct sedimentation and filtration tests to establish type of flocculant and dosage in flotation concentrate and tails slurry solids thickening and filtration unit operations.
- Perform drying/calcining 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 x-ray diffraction 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 x-ray diffraction and microscopic examination.
- Conduct bench-scale 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.

14 MINERAL RESOURCE ESTIMATES

Four potential mine development targets have been identified within the PRC 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 respectively along with surface mapping. 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 403 drill holes including 34 PRC twin validation holes in Area 1 and 50 PRC infill holes in Area 2 have been completed. The geologic models are built from the assays and records of 222 drill holes. As discussed in Section 10, 147 of the drill holes were not used in the geologic database. A majority of these holes were air track holes. The air track drill was often used to prospect for bedrock under alluvium or to spot rotary holes. There are insufficient records for these holes to include them in the geologic model.

A number of criteria were established for determination of resources:

1. The validation process indicated from recent laboratory results that constituent values were as much as 20% higher than historical results for Area 1. It was therefore decided to utilize only the historical results for Area 1. This established more conservative analytical values but also ensured that a larger number of control points of common program origins would be incorporated into the quality characterization of the deposit. By adopting this approach 60 holes were used to establish mineral grades for Area 1.
2. 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 (Figure 14.1) were also prepared and showed that there were no significant nugget effects or directionality to the data that would require more robust kriging approaches.
3. Analytical results were based on composites developed over 10ft intervals in each hole.
4. Four lithologic domains are represented in the geologic block model: Alunite, Clay, Dolomite, and Silica.
5. 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.

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	1,423,2000	1,432,000	20
Northing	583,540	595,500	20
Elevation	6,000	7,700	20

6. 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.
7. 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 1000ft for Area 1 and 2000ft for Area 2. The larger search readii for the Area 2 model was used to account for the more widely spaced drilling.

TABLE 14.3 CLASSIFICATION CRITERIA

	Measured	Indicated	Inferred
K_2O	<150ft	<350ft	<1,000ft*
Al_2O_3	<150ft	< 250ft	<1,000ft*

8. 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 5 m (16.4ft) resolution. 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 and Al_2O_3 sample data. Table 14.3 outlines the distance from drill hole samples used to classify resource blocks within the alunite zone.
9. The assumed density of alunite and waste was established at 13ft^3/short ton (2.076923 short tons/yd^3) as derived from estimates used previously by ESI (1974). Norwest believes that this bulk density factor is reasonable for this deposit type.
10. 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.

11. Both visual and calculated validation of model block values to posted drill assay values show strong correlation.

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 14.2, 14.3, and 14.4 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 14.5 identifies the classified resources 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 156Mt of material carrying an average grade of 3.22% K_2O and 13.90% Al_2O_3. The calculated potassium sulfate grade (K_2SO_4) at a 1% K_2O cut-off grade is 5.96%. 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 81Mt. Average grade at a 3% K_2O cut-off is 4.20% K_2O and 15.35% Al_2O_3 with a calculated equivalent grade of 7.77% K_2SO_4. Approximately 66% of the identified resources are classified as measured and 34% as indicated resource.

TABLE 14.4 CLASSIFIED RESOURCE ESTIMATE FOR THE AREA 1 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 (%)	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	105,670,000	3.18	5.88	13.94	27.95	37.74	3,359,000	6,212,000	14,730,000	29,531,000	39,878,000
	1	103,838,000	3.22	5.96	13.90	28.32	37.63	3,345,000	6,185,000	14,434,000	29,403,000	39,074,000
	2	84,454,000	3.59	6.64	14.23	31.54	38.52	3,030,000	5,604,000	12,017,000	26,638,000	32,532,000
	3	53,707,000	4.21	7.78	15.34	36.98	41.53	2,259,000	4,178,000	8,238,000	19,863,000	22,302,000
	4	26,949,000	4.93	9.11	17.18	43.30	46.52	1,327,000	2,454,000	4,631,000	11,668,000	12,537,000
	5	10,157,000	5.76	10.64	20.03	50.59	54.23	585,000	1,081,000	2,035,000	5,139,000	5,508,000
INDICATED	0	53,306,000	3.19	5.90	13.98	28.03	37.85	1,699,000	3,143,000	7,453,000	14,939,000	20,176,000
	1	52,447,000	3.23	5.97	13.91	28.37	37.66	1,692,000	3,130,000	7,296,000	14,878,000	19,751,000
	2	42,897,000	3.58	6.61	14.25	31.44	38.59	1,534,000	2,837,000	6,114,000	13,485,000	16,552,000
	3	27,450,000	4.20	7.76	15.36	36.89	41.59	1,152,000	2,130,000	4,217,000	10,125,000	11,417,000
	4	14,351,000	4.86	8.99	16.85	42.72	45.62	697,000	1,290,000	2,419,000	6,130,000	6,548,000
	5	4,443,000	5.85	10.81	20.69	51.40	56.02	260,000	480,000	919,000	2,284,000	2,489,000
MEASURED AND INDICATED	0	158,977,000	3.18	5.88	13.95	27.97	37.78	5,059,000	9,355,000	22,183,000	44,471,000	60,054,000
	1	156,285,000	3.22	5.96	13.90	28.33	37.64	5,037,000	9,315,000	21,729,000	44,281,000	58,826,000
	2	127,350,000	3.58	6.63	14.24	31.51	38.54	4,564,000	8,441,000	18,131,000	40,124,000	49,084,000
	3	81,156,000	4.20	7.77	15.35	36.95	41.55	3,411,000	6,308,000	12,455,000	29,988,000	33,719,000
	4	41,300,000	4.90	9.07	17.07	43.09	46.21	2,025,000	3,744,000	7,050,000	17,798,000	19,084,000
	5	14,600,000	5.78	10.69	20.23	50.84	54.77	844,000	1,561,000	2,954,000	7,422,000	7,997,000
INFERRED	0	392,000	3.29	6.08	17.17	28.92	46.47	13,000	24,000	67,000	113,000	182,000
	1	392,000	3.29	6.08	17.17	28.92	46.47	13,000	24,000	67,000	113,000	182,000
	2	329,000	3.58	6.62	17.96	31.48	48.62	12,000	22,000	59,000	104,000	160,000
	3	175,000	4.54	8.39	22.43	39.87	60.71	8,000	15,000	39,000	70,000	107,000
	4	92,000	5.64	10.42	29.23	49.55	79.12	5,000	10,000	27,000	46,000	73,000
	5	76,000	5.90	10.91	30.28	51.88	81.96	4,000	8,000	23,000	40,000	62,000

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 14.6, 14.7, and 14.8 show cross sections through the block model for Area 2. Figure 14.9 identifies the classified resources for the Area 2 Blawn Mountain property.

At a 1% cut-off grade, there is a combined measured plus indicated resource of 464Mt of material carrying an average grade of 3.07% K_2O and 13.16% Al_2O_3. The calculated potassium sulfate grade (K_2SO_4) at a 1% K_2O cut-off grade is 5.68%. 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 260Mt. Average grade at a 3% K_2O cut-off is 3.74% K_2O and 13.95% Al_2O_3 with a calculated equivalent grade of 6.92% K_2SO_4. Approximately 15% of the identified resources are classified as measured, 50% as indicated resource and 35% as inferred resource.

TABLE 14.5 CLASSIFIED RESOURCE ESTIMATE FOR THE AREA 2 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 (%)	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	110,164,000	3.05	5.64	13.02	26.79	35.26	3,358,000	6,210,000	14,347,000	29,518,000	38,839,000
	1	106,224,000	3.14	5.81	13.24	27.61	35.84	3,336,000	6,170,000	14,064,000	29,331,000	38,074,000
	2	92,222,000	3.37	6.24	13.49	29.66	36.52	3,112,000	5,754,000	12,441,000	27,354,000	33,680,000
	3	62,137,000	3.77	6.96	13.99	33.10	37.86	2,339,000	4,326,000	8,690,000	20,566,000	23,527,000
	4	19,232,000	4.50	8.33	15.36	39.59	41.59	866,000	1,602,000	2,954,000	7,615,000	7,998,000
	5	2,605,000	5.49	10.15	17.64	48.24	47.77	143,000	264,000	460,000	1,257,000	1,244,000
INDICATED	0	368,380,000	2.99	5.52	13.01	26.24	35.21	10,996,000	20,335,000	47,908,000	96,667,000	129,696,000
	1	358,218,000	3.05	5.65	13.14	26.84	35.58	10,936,000	20,225,000	47,073,000	96,142,000	127,437,000
	2	301,900,000	3.32	6.15	13.40	29.21	36.28	10,032,000	18,552,000	40,464,000	88,192,000	109,543,000
	3	197,469,000	3.73	6.90	13.94	32.79	37.72	7,366,000	13,621,000	27,517,000	64,751,000	74,495,000
	4	58,500,000	4.47	8.26	15.25	39.25	41.27	2,612,000	4,830,000	8,919,000	22,962,000	24,145,000
	5	6,417,000	5.46	10.09	17.89	47.96	48.43	350,000	647,000	1,148,000	3,078,000	3,107,000
MEASURED AND INDICATED	0	478,545,000	3.00	5.55	13.01	26.37	35.22	14,354,000	26,545,000	62,255,000	126,186,000	168,535,000
	1	464,442,000	3.07	5.68	13.16	27.02	35.64	14,273,000	26,395,000	61,137,000	125,473,000	165,511,000
	2	394,122,000	3.33	6.17	13.42	29.32	36.34	13,144,000	24,307,000	52,904,000	115,546,000	143,222,000
	3	259,606,000	3.74	6.91	13.95	32.86	37.76	9,705,000	17,947,000	36,208,000	85,317,000	98,021,000
	4	77,733,000	4.47	8.27	15.27	39.34	41.35	3,478,000	6,432,000	11,873,000	30,577,000	32,143,000
	5	9,022,000	5.47	10.11	17.82	48.04	48.24	493,000	912,000	1,607,000	4,335,000	4,352,000
INFERRED	0	259,034,000	2.84	5.25	12.77	24.96	34.57	7,354,000	13,600,000	33,079,000	64,649,000	89,550,000
	1	250,769,000	2.91	5.37	12.83	25.55	34.73	7,287,000	13,476,000	32,166,000	64,063,000	87,080,000
	2	210,440,000	3.15	5.83	13.12	27.69	35.53	6,629,000	12,259,000	27,616,000	58,274,000	74,762,000
	3	123,406,000	3.63	6.71	13.80	31.88	37.36	4,475,000	8,275,000	17,029,000	39,337,000	46,100,000
	4	31,293,000	4.43	8.19	15.19	38.95	41.11	1,387,000	2,564,000	4,752,000	12,190,000	12,865,000
	5	3,013,000	5.35	9.89	18.36	47.02	49.69	161,000	298,000	553,000	1,417,000	1,497,000

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

There are no mineral reserve estimates associated with this report.

.

16 MINING METHODS

Mining operations at Blawn Mountain will utilize conventional truck/shovel mining techniques to remove alunite and waste material from mining areas. The alunite deposit at Blawn Mountain is unique in that very little waste material will be encountered during mining. Due to this, the mining areas developed for this PEA were defined mainly by the geologic resources and extent of the alunite body as well as the potassium oxide grade. The targeted mining production levels are defined by the amount of K_2SO_4 required. On a steady-state basis, K_2SO_4 requirements are 750,000 tons per year (tpy) which equates to roughly 17.2Mt of ROM alunite. In some years, ROM production requirements are higher or lower depending on the grade of K_2O being released from mining operations. Mine quantities were generated using the geologic model described in previous sections of this report.

16.1 GENERAL MINING METHOD

Future operations at Blawn Mountain will begin mining in Area 1 and transition to Area 2 once the targeted alunite has been removed from Area 1. Figure 16.1 details the locations of Area 1 and 2. alunite and waste will generally be removed using area and bench mining techniques utilizing multiple 20ft vertical lifts. Additionally, multiple mining faces will be used to assist with in-pit alunite blending efforts. A conventional truck/shovel method will be used employing a large hydraulic backhoe as well as a front-end-loader to load end-dump mining trucks. Prior to alunite and waste removal, the material must be drilled and blasted.

16.2 AREA PREPARATION

Prior to removing alunite and waste, all growth-media material that is present within the mining areas will be removed. This material will be salvaged and hauled to a temporary storage pile outside the mining area. Once required for reclamation purposes, the growth-media material will be loaded into mining trucks and hauled to final placement.

16.3 MINING OPERATIONS

Two areas (Area 1 and Area 2) will be developed to mine alunite at Blawn Mountain. At this stage of development, the anticipated mine life is 30 years, with a two year ramp-up period. This first two years of mining (Year -1 and Year 1) are considered a ramp-up period with full production occurring in Year 2. The material removed during Year -1 is considered pre-production and is not considered part of the 30 year mine life. alunite material removed in Year -1 will be stockpiled near the crusher site as the processing facilities are not ready to accept and process alunite at this stage. During Year 2, alunite produced from the mine as well as the stockpiled material from Year -1 will be sent to the processing facilities. As previously mentioned, minimal amounts of waste material are encountered during mining. The strip ratio over the mine life averages approximately 0.12:1 (cubic yards of waste: ROM alunite tons). Waste material consists primarily of overburden and alunite that is not of sufficient quality to process. Table 16.1 illustrates waste volumes, ROM alunite tons, ROM alunite qualities and stripping ratios associated with each mining area. Additionally, ROM alunite tons are further delineated into the assurance categories of measured, indicated and inferred. A mining recovery of 98% was utilized and reflects that minimal alunite loss is anticipated as the majority of the mining will occur in the alunite zone.

TABLE 16.1 MINE QUANTITIES & QUALITIES

Area	*Alunite (Mt)	Al_2O_3 (%)	K_2O (%)	NA_2O (%)	Waste (Myd^3)	Strip Ratio (yd^3/ton)	*Alunite (Mt) Measured	*Alunite (Mt) Indicated	*Alunite (Mt) Inferred
1	121.6	13.77	3.40	0.51	8.1	0.07	81.1	40.4	0.07
2	387.9	13.14	3.22	0.33	52.7	0.14	63.7	202.8	121.4
Total	**509.5**	**13.30**	**3.26**	**0.37**	**60.8**	**0.12**	**144.8**	**243.2**	**121.5**

*ROM

As alunite is mined, it will be transported to either the stockpile area or directly to the primary crusher. Figure 16.1 shows the stockpile and crusher locations. Waste that is generated from mining will initially be placed in out-of-pit piles. As mining progresses and sufficient area becomes available, it is anticipated that waste material will be placed back into prior excavations. In a general sense, the majority of waste from Area 1 will be placed out-of-pit and waste from Area 2 will be hauled and placed into the Area 1 mine void.

16.4 MINING PARAMETERS

Mining at Blawn Mountain will be performed using typical truck/shovel mining techniques. A large percentage of the alunite mined from both Area 1 and 2 will occur on the ridges and hilltops at elevations higher that the surrounding drainages. It is anticipated that benched pits will be established and material removed in 20ft lifts. As sufficient area become available, multiple mining faces will be utilized to facilitate in-pit blending to achieve desired K_2O grades. As mining progresses deeper into the deposit and excavations move sub-surface, pit slopes of 45 degrees will be utilized for safe and productive operations.

16.5 MINE SCHEDULE

The Mine schedule is summarized in Table 16.2. As previously discussed, mining initiates in Area 1 and when this area is depleted of targeted alunite, operations will shift to Area 2. Full Production is anticipated in Year 2.

TABLE 16.2 MINE SCHEDULE

Year	-1	1	2	3	4	5	6-10	11-15	16-20	21-30
Waste (Myd^3)	0.4	0.7	0.9	1.2	1.2	1.2	3.6	14.4	14.0	23.2
Alunite (Mt)	4.5	9.2	15.1	17.5	17.9	17.6	83.6	86.3	86.0	172.0
Al_2O_3 (%)	14.28	14.77	14.10	12.99	12.88	13.07	13.56	12.99	13.18	13.29
K_2O (%)	4.12	4.02	3.67	3.17	3.10	3.14	3.31	3.21	3.22	3.22
NA_2O (%)	0.20	0.34	0.42	0.47	0.49	0.54	0.42	0.30	0.36	0.35

16.6 MINING EQUIPMENT

Two spreads of mining equipment are envisioned to remove the required amount of alunite and waste volumes. Table 16.3 illustrates the type, size and quantity of equipment used. The quantities presented are at maximum levels. The quantity of trucks varies throughout the mine life.

TABLE 16.3 MINING EQUIPMENT

Primary Equipment			
Hitachi EX3600	Backhoe	28yd^3	1
CAT 992K	Front-End-Loader	16yd^3	1
CAT 785C	Truck	148 ton	12
Support Equipment			
CAT 785C H_2O	Water Truck	16,000 gallons	1
CAT 16M	Grader	297Hp	2
CAT D10T	Dozer	580Hp	2
AC DM45	Drill	45,000Lb	1

The equipment spreads detailed above have the capacity to remove all the waste and alunite required for the mine plan. In addition to the above equipment, several other pieces of smaller support equipment will be required such as pumps, light plants, lube and fuel trucks, pick-up trucks, mechanics trucks, etc.

16.6.1 Geotechnical Consideration

As this property is in the initial stages of study, a detailed geotechnical investigation of mining operations has not been completed to date. Norwest has utilized conservative pit design parameters in this evaluation. Bench heights of 20ft and overall highwall slope angles of 45 degrees have been utilized.

16.7 HYDROLOGICAL CONSIDERATIONS

At this stage of study, no detailed hydrological assessment has been performed for Blawn Mountain.

16.8 WATER MANAGEMENT

As this report is a PEA, no detailed designs or plans have been developed to date regarding water management. It is envisioned that the majority of surface water at the site will be controlled with a series of adequately sized channels that will divert water away before it comes in contact with any mine disturbance. Water that does come in contact with mine disturbance will be collect in appropriately sized channels, sumps and ponds and allowed time for sediment to settle. This captured water will likely be used for mine haulroad watering and also utilized for the processing of the alunite. At this stage of study, groundwater is not considered to be significant and should not be a hindrance to mining.

16.9 WORKFORCE

The production requirements detailed above necessitate a 7 day per week schedule operating approximately 340 days per year. The remaining idle days are to account for holidays and other unplanned events such as inclement weather. The manpower estimate assumes hourly staff will work a 4 days on and 4 days off schedule. Table 16.4 details the manpower requirements at full production levels.

TABLE 16.4 MANPOWER SUMMARY AT FULL PRODUCTION

Category	Employees per Crew	Number of Crews	Subtotal
Hourly Workers			
Excavator Operators	1	4	4
Truck Drivers	8	4	32
Dozer Operators	2	4	8
Grader Operators	2	4	8
Drill Operators	1	4	4
Blasters	1	2	2
Front-End-Loader Operators	1	4	4
Maintenance Labor	4	4	16
Sub-Total Hourly	20		78
Management			
Exec., Staff, Tech			10
Maintenance			6
Operations			6
Sub-Total Management			22
Total Surface Mine Employees			**100**

17 RECOVERY METHODS

17.1 OVERVIEW OF ALUNITE PROCESSING

Alunite is a semi-soluble salt mineral which occurs as a hydrothermal-alteration product in feldspathic (orthoclase feldspar-rich) igneous rocks. The Potash Ridge Alunite Project will mine alunite, or hydrous potassium aluminum sulfate, $KAl_3(SO_4)_2(OH)_6$, from the Blawn Mountain deposits in Utah.

The alunite will be processed for the production of an estimated 750,000tpy of fertilizer-grade (92.5%) potassium sulfate (K_2SO_4), also called sulfate of potash (SOP), approximately 5,000 short tons per day (tpd) or 1,800,000tpy of concentrated (98wt.%) sulfuric acid (H_2SO_4) as a co-product, and approximately 9,920 short tpd or 3,620,000tpy of an alumina/silicate [$(AlO)_2SiO_3$] co-product consisting of a mixture of alumina, silica, and oxygen containing approximately 50% Al_2O_3. It is anticipated that the stockpiled alumina/silicate co-product will eventually be re-processed into an alumina (Al_2O_3) product.

17.1.1 Process Development in Historical Perspective

In 1970, ESI, began to explore for alunite in what it referred to as “the NG Alunite Property” in Blawn Mountain, the primary objective of which was to develop its property as a domestic source of alumina. During the early and mid-1970s, ESI commissioned Hazen Research, Inc. (Hazen) to develop and perform an extensive pilot-scale metallurgical testing program on composite samples from the Blawn mountain deposit.

On April 27, 2011, Potash Ridge Corp. (PRC) acquired from a third party historical information pertaining to the NG Alunite property, including data on drilling results, resource estimates, pilot plant testwork, mine plan, a feasibility study and engineering work performed by ESI.

In 2011, PRC commissioned Hazen to perform confirmatory testwork on a bulk sample collected from Test Pit No. 5, located near the center of the envisioned starter pit for the mine. The objective of this preliminary test program was to confirm the results of the original Hazen testwork. This historical information on the development of an alumina mine with SOP as by-product is being supplemented and refined by PRC during 2012 from additional exploration and mine planning by Norwest and metallurgical testwork at Hazen in support of the Blawn Mountain Alunite Project.

The proposed SOP and by-product sulfuric acid recovery methods discussed below are based on information obtained from both historical and recent exploration and process development activities. Figure 17.1 is a block flow diagram (BFD) which illustrates the major unit operations.

Metallurgical test work is being planned to investigate flotation as a method of recovering alunite from the mined material as concentrate with concomitant reduction in the silica content of the feed to the drying and calcining steps. This would also result in enhanced product quality, conservation of energy, and smaller equipment sizes in downstream drying/calcining, leaching, and solid/liquid separation operations as well as reduced footprint of the process plant.

17.1.2 Major Unit Operations

The recovery methods for extracting potassium as sulfate of potash from the alunite, as envisioned, consists of the following major unit operations:

- Crushing and grinding of run-of-mine alunite.
- Conditioning with flotation reagents the finely-ground alunite pulped with water.
- Multi-stage flotation to recover alunite concentrate and tailing as slurry.
- Thickening and clarification of alunite concentrate and tailings slurry.
- Filtration of concentrate thickener underflow to obtain alunite filter cake.
- Pumping tailings to impoundment.
- Reuse of thickener overflow and tailings pond water for water conservation.
- Drying and Calcining alunite concentrate obtained as filter cake.
- Recovery of alunite -bearing dust in calciner off-gases.
- Routing sulfur oxides-rich calciner off-gases as feed to sulfuric acid plant.
- Production of concentrated sulfuric acid as by-product.
- Leaching calcines with hot water to extract SOP.
- Filtration of water-leached slurry to recover SOP-rich solution.
- Stockpiling alumina/silicate-rich filter cake for alumina recovery at a later date.
- Evaporation-Crystallization of SOP crystals from SOP-rich solution.
- Compacting for densification of SOP product crystals.
- Packaging, storage and load out of product SOP.
- Bleed stream treatment for controlling impurity buildup.

The process plant for recovery of sulfate of potash is divided into the following areas:

- **AREA 100.** Primary Crushing.
- **AREA 200.** Ore Stockpile and Reclaim.
- **AREA 300.** Wet Grinding and Classification.
- **AREA 400.** Flotation and Flotation Concentrate Solid/Liquid Separation.
- **AREA 500.** Flotation Concentrate and Tailings Dewatering.
- **AREA 600.** Concentrate Drying and Calcination.
- **AREA 700.** Calcine Leaching and Solid/Liquid Separation.

- **AREA 800.** Crystallization and SOP Product Solid/Liquid Separation.
- **AREA 900.** Product Drying, Compaction, Storage and Load Out.
- **AREA 1000.** Powerhouse Services.
- **AREA 1100.** Sulfuric Acid Plant.
- **AREA 1200.** Water Treatment Plant.

Major unit operations or process steps (Drawing No. BFD-001 in Appendix A) in the proposed methods of processing alunite for the production of SOP product, recommendations on additional test work for process optimization as well as for assuring product quality, and trade-off studies required for equipment selection for the commercial plant are described below.

17.1.3 Product Specifications

Table 17.1 summarizes the three main parameters of purity, particle size and chloride content that determine product quality and grades.

TABLE 17.1 TYPICAL MARKET PRODUCT GRADES

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

17.2 PRIMARY CRUSHING

The Primary Crushing circuit consists of a Primary Gyratory Crusher, the Truck Dump Hopper, the discharge Apron Feeder and the Primary Crusher Conveyor which will deliver the 6 inch product to a Cross Country Conveyor and to the Coarse Alunite Stockpile.

17.2.1 Gyratory Crusher

Approximately 2,535stph (short tons per hour) of ROM alunite from the mine will be delivered to the Primary Gyratory Crusher dump hopper. A rail like grizzly feeder screen above the hopper will allow the minus 24 inch rock to pass into the Gyratory. A rock breaker located beside truck dump will break up the larger boulders (i.e. +24 inches) and allow them to pass through the grizzly. An Apron Feeder will deliver the crushed alunite to the Primary Crusher Conveyor and then to a Cross Country Conveyor estimated to be about two miles long. It is estimated that the

Gyratory Crusher size reduction will be approximately 4:1 that is, the crushed rock will be minus six inches.

17.2.2 Coarse Alunite Stockpile

The minus 6 inch alunite from the Cross Country Conveyor will discharge into a splitter chute. The alunite will normally be transferred to a Coarse Alunite Stockpile Tripper Conveyor. The Coarse Alunite Stockpile will be conical in shape and will contain up to 100,000 tons of alunite for a two-day supply for primary grinding. Five Reclaim Apron Feeders (four operating and one on standby) will deliver the alunite to a Stockpile Transfer Conveyor and then to the Semi-Autogenous Grinding (SAG) Mill Feed Conveyor. Alternatively, the splitter chute will be able to bypass the Coarse Alunite Stockpile and route the Coarse Alunite to the SAG Mill Conveyor. Currently on hold and being considered for a future/optional alternative is the Secondary Cone Crusher, briefly described below. The Coarse Alunite is conveyed to the Secondary Crusher Grizzly Feeder.

17.3 SECONDARY CRUSHING - OPTIONAL

A Secondary Cone Crusher was considered early in the development of the major unit operations and the mass balance. Since the current Primary Gyratory Crusher can crush the alunite ROM alunite to minus six inches and the SAG Mill will accept this size, it was decided to consider the Secondary Cone Crusher as a future option. Stockpile reclaimed alunite is delivered to the SAG Mill Conveyor. In the event the tonnage is increased or the ROM size exceeds 6 inch, the Secondary Cone Crusher will be evaluated and considered a viable option.

17.3.1 Semi-Autogenous Grinding

The SAG Mill is the primary or first stage of grinding. The SAG Mill Feed Conveyor delivers the crushed stockpile alunite to the feed chute of the SAG Mill. The alunite feed rate is 2,535dstph. In addition, sufficient process water is added at the feed chute to provide a 65% by weight solids density for the grind cycle. A grinding ball charge is maintained at between 10% and 12% by volume in the SAG Mill. The SAG mill discharge slurry flows over a trommel screen, which is attached to the mill discharge end. The trommel undersize material flows to the Grinding Mill sump, while the trommel oversize material, which consists of steel ball chips and rejected critical sized pebble, will discharge into a bin or onto a reject pebble conveyor and transferred to a rejected pebble stockpile.

In the event that the critical sized pebble material becomes excessive and begins to load up the mill, pebble ports can be installed in the mill discharge grates and an optional pebble crushing circuit can be installed. The pebble crushing circuit would consist of mill discharge screens, conveyors and a cone crusher to reduce the pebble size and then recycle the crushed material to the mill feed chute. The steel ball chips would be removed by a continuous belt magnet.

17.3.2 Pebble Milling - Optional

The Pebble Milling circuit was considered early in the process development to further crush the SAG Mill Trommel oversize and recycle it back to the SAG Mill. The Trommel oversize typically would discharge to a vibrating screen where most of the SAG Mill discharge (the undersize) would drop to the Grinding Mill Sump. This oversize discharges to a Pebble Crusher Feeder Belt and is conveyed to cone type crusher called the Pebble Mill. The crushing reduction ratio is estimated to be about 3:1. The crushed product is conveyed to the SAG Mill feed chute. It is conceivable that the SAG Mill oversize will be considerably less than 10% and a Pebble Mill circuit may not be necessary. For now, this circuit is a future option.

17.3.3 Ball Mill Grinding

Secondary grinding begins when slurry from the Grinding Mill Sump is pumped to the Grinding Cyclone Clusters. As envisioned, the discharges from the SAG Mill and the two Ball Mills cascade into a single 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 slurry retention time of 3 to 5 minutes.

Two large Cyclone Feed Pumps deliver slurry to each respective Cyclone Cluster. Sufficient pressure is maintained in the Cyclone header (i.e., about 10 to 12psig) to begin separation of the solid particle sizing process.

The Cyclone Underflow contains the majority of the coarse particles – usually plus 100 mesh Tyler and is thickened to about 70% by weight solids and reports to the respective Ball Mill. Also, approximately 75% of the total slurry flow to the Cyclone Clusters reports to the Underflow. The recirculating flow through the Ball Milling circuit could be between 300% and 400%. This means that the average particle is recycled three to four times through the milling circuit before being fine enough as the final product – Cyclone Overflow.

Each Ball Mill contains a ball charge of approximately 35% by volume. The ground product from each Ball Mill is discharged from the Trommel to the Grinding Mill Sump.

The Cyclone Overflow contains the majority of the fine particles – approximately 80% passing minus 100 to 150-mesh Tyler. The finer particles tend to spiral upward in the high velocity centrifugal flow as does the liquid part of the slurry. As a result, the slurry density decreases and the volumetric flow of the finer particles increases compared to the underflow. Cyclone Overflow reports to the Flotation Conditioner Tank for a surge supply to the Rougher Flotation circuit.

17.4 FLOTATION

The Flotation process as presently configured will include Rougher, Cleaner and Scavenger circuits. In addition, a Regrinding Ball Mill circuit is included to reduce the Rougher Concentrate to a target grind of 80% passing 270-mesh Tyler. Each flotation circuit will consist of its respective concentrate and tailing transfer sumps, tanks and pumps. The final concentrate reports to the Concentrate Thickener and the tailings to the Tailings Thickener. A description of each circuit is provided below.

17.4.1 Rougher Flotation

Grinding Mill Cyclone overflow is pumped to an agitated Conditioning Tank. This tank is sized to hold approximately two hours slurry feed for Rougher Flotation. Flotation reagents, are metered into the Conditioning Tank ahead of flotation, preparing for the concentrate and tailings separation downstream. The Rougher Flotation Feed Pump delivers the conditioned slurry to the Rougher Feed Distribution Tank, which in turn delivers slurry to the Rougher Flotation Cells. The preliminary estimate slurry retention time is 10 minutes for each cell or approximately 60 minutes for the Rougher Flotation train.

The Rougher Concentrate from the cells is channeled through peripheral launders in each cell and collected in Launders. The Launders discharge the Rougher Concentrate slurry to a sump and is pumped to the agitated Rougher Concentrate Tank. The Rougher Concentrate Tank has at least two hours storage capacity for feed to the Regrind Milling circuit. A preliminary estimate is that about 35% of the total feed to Rougher Flotation will report as Rougher Concentrate and the remainder is tailings.

Rougher Tailings discharge from the final Rougher Flotation Cell reports to the Rougher Tailings Sump and is pumped to an agitated Rougher Tailings Tank. This tank has retention of 30 minutes of tailings slurry. Finally, the Rougher Tailings is pumped to the Tailings Thickener and subsequent disposal to Tailings Impoundment.

17.4.2 Regrind Ball Mill Circuit

It is estimated that approximately 34% of the feed or about 908stph solids report to Rougher Flotation Concentrate. Primary Ball Mills will reduce the particle size to approximately 80% passing 100 to 150 mesh Tyler. Unless the grinding tests prove otherwise, a regrind ball milling circuit will be necessary to produce the required 80% passing 270 mesh Tyler for liberation of alunite and separation of the silica which reports to Tailings.

The Rougher Flotation Concentrate is pumped from the Rougher Concentrate Tank to the Regrind Mill Sump. The Regrind Mill circuit configuration is similar to that of the Primary Ball Mill/Cyclone circuit. Using a preliminary 250% recirculating load (i.e., the Cyclone Underflow

flow at 2.5 times the Cyclone Overflow), the Regrind Mill should be able to reduce the particle size at least 3:1 and achieve the 80% passing 270 mesh Tyler product. The Cyclone Overflow reports to Cleaner Flotation.

17.4.3 Scavenger Flotation

Cleaner Tailings discharge by gravity to the Scavenger Feed Distribution Tank. Flotation reagents are metered into the Distribution Tank ahead of flotation as required. The Scavenger Feed Distribution Tank delivers slurry to the first of four Scavenger Flotation Cells. The preliminary estimate slurry retention time is 10 minutes for each cell or approximately 40 minutes for the Scavenger Flotation train.

Scavenger Concentrate of alunite is channeled through peripheral launders in each cell and collected in Launders. The Launders discharge the Scavenger Concentrate slurry to a sump and is pumped to the agitated Scavenger Concentrate Tank. The Scavenger Concentrate Tank has two hours storage capacity for the feed to the Cleaner Concentrate Thickener circuit. A preliminary estimate is that about 25% to 30% of the total feed to Cleaner Flotation will report as Cleaner Concentrate and the remainder is tailings.

Scavenger Tailings discharge from the final Scavenger Flotation Cell to the Scavenger Tailings Sump and is pumped to the Scavenger Tailings Tank with a retention time of 30 minutes. Finally, the Scavenger Tailings is pumped to the Tailings Thickener and subsequent disposal to Tailings Impoundment.

17.4.4 Cleaner Flotation

Reground Rougher Concentrate is pumped to the Cleaner Feed Distribution Tank. Flotation reagents are metered into the Distribution Tank ahead of flotation as required. The Cleaner Feed Distribution Tank delivers slurry to the first of four Cleaner Flotation Cells. The retention time is estimated to be 10 minutes for each cell or approximately 40 minutes for the Cleaner Flotation train.

The Cleaner Concentrate is channeled through peripheral launders in each cell and collected in Launders. The Launders discharge the Cleaner Concentrate slurry to a sump and is pumped to the agitated Cleaner Concentrate Tank. The Cleaner Concentrate Tank has at least two hours storage capacity for the feed to the Cleaner Concentrate Thickener circuit. A preliminary estimate is that about 25% to 30% of the total feed to Cleaner Flotation will report as Cleaner Concentrate and the remainder is tailings.

Cleaner Tailings discharge from the fourth Cleaner Flotation Cell discharges by gravity to the Scavenger Feed Distribution Tank. Flotation reagents, Amine and Flotation Oil, are metered into the Distribution Tank to facilitate the concentrate and tailings separation in the Scavenger Flotation Cells.

17.5 THICKENING AND DEWATERING

Flotation Concentrates and Tailings are separated in the Flotation circuits and pumped to their respective Thickeners. In particular, Cleaner Concentrate reports to the Concentrate Thickener and Tailings from the Rougher and Scavenger Flotation report to the Tailings Thickener. Thickener Cleaner Concentrate Underflow is pumped to a Concentrate Filter Press for dewatering and the Thickener Tailings Underflow is pumped to Tailings Impoundment.

17.5.1 Flotation Concentrate Thickening

Cleaner Flotation Concentrate is collected and pumped to the Cleaner Concentrate Tank and the concentrate is in turn pumped to the Concentrate Thickener. Flocculent is metered into the Thickener feed well launder with the Concentrate slurry to condition solids to settle. The Thickener rake plows the precipitated solids to the center discharge well. A preliminary estimate is that the Concentrate will collect and thicken to about 50% by weight solids and the Concentrate Thickener Underflow Pump will deliver this slurry to a Concentrate Filter press.

17.5.2 Thickened Concentrate Filtration

The Concentrate Thickener Underflow is pumped to the Concentrate Filter Press where the slurry is dewatered to approximately 90% by weight solids.

17.5.3 Flotation Tailings Thickening

Tailings from Rougher and Scavenger Flotation enter the Tailings Thickener feedwell. A flocculent is metered in with the slurry to condition the solids to settle. The Thickener rake plows the precipitated solids to the center discharge well. A preliminary estimate is that the Tailings will collect and thicken to about 60% by weight solids. The Tailings Thickener Underflow Pump will deliver this tails slurry to Tailings Impoundment.

17.6 RECLAIMED WATER MANAGEMENT

Fresh water is required for the alunite recovery circuits. It is critical to recover as much of the water used as possible in the mill circuits, both to conserve water as well as to minimize the cost of pumping and delivering fresh water. Reclaim water is generally recovered from the Tailings Impoundment, Thickener Overflow, Filter Press Filtrate, or from any process involving solids and liquid (water) separation.

17.6.1 Concentrate Thickener Overflow

When the Flotation Concentrate slurry and flocculent are mixed in the Thickener feedwell, flocculation begins and virtually all solids drop to the Thickener bottom and are raked to the center discharge well. If the flocculation is successful the water overflowing the Thickener peripheral launder will contain less than about 250 parts per million (ppm) solids. This Overflow is collected and pumped to the Reclaim Water Tank for storage and subsequently to the reclaim or plant water header for use in the plant process. In the event that the Overflow is murky or “muddy” indicating high solids, a Clarifier unit may be required to further remove solids. It is important to note that since this Overflow water has residual flotation reagents, flocculants and other chemicals that it is not combined with or added to domestic water supplies.

17.6.2 Tailings Thickener Overflow

Similar to the Concentrate Thickener Overflow, the Tailings Thickener Overflow is pumped to the Reclaim Water Tank for storage and to the plant water header. In the event that the Overflow is murky or “muddy” indicating high solids, a Clarifier unit may be required to further remove the suspended solids. It is important to note that since this Overflow water has residual flotation reagents, flocculants and possible other chemicals added that it not commingled with or added to domestic water supplies.

17.6.3 Filtrate from Concentrate Filter Press

Filtrate from the Concentrate Filter Press is generally collected in a sump and would be pumped to the Reclaim Water Tank.

17.7 FILTRATION OF ALUNITE CONCENTRATE AND LEACH RESIDUES

The alunite -rich concentrate from the cleaner flotation circuit is thickened and the thickener underflow at about 65wt.% solids is filtered to obtain filter cake at about 90wt.% solids as feed to the drying/calcining operations.

The slurry from the water leach circuit is filtered to recover SOP in filtrate as feed to the evaporator/crystallizer circuit. The alumina/silicate filter cake is stockpiled as a potential alumina resource for reprocessing at a future date.

Filter cake may be produced by pressure, or applying vacuum, in batch or continuous mode operation. The list of major types of cake filters includes the plate and frame press, the horizontal leaf, or tray filter, the continuous vacuum filter of the rotary-drum type, rotary-disc filter, and the horizontal vacuum belt filter.

For tonnages involved, the horizontal vacuum belt filter (HVBF), which applies a continuous vacuum on a horizontal plane is a likely filter press of choice. The thickener underflow slurry is fed onto a filter cloth supported by a traveling drainage belt. The HVBF is adaptable to applications where low cake moisture is desirable, can wash the filter cake in either co-current or countercurrent mode, and can provide for compression and/or thermal drying features to recover drier solids.

17.8 DRYING AND CALCINING

Hazen performed (R. J. Mellon, May 21, 2012) Thermal Gravimetric Analysis (TGA) by heating the alunite to 1000°C at a rate of 10°C/min and held at 1000°C. The results indicated that:

- Initially, the weight loss observed was 5.6% of the sample, which commenced at 505°C.
- The weight loss slowed between 560 and 720°C, and the total loss was 14%.
- An additional 8.3% weight loss began at 735°C and continued to 780°C, which is associated with the generation of sulfur oxides. This was confirmed through the SO_2 analyzer during the batch kiln runs.
- The final weight loss at the end of analysis was 25%.

The investigations at Hazen (Mellon, Robert J. May 21, 2012) also demonstrated that the calcination reaction at 800 to 900°C (1472 to 1652°F) is as follows:

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

Hazen has conducted 18 calcination experiments (R. J. Mellon, May 21, 2012) on alunite that was ground to minus ½ inch and crushed to ¾ by ½ inch and by varying the residence time (30, 60, 90, and 120 minutes) and temperature (800, 850, and 900°C) in the kiln to study the effect of process variables on potassium extraction. It was determined that:

- Sulfur evolution for the 18 calcining experiments ranged from 0.03 to 0.06 lb. per lb. alunite.
- The concentration of SO_2 as measured with a continuous online analyzer was as high as 31% in some tests.

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 reaction:

$$SO_3 \Leftrightarrow SO_2 + O_2$$

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

- Relatively coarse alunite 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 alunite 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).
- SO_2 and SO_3 in the roaster off-gases are a potential feedstock to the sulfuric acid plant.

17.8.1 Drying and Calcining Equipment

A number of manufacturers supply a variety of drying/calcining equipment:

- Alstom Power, Bartlett-Snow Thermal Products (Rotary Kilns)
- BSP Thermal Systems, Inc. (Multiple Hearth Furnace)
- Carrier Vibrating Equipment, Inc. (Vibrating Fluid Bed Dryer)
- FEECO International, Inc. (Rotary Kilns)
- GEA Barr-Rosin (Fluid Bed Dryer)
- Hankin Environmental Systems, Inc. (Multiple Hearth Furnace)
- Harper International (Rotary Kilns)
- Industrial Furnace Company (Multiple Hearth Furnace)
- Outotec (Fluidized Bed)
- Wyssmont Company, Inc. (Turbo-Dryer)

It is envisioned that rotary kilns, or a fluidized beds, or multiple hearth furnaces would be candidate technology for drying and calcining the alunite flotation concentrate. A major consideration is the concentration of sulfur dioxide generated in the furnace off-gases. Typically, it is recommended that the feedstock to the modern day double contact sulfuric acid plant contain from 10 to 12% SO_2 by volume. When the feedstock off-gases contain less than 10% by volume SO_2, a sulfur burner system to maintain the desired concentration of SO_2 is required.

Furnace manufacturers require pilot-scale testing of the feed to the drying/calcining system before scale-up of the commercial unit as well as to determine the composition of the flue gases.

A brief description of types of dryers and calciners and their operating characteristics is given below.

Dryers remove unbound moisture from raw materials. The dryer types used in mineral processing industry include direct-fired rotary, indirect-fired rotary, fluid bed, flash, spray, and vibrating-grate. Dryers use either convection (direct) or conduction (indirect) of heat as the method of drying. Combustion products are in direct contact with the material in the convection-type dryers whereas heat is transmitted indirectly by contact of the wet material with a heated surface (refractory bricks) in the conduction-type dryers. The thermal efficiency of direct-fired dryers is higher than the thermal efficiency of indirect-fired dryers.

Calciners bring about thermal decomposition, phase changes, or removal of volatile compounds in the presence of air and operating at temperatures below the melting point of the product materials. The types of calciners used in mineral processing are rotary, flash, fluid bed, and multiple hearth furnaces.

- The Rotary Calciner is used when removal of both combined and uncombined (surface) moisture from the material is required. A rotary calciner is a refractory brick-lined, rotating cylindrical steel shell, ranging in length from 10 to 30 times its diameter. The wet-feed or pre-dried material is fed at one end and the calcined product is discharged at the other end.
- The material moves through the dryer because of the combined effects of the slope of the shell to the horizontal and the action of lifting flights within the shell as the shell rotates. Radiation from the refractory to the material bed is the primary source of heat transfer in rotary calciners. The production rates range from 1 to 100t/hr. and the residence time in these units is from 15 minutes to 15 hours.
- The Flash Calciner is designed to dry as well as calcine the material and convey it by a stream of hot gases from the feed point to some other point of delivery. Typically, it is a cylindrical vessel with a conical bottom and lined with refractory bricks. A bank of cyclones or a bag house is used for separation of the calcined product from the carrier gas stream.
- The Fluid Bed Calciners vary from a vertical cylinder to oblong and rectangular units. A vertically rising, hot stream of gas is introduced through a gas distributor plate at the base of a bed of particulate solids. The velocity of the hot gas stream through the bed is usually between 0.5 and 10ft/sec. which expands the wet feed bed to allow the particles to move within the bed, i.e., the bed becomes fluidized and may resemble a boiling liquid. The size of the particles is between 10 and 210µm. From a mixing point, the fluidized bed is a continuously stirred reactor with a predetermined residence time.
- The Multiple Hearth Furnace consists of a refractory brick-lined steel shell with a number of hearths mounted one above the other. Rabble arms on each hearth are driven from a common center drive shaft. A screw feeder discharges the material into the furnace at the center of the upper hearth. Rabble arms agitate and move the charge to the periphery of the hearth where it falls to the next hearth. The material is then moved to the center of this second hearth from which it falls to the next hearth, and the cycle continues down the furnace. Gas burners may be mounted at any of the hearths and the air circulated internally to cool the hollow shaft is used for combustion. Feed rate and retention time can be varied.

Each dryer/calciner and evaporator/crystallizer equipment manufacturer has developed theoretical models and specialist scale-up know-how. Pilot plant investigations are required to configure the equipment in terms of operating parameter for a particular application, either as single or multistage units, and based on heat transfer calculations and the equipment manufacturers' past experience with similar applications.

Test results from calcining flotation concentrate will be used in the selection of the most cost-effective and energy-efficient calciner. Composition of off-gases the from the calciner, including concentration of SO_2 and dust loading, will be the basis for selecting the gas cleaning equipment and the sulfuric acid plant.

17.9 WATER LEACHING CALCINES

Horizontal belt filters maximize cake purity at minimum specific cost. Our filters are especially suited for applications requiring low cake moisture and multi-stage washing. Horizontal belt filters maximize cake purity at minimum specific cost. Our filters are especially suited for applications requiring low cake moisture and multi-stage washing. Leaching the calcine with water consists of dissolving potassium and sulfate values in the feed (the solid phase) using an aqueous solution. The leachate or the potassium sulfate-rich solution is subsequently processed for the recovery of SOP in the Crystallizer/Evaporator system. The operating conditions of the water leach circuit are as prescribed in U.S. Patent 4,031,182 (K. W. Loost, July 21, 1977):

- Water is the leaching medium (lixiviant).
- Leaching at 90^{o}C
- Residence time 1 hour.
- Pulp density of leach slurry at 20wt.% solids

During leaching, the mineral of value dissolves only partially and another solid product forms. The diffusion of the solvent (water) to the solid-solution interface and the transport of the product species through the reaction product layer to the bulk of the solution are two rate-determining steps in all leaching reactions. If diffusion is rate-limiting, vigorous agitation of the solution can speed up the leaching.

Additionally, the pH of the solution, temperature, pressure, particle size, leaching time, and the chemical properties of the leaching medium (lixiviant) also influence the rate of dissolution of the solute. The rate of dissolution (in kg/s) is related to the solubility product and the surface area of the material.

The solubility product constant, or simply the *solubility product* (K_{sp}) of a compound is the product of the molar concentrations (mol/L) of the constituent ions, each raised to the power of its stoichiometric coefficient in the equilibrium equation. The K_{sp} is the equilibrium constant for the equilibrium between an ionic solid and its saturated solution. For potassium sulfate:

$$K_2SO_4\,(s) \Leftrightarrow 2K^+(aq) + SO_4^{2-}(aq) \qquad K_{sp} = [K^+]^2[SO_4^{2-}]$$

Solubility of potassium sulfate is 111 g/L at 20°C; 240 g/L at 100°C. The solubility of sodium sulfate is 161 g/L at 20^{o}C and 430 g/L at 100^{o}C. Soluble salts have very large K_{sp} values. The common ion effect *decreases* the solubility of a slightly soluble ionic compound. When both Na_2SO_4 and K_2SO_4 are present in the solution, the concentration of the common ion, SO_4^{2-}, increases, and some of it combines with K^+ ion to form solid K_2SO_4, thereby lowering the amount of dissolved K_2SO_4. Concentration of Na_2SO_4 and the common ion effect, if any, on potassium solubility in the water leach circuit should be investigated.

17.10 EVAPORATION AND CRYSTALLIZATION

Equipment manufacturers have recommended a “quadruple effect” evaporator and the draft-tube-baffle (DTB) evaporator-crystallizer system for the production of SOP crystals.

Evaporation involves supplying the latent heat of vaporization to the solution to vaporize the solvent. Typically, the normal heating medium is low pressure steam. By adopting methods for recovery of heat from the vapor, it is possible to achieve great economy in heat utilization.

The boiling point corresponds to the temperature at which the vapor pressure of the liquid equals the surrounding atmospheric pressure. The boiling point of the liquid is lowered as the surrounding pressure is decreased. Operation under a vacuum increases the temperature difference between the steam and the boiling liquid and therefore tends to increase the heat flux. At the same time, the reduced boiling point will usually result in a more viscous material and a lower film heat transfer coefficient.

In the proposed “quadruple effect” evaporator with forward feed arrangement, neglecting the heat required for heating the feed from its initial temperature to its boiling point, the heat transferred appears as latent heat in the vapor entering the second effect and is used as steam in the second effect. The liquor passing from the first evaporator into the next carries heat into the second effect, and this is responsible for some evaporation; similarly for the third effect and the fourth effect.

Therefore, 1kg of steam fed to the No. 1 effect (vessel) evaporates approximately 1kg of water in #1 vessel. Again the 1kg of steam from #1 evaporates approximately 1kg of water in No. 2 and so on. Thus, in a system of N effects, 1kg of steam fed to the first effect will evaporate a total of approximately N kg of liquid. The economy of the system, measured in kilograms of water vaporized per kilogram of steam condensed, increases with the number of effects.

- The value of the multiple effect system is the higher efficiency in the use of steam. However, to achieve this, it will be necessary to make a bigger capital outlay for the increased number of units and accessories.
- One important component of any evaporator installation is the equipment for condensing the vapor leaving the last effect of a multiple-effect unit This is achieved either by direct contact with a jet of water, or in a normal tubular exchanger.

Nucleation and crystal growth are two essential and sequential steps in crystallization, which determine the number and size of crystals present in the crystallizer at the end of the operation. Supersaturation is achieved by evaporation or evaporative cooling under vacuum.

Mixing has an effect on the properties of the product, such as crystal size distribution, purity, and morphology. The physical properties of the crystals formed will influence the choice of downstream processes such as centrifugation, filtration, drying, solids handling, and compaction of product.

The size of crystals formed can be regulated if the number of nuclei can be controlled. For a solution, the solubility of a substance and the supersaturation of solution in terms of temperature and of concentration are important parameters in the design of crystallizers.

The selection of a crystallizer system for a particular task will depend upon the feed material available, the properties of the system and the product requirements of the customer.

The sequence in the design of a typical crystallizer involves:

- Basic data collection by bench-scale or pilot tests.
- Selection of supersaturation generation method.
- Choice of batch or continuous operation.
- Choice of specific equipment type.
- Full scale design.

The DTB evaporator-crystallizer is a forced-circulation crystallizer of the “mixed suspension mixed product removal” (MSMPR) design. It consists of a slow-moving propeller surrounded by a draft tube within the body which maintains a suspension of the product. Slurry which has been cooled is returned to the bottom of the vessel and recirculated through the propeller. The heated solution is mixed with the recirculated slurry at the propeller. Coarse crystals are separated by gravity (sedimentation) in the baffled area (settling zone) that surrounds the crystallizer body. The heating element supplies the heat to evaporate water and raise the temperature of the solution removed from the settler.

Typically low-pressure (30 to 40 psi absolute) exhaust steam from turbines is adequate as heating medium for evaporator crystallizers and is preferred if crystal deposition on heating surfaces is to be avoided. However, if the crystallizer has to be operated under low absolute pressures, a vacuum augmenter or booster may be required. This equipment requires steam at not less than 120 psi gage pressure.

It is conceivable that sodium sulfate (Na_2SO_4) is also crystallized as the binary salt glaserite [$(K_2SO_4)_3.Na_2SO_4$] thus affecting the quality of the SOP product. A potential method of minimizing the amount of glaserite formation is to establish a bleed stream and treat it with potassium chloride to convert Na_2SO_4 into sodium chloride and recover an additional amount of K_2SO_4. The two-step reactions, which should be investigated by bench-scale experiments, are as follows:

$$3KCl + 2Na_2SO_4 \rightarrow K_3Na(SO_4)_2 + 3NaCl$$

$$K_3Na(SO_4)_2 + KCl \rightarrow 2K_2SO_4 + NaCl$$

Membrane separation of K_2SO_4 and NaCl in solution followed by recycling the SOP-rich solution to the evaporator/crystallizer circuit should also be investigated.

17.11 PRODUCT COMPACTION AND LOAD OUT

Evaporator/Crystallizer equipment manufacturers recommend that the product discharged from the crystallizer would require a centrifuge followed by drying and compaction to produce the desired size-range (granular) marketable SOP product.

SAHUT-CONREUR (SC), affiliated with FEECO International, is a recognized designer and manufacturer of briquetting, compaction, granulation, and grinding plants using double roll presses. The largest compactor that SC (or any other manufacturer) has on the market (and in use worldwide by the potash industry) consists of 1050mm dia. x 1000mm wide (41in. dia. x 39in. wide) rolls. The rolls are powered by 900kW (1200hp) drive and complete with force feeder assembly and hydraulic assembly for applying pressure to the rolls. The throughput capacity of the machine is approximately from 100 to 120tph.

17.12 SULFURIC ACID PLANT

The investigations at Hazen (Mellon, Robert J. May 21, 2012) also demonstrated that the calcination reaction at 800 to 900°C (1472 to 1652°F) is as follows:

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

Hazen has conducted 18 calcination experiments (R. J. Mellon, May 21, 2012) on alunite that was ground to minus ½ inch and crushed to ¾ by ½ inch and by varying the residence time (30, 60, 90, and 120 minutes) and temperature (800, 850, and 900°C) in the kiln to study the effect of process variables on potassium extraction. It was determined that:

- Sulfur evolution for the 18 calcining experiments ranged from 0.03 to 0.06 lb. per lb. alunite.
- The concentration of SO_2 as measured with a continuous online analyzer was as high as 31% by volume percent in some tests.

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:

$$SO_3 \Leftrightarrow SO_2 + O_2$$

The list of suppliers of turnkey sulfuric acid plants includes:

- MECS, Inc. (formerly Monsanto Company and currently owned by Du Pont).
- Simon Carves Engineering, Ltd.
- Outotec North America.

The turnkey sulfuric acid manufacturing plant with proprietary technology for sulfur burning plants includes:

- Catalyst
- Drying Tower mesh pad
- Interpass and Final Tower Mist Eliminators
- Strong Acid Coolers
- Drying Tower, Interpass Tower, and Final Tower Acid distributors
- Gas to Gas Heat Exchangers.

Contact sulfuric acid plants either use furnace off-gases containing SO_2 or burn elemental sulfur as the raw material to produce SO_2 and/or to enrich the furnace off-gases as feedstock to the acid plant. The SO_2-bearing feedstock off-gases contaminated with dust and water vapor passes through a train of gas cleaning steps that cool and remove the entrained dust. The gas is then passed through electrostatic precipitators to remove acid mist and any remaining dust particles. The wet gas is dried in a tower by contacting with 93% sulfuric acid. When the concentration of SO_2 is less than approximately 10 to 12% by volume in the dust-free and dried furnaces off-gases, a sulfur burner system is required. A blower forces the dried gases through a series of heat exchangers and converter beds.

Vanadium pentoxide catalyst promotes oxidation of SO_2 to SO_3. In the normal converter system, approximately 70% of the SO_2 is converted to SO_3 in the first pass, with a resultant rise in temperature. The heat exchangers utilize the heat of reaction in the converter beds to heat the incoming cool gas to the reaction temperature.

The gases containing from 8 to 10% SO_3 and at about 232^{o}C (450^{o}F) enter the absorption tower. The sulfur trioxide is absorbed in the circulating 98% sulfuric acid and subsequently combines with the free water present in the acid to produce additional sulfuric acid.

Storage tanks for 93% through 99% sulfuric acid are fabricated from carbon steel. Smaller storage tanks (under 10,000 gallons) are constructed from stainless steel or a type of HDPE (high density polyethylene) made especially for sulfuric acid service.

The sulfuric acid is transported to markets either by tank truck, railway tank car or by ship. Truck tanks hold 25 to 40 short tons and are constructed of stainless steel. Railway tank cars are 100 short tons capacity, non-pressure general service tank cars with a baked phenolic interior coating to ensure product quality and protect the interior of the tank car against corrosion. Tank cars for shipping 96%, 98% and 99% acid are insulated for winter use and may also have steam heating coils.

17.13 PILOT PLANT INVESTIGATIONS AND TRADE-OFF STUDIES

Recommendations on additional metallurgical test work and trade-off studies required for optimized flowsheet development and process plant design are as follows:

17.13.1 Mineralogical Analysis

"Process Mineralogy," in current practice, emphasizes use of "End-Members" as a basis for metallurgical tests designed to produce grade and recovery information that will be used to make processing or project decisions. An End-Member is a geological entity that has unique features such as:

- Host rock type
- Size of mineralization
- Degree of geological alteration
- Hardness.

A combination of texture, grain sizes, and mineralogical features of the alunite has a direct bearing on product grade and recovery. To efficiently identify these features, the use of QEMSCAN (Quantitative Evaluation of Minerals by Scanning electron microscopy) technology is recommended.

QEMSCAN is an integrated automated mineralogy and petrography system consisting of Scanning Electron Microscope (SEM) with Energy-dispersive X-ray spectroscopy (EDS) detectors, which has found extensive use since 2009.

Several benefits of QEMSCAN include:

- Assessment of alunite mineralogy and disposition of interlocked quartz particles in the alunite matrix.
- Determination of the amount of grinding of ROM alunite required for liberation of alunite.
- Determination of the amount of re-grinding of the flotation concentrate and tailings, if required, to improve product grade and recovery.
- Assessment of impact of impurities present, if any, on product quality and/or on selection of unit operations.

QEMSCAN measurement modes should include, at a minimum:

- Bulk mineralogical analysis.
- Particle mineralogical analysis.
- Specific mineral search.
- Trace mineral search.

QEMSCAN creates mineral compositional maps by scanning the surface of a specimen with a high-energy electron beam. Information is provided on elemental composition at each measurement point. This information is converted into mineral phases.

QEMSCAN data includes mineral and calculated chemical assays. By mapping the surface of the sample, textural properties such as particle and mineral grain size and shape, mineral associations, elemental deportment, porosity, and matrix density are captured and can be reported numerically.

17.13.2 Crushing and Grinding

Conduct tests on dry ROM alunite samples to obtain both crushing and grinding Bond Work Index (kWh/t) measurements.

- Develop quantitative data on liberation of alunite as a function of grain size screen fractions 100 x 150-mesh, 150 x 200-mesh, 200 x 270-mesh, 270 x-400 mesh, and 400 x 10 µm.
- Perform re-grinding and flotation tests on rougher flotation concentrate and scavenger circuit tails to determine the relationship between grinding time, liberation of locked-in

alunite /quartz, and size distribution in the re-ground product, as well as percent recovery and grade of the flotation concentrate recovered.

- Estimate the amount and composition of fines generated during crushing and grinding which can be sent directly to leaching.
- Estimate grinding media consumption.

17.13.3 Flotation Tests

Metallurgical test work is being planned to investigate flotation as a method of reducing the silica content of the feed to the drying and calcination steps. This would also result in enhanced product quality, conservation of energy, smaller equipment sizes in downstream drying/calcining, leaching, and solid/liquid separation operations as well as reduced footprint of the process plant.

Perform flotation tests on ROM alunite, flotation concentrate, and tailings to include:

- Bench-scale tests to evaluate the effect of particle size (coarse and/or fine flotation feed) on flotation efficiency and to optimize grinding requirements.
- Selection and initial dosage of flotation reagents.
- Evaluation of particle size on consumption of reagent(s).
- Coarsest grind size for the rougher flotation circuit with which target recovery may be achieved.
- Bench-scale tests to evaluate water-insoluble, alcohol-based solution of alkyl hydroxamic acid as selective collector instead of the traditional fatty acid or sulfonate and fuel oil collector for Coarse Alunite Feed.
- Pulp density and conditioning time.
- Hot conditioning (at 30°C, 60°C, 90°C) of feed with fatty acids before rougher and cleaner flotation to evaluate the effect of conditioning temperature on selectivity and recovery.
- Pulp density of diluted flotation slurry.
- Aeration, pH, temperature, and residence time in flotation cells.
- Number of cells in rougher, scavenger and cleaner flotation circuits.
- Reagent(s) addition points.
- Regrinding rougher concentrate fine enough to liberate silica inclusions in alunite and desliming re-ground concentrate as feed to cleaner cells.
- Evaluation of stage addition of collector in scavenger circuit and its effect on alunite recovery.
- Regrinding scavenger concentrate, if required and desliming re-ground concentrate as feed to cleaner cells.
- Regrinding scavenger tails as feed to cleaner cells, if required.

- Number of cells (stages) in each circuit.
- Flotation cell residence time.
- Cumulative percent recovery versus flotation time.
- Weight of material floating versus frother addition.
- Effect of recycling process effluents with residual reagents on flotation recovery and reagent consumption.
- Effect of temperature on adsorption of collectors (endothermic chemisorption mechanism).
- Reverse flotation of silica from alunite using cationic amine-salt collectors for silica.

17.13.4 Regrinding and Desliming

- Amount of regrinding of Rougher and Scavenger concentrate.
- Particle size of re-ground product as feed to flotation cells.
- Amount of regrinding of Scavenger tails, if required.
- Desliming of re-ground concentrate or tails.
- Dispersant dosage for desliming, if required.
- Pulp density of feed to hydrocyclone.
- Pulp density of hydrocyclone underflow.
- Percent solids in hydrocyclone overflow.

17.13.5 Thickening Flotation Concentrate and Tails

- Kynch procedure for flotation concentrate and tails slurry solids settling and thickening.
- Thickener/clarifier unit area, overflow rate and detention time requirements.
- Pulp density of thickener underflow as feed to filtration.
- Solids content of thickener overflow).
- Flocculant type and dosage.

17.13.6 Filtration of Thickener Underflow and Leached Slurry

- Filter media type.
- Maximum filtering pressure.
- Maximum operating temperature.
- Filter cake thickness.
- Moisture content of filter cake.
- Filtration rates.
- Washability characteristics of filter cake.

- Filtration cycle time.
- Bulk density of filter cake (feed to dryer).
- Type and dosage of selective flocculation reagents.

17.13.7 Drying Flotation Concentrate Filter Cake

Each dryer/calciner and evaporator/crystallizer equipment manufacturer has developed theoretical models and specialist scale-up know-how. Pilot plant investigations are required to configure the equipment in terms of operating parameter for a particular application, either as single or multistage units, and based on heat transfer calculations and their past experience with similar applications.

- Type of dryer for drying filter cake.
- Particle size of feed to dryer.
- Drying temperature.
- Drying cycle (residence) time.
- Bulk density of dried product (feed to calciner).
- Composition of off-gases leaving the dryer.
- Temperature of off-gases leaving the dryer.
- Estimated flow rate of off-gases from the dryer.
- Moisture content of dried product.
- Friability of dried product.
- Estimated energy consumption.

17.13.8 Calcining Dried Flotation Concentrate

- Calcine the dried flotation concentrate **at a range of temperatures**.
- Type of calciner for dried flotation concentrate.
- Particle size of feed to calciner.
- Bulk density of feed.
- Re-crushing of dried feed to calciner, if required.
- Evaluate the heating rate versus particle size to determine if delumping or re-grinding of dried feed to calciner is required.
- Fuel firing rate.
- Calcining temperature.
- Calcining cycle (residence) time.
- Composition of off-gases leaving the calciner.
- Estimated flow rate of off-gases from the calciner.
- Temperature of off-gases leaving the calciner.
- Composition of calcined product.

- Particle size of calcined product.
- Identify the phases (potassium sulfate, crystalline alumina, and residual alunite) in the respective calcines produced at **a range of temperatures** by x-ray diffraction and microscopic examination.
- Tendency of calcined product to agglomerate (sintering, if any).
- Estimated energy consumption.

17.13.9 Leaching Calcines

Determine by bench-scale tests the operating conditions of the leach circuit, which includes but not necessarily limited to:

- The solubility of potassium and sulfate in the calcines during leaching.
- Identify the phases (potassium sulfate, crystalline alumina, and residual alunite) in the respective calcines produced at **a range of temperatures** by x-ray diffraction and microscopic examination.
- Particle size of feed (calcine) to leach tank(s).
- pH of pulp.
- Temperature of leach.
- Determine percent extraction of potassium and sulfate in the calcine as a function both of calcining temperature and temperature of leach.
- Determination of assay values of potassium and sulfate in the leach residues.
- Pulp density (wt.% solids).
- Leaching time.
- Intensity of agitation (kWh/t solids).

The pulp density of solids at 20wt.% is specified as an operating condition of the water leach circuit in U.S. Patent 4,031,182 (K. W. Loost, July 21, 1977). Leaching calcines at a relatively high (≈40 to 50wt.%) solids content and/or recycling the leach solution to build up the concentration of potassium merits investigation to reduce the large (≈90%) evaporation requirement, the size of leach tanks as well as to improve the potassium recovery without co-crystallization of contaminants in the solution.

17.13.10 Evaporation and Crystallization

Perform bench-scale tests on evaporation and crystallization of SOP from pregnant leach solution to develop information such as:

- Solid-liquid phase diagrams.
- Solubility curves.
- Separation barriers with respect to SOP/glaserite [K_2SO_4/ $K_3Na(SO_4)_2$)] system.

- Solid complexes and co-crystals formed.
- Type of evaporator and crystallizer.
- Batch or continuous mode of equipment operation.
- Percent recoveries.
- Crystallizer effect on particle size distribution.
- Seeding (recycling) requirements, if any.
- Materials of construction.
- Steam temperature and pressure in multiple-effect systems.
- Cooling water requirements.
- Energy requirements.
- Bleed for impurity build-up control.

The crystal growth rate in a solution is dependent on the temperature and concentration of the liquid at the crystal face. These conditions, however, are not the same in the bulk of the solution because a concentration gradient is necessary for the transfer of solute towards the face, and a temperature gradient for the dissipation of the heat of crystallization.

- Bench-scale investigations are, therefore, required to evaluate the problem which involves both heat and mass transfer.
- Additionally, the effect of impurities on crystal formation, the effect of temperature on solubility, changes in viscosity, etc. also should be evaluated.

Bench-scale tests should be conducted on plant feedstock and not on synthetic liquor because trace impurities overlooked or undetectable during analyses can have an effect upon the crystal habit and operating conditions of the crystallizers.

The throughput capacity of the compactor rated at 100 to 120tph should be verified based on test results on the material being processed.

17.13.11 Analytical Requirements

- Composition of ground ROM alunite as feed to Rougher circuit.
- Composition of flotation concentrate from Rougher, Scavenger and Cleaner circuits.
- Composition of flotation tails from Rougher, Scavenger and Cleaner circuits.
- Composition of overflow from desliming hydrocyclone(s).
- Composition of flotation concentrate filter cake.
- Composition of calcined product (feed to leach tanks).
- Composition of off-gases from dryer and calcine.
- Composition of leach tank residues (insolubles).
- Composition of filtrate from leach circuit (feed to evaporator).
- Composition of sulfate of potash product from evaporator.
- Moisture content of sulfate of potash product from evaporator.

.

18 PROJECT INFRASTRUCTURE

Project infrastructure includes road access to the site, gas and power utilities, water supply, and rail spur access to the site. Each of these facilities are described below.

18.1 ROAD

The Interstate Highway nearest to the project site is I-15, accessible in the City of Beaver, Utah. From I-15 in Beaver, the site is currently accessed by traveling west on State Route 21, a two lane, undivided highway, for 31 miles to Milford, Utah, and continuing west on State Route 21 for another 23.5 miles until turning south onto an un-paved County Road. The site is then accessed by traveling to the southwest on this county road for approximately 18 miles.

The county road which begins at SR-21 will be the main access road to the project site. Currently this road is an unpaved, dirt surface road, which is maintained by Beaver County Road Department. Beaver County has a 66ft wide right-of-way (ROW) throughout the length of the roadway. The roadway width varies from approximately 12’ in the mountains to 24’ in the valley floor. The road is primarily used for hunting, recreation, and livestock grazing access.

After years of grading and maintenance by Beaver County, the roadway is approximately 1’ to 2’ lower in elevation than the surrounding ground surface throughout most of its length. Currently surface runoff travels directly over and through the roadway due to the road’s depressed profile and the absence of any cross draining culverts.

Improvements to this road will be necessary to accommodate vehicles transporting material to and from the project site. Improvements may include, raising the existing grade, installing drainage culverts and roadside drainage ditches, and providing a gravel or asphalt paved driving surface. Beaver County prepared a ROW application and Plan of Development to the BLM to upgrade the road so that it can support general economic development in the valley, including the Blawn Mountain Project.

18.2 RAIL

A Railway Terminal will be required to transport approximately 750,000 tons of Potassium Sulphate (SOP) per year from the mining operation in Utah. The SOP product will arrive to the terminal by truck, then transferred and carried by rail to its offload destination. In addition, sulphuric acid would also arrive by truck, transferred and shipped by rail. Diesel fuel will be required to operate machinery and equipment at the mine site and would be required to be delivered by rail to the terminal for offloading into tank trucks.

Union Pacific operates the Lyndyll Subdivision mainline railroad and passes along the Town of Milford Utah. Milford is approximately 200 track miles south of Salt Lake City. The Lyndyll Subdivision is a part of a vital link between southern California and the eastern USA. There is an existing railway yard on the Lyndyll Subdivision in Milford.

18.2.1 Existing Milford Yard

Milford Yard is situated in an ideal location for the transfer and shipment of the mining material to railway outpost destinations; however it would be impractical to route trucks through the town of Milford to access the terminal. Therefore it would be preferable to build a new rail siding including the required storage and loading facilities along the rail line to the north or the south of Milford. There may be operating synergies with the existing rail yard. Discussions will be held with Union Pacific as the project progresses.

Bulk solid and fluids terminals are specialized freight terminals. Yard simulations are often carried out in the development of large rail expansion projects at the initial stages to determine optimum track and transfer facility designs, leading to efficient and safe yard operations.

Some of the key issues to be determined in the development of the yard expansion will be:

- Rail car cycling procedures with the servicing railway and within the facility to support plant operations.
- The need for yard switching.
- Rail car storage requirements.
- Loading and unloading equipment needs.
- Weigh scale requirements and locations.
- Receiving and departure tracks.

18.2.2 Potassium Sulphate (SOP)

A net production of 750,000 tons of SOP per year equates to approximately 15,000 tons per week. Average weekly requirement:

- 100 ton cars = 150 cars/week.
- 120 ton cars = 125 cars/week.

Further information regarding equipment availability, power requirements, scheduling, turnaround times, etc. will be required to proceed into track and yard design.

18.2.3 Diesel Fuel

The majority of the diesel fuel is consumed by the mining operation. Diesel fuel is available in the region and is easily transported to the site by tanker truck where it will be stored in tanks near the mining operation.

18.2.4 Sulphuric Acid

Sulphuric acid is one of the three main commodities obtained from the alunite mineral. Large volumes of acid will be transported by truck to the rail terminal where it will be stored and transferred into railway tank cars. One of the most effective means of moving large quantities of acid overland is by rail. The most economical way to transport this commodity by rail is the use of unit trains. These unit trains vary in length and tonnage and are determined by applicable railway logistics and operating regulations and requirements. Acid tank cars are product specific. Types and sizing are determined based on the product and concentration levels.

Sulfuric acid production is estimated to be approximately 1.8Mtpy. Assuming 10,000 gallon sulfuric acid rail cars are used, a minimum of 441 cars/week or 64 cars/day will be required. Discussions with Union Pacific will be required to identify equipment availability/supply requirements, storage and facility location, yard and mainline operation and other associated railway related issues.

18.2.5 Next Steps

Initiate discussions with Union Pacific (UP) and identify railway equipment requirements and UP operating procedures to allow for required track and transfer facility designs.

18.3 ENERGY CORRIDORS

Two energy corridors pass to the east of the Blawn Mountain tract 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 (U.S. 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 (U.S. BLM, 2011).

18.3.1 Electrical Power

Based on initial review of the availability of electrical power from the power, it is assumed that initial power for the project will be supplied by a stand alone on-site power plant.

18.3.2 Natural Gas

Natural gas is assumed to be supplied by pipeline from the Questar gas pipeline.

18.4 RESOURCES OF CONSTRUCTION LABOR AND MATERIALS

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 to the 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 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.

19 MARKETS AND CONTRACTS

Norwest understands that PRC has conducted at least two market and price studies relative to this project and that there are several other studies and forecasts that are publicly available. Norwest has not independently reviewed this information. The information presented below was provided to Norwest from PRC.

19.1 MARKETS

19.1.1 Agricultural Fertilizer Market

Fertilizers, a large component of the global chemicals industry, consist of essential plant nutrients which 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 173Mt (100% nutrient basis) in 2010/11 and is projected to reach 190Mt by 2015/16.

Sulphur has gained increased attention in the fertilizer industry over the past several years due to the realization that crops were becoming sulphur deficient. Sulphur is necessary for the production of protein, fostering activity and the development of enzymes and vitamins. In addition to these benefits, sulphur also improves root growth and seed production, aids in the creation of chlorophyll and increases resistance to cold temperatures. The addition of sulphur to fertilizer (usually in the form of ammonium sulphate or SOP) creates a blend known as "**NPKS**".

19.1.2 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.

19.1.3 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 "pot ash".

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.

19.1.4 Potash Demand Drivers

Potash demand depends on the demand for fertilizer, which 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 grain and other staple 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. Potash consumption grew at an average annual rate of 4.4% from 2002 to 2007 compared to 3.3% for all NPK nutrients and is forecast to grow at an average rate of 4.7% from 2010 to 2016 compared to 1.9% for all NPK nutrients. This growth will require increased potash production of approximately one additional greenfield potash mine every year.

19.1.5 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 has 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 66.9Mt 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 65.5Mt 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 21.0Mt. 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 86.5Mt of KCl by 2020 with operating rates of 86%.

19.1.6 Chloride-Free Potassium Fertilizer – Potassium Sulphate (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. The chloride ion also inhibits plant growth in dry soils and saline areas. Consequently, the soil's chloride content needs to be carefully managed as it can build up over time, leading to low quality crops.

SOP, comprising approximately 10% of all potash consumption, is a chloride-free potassium fertilizer and is the most commonly used alternative to MOP. 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. The potassium content of SOP is approximately 50% K_2O.

In addition to potassium, SOP contains sulphur which is also an important plant nutrient. The current trend towards fertilizers with a larger component of target nutrients may cause greater value to be placed on the sulphur content of SOP. SOP can be sold as a standard powder or as a premium granular or soluble grade product.

19.1.7 SOP Specific Demand Drivers

There is a price premium attributed to SOP due to the scarcity of primary production relative to MOP. Notwithstanding this price premium, SOP is the most commonly used alternative to MOP when the presence of chloride ions is undesirable. 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, nuts, turf grass and other fruit and vegetable crops. In the United States, these specialty crops account for only approximately 4% of harvested acreage but account for approximately 40% of crop revenues. The potential to increase the yield and quality of these high value crops make SOP a cost-effective input.

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 more than 45% of global demand. Over the past 20 years, the demand for SOP in China has experienced significant growth, growing from approximately 0.5Mtpy in the early 1990s to 1.9Mt in 2011. SOP is also widely used in the fruit growing regions of Mediterranean Europe and the United States. India and Brazil represent markets of large potential SOP demand growth. 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. Brazil used 32,000 tonnes of SOP in 2011 (or 0.4% of total potash consumption) despite growing premium crops which are typically better suited to SOP on 20% of planted land.

According to the IFA, 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 6.0Mt. The low volume of SOP consumption relative to its potential market demand is a result of the scarcity of SOP supply and its price premium over MOP. If the price of SOP decreased relative to the price of MOP, growers may elect to switch to the premium product. The following chart displays historic and forecast global SOP consumption and prices. Table 19.3 illustrates the historical and future predictions of global SOP consumption and commodity prices.

SOP demand has increased from 5.0Mt per annum in 2008 to 6.0Mt per annum in 2011, representing a cumulative average growth rate of 6.0%.

The trend towards fertilizers with a larger component of target nutrients may also have a positive impact on SOP demand that is not reflected in the above forecast. For example, an analysis by Serecon Consulting Group (Serecon) estimated the premium that producers would be willing to pay for SOP over MOP based on the absence of chloride and the presence of sulphur in SOP versus MOP. The analysis was based on the theory that a grower will be willing to pay more for an input if a higher net income would be achieved by using the input. In the case of switching from MOP to SOP as a fertilizer input, increased net income would result from increased yield or increased crop quality.

In estimating the premium for highly chloride-sensitive crops, Serecon used a cap of 40% to reflect the current approximate premium for SOP over MOP. Growers of some highly-sensitive crops, such as almonds and bananas, may be willing to pay even more for SOP than the current premium would suggest.

Serecon concluded that the potential also exists for SOP to become a component in typical fertilizer blends for cereal crops, such as wheat and corn. Sulphur is increasingly being included in recommendations for cereal crop fertilizer blends, as its essential role in the growth and health of plants has become better understood. In the spring of 2012, the premium was approximately 7%. However, the extent of demand for SOP in cereal crop fertilizer blends will largely be dependent upon the relative pricing of sulphate and MOP fertilizer prices. For example, all other things being constant in the model and assuming a $550 per tonne MOP price and a $600 per tonne ammonium sulphate price, SOP would be a preferred component in a typical wheat fertilizer blend if the premium of SOP over MOP was 26.9% or less. This SOP price premium over MOP would be reduced to 9.4% assuming a $600 per tonne MOP price and a $500 per tonne ammonium sulphate price. Cereals are by far the largest fertilizer-consuming crop in the world, accounting for over 60% of total micronutrient fertilizer consumption according to the FAO.

Serecon also concluded that the impact of additional SOP production on the world market, if priced appropriately, would be to potentially replace MOP and fertilizers containing sulphur as well as create additional demand for potassium fertilizer since a large quantity of potassium could be safely applied without the negative impact of chloride.

Based on a preliminary analysis conducted by PRC, the potential exists for United States producers of SOP to displace almost 100,000 tonnes of annual European SOP imports into the Eastern United States. Florida is a key target market for growth as it is home to almost 40% of United States' fruit producing land but only consumes 9,000 tonnes of SOP per year or 3% of United States SOP demand. Outside of the United States, significant market potential is seen in China, Brazil and India. Current annual SOP consumption in Brazil of about 32,000 tonnes represents only 0.4% of total Brazilian potash consumption but premium crops cover almost 20% of planted land area in the country. China consumes 1.9Mt of SOP per year. By way of contrast, India has a population of 1.2 billion and consumes only 50,000 tonnes of SOP per year. The low consumption of SOP in India is primarily due to MOP subsidies, however the Indian government has started reducing the subsidies applicable to MOP. If this trend continues, India could become a major consumer of SOP.

19.1.8 SOP Specific Supply Environment

Total worldwide production capacity of SOP in 2011 was 6.7Mt with approximately 48% located in China, 37% in continental Europe, 9.5% in the Americas and the remaining 5.7% in various other countries. The international market is supplied mainly by two European companies, K+S and Tessenderlo Chemie NV, however, the largest producer is SDIC Xinjiang Luobupo Potash Co., Ltd. (**Luobupo**) in Western China.

In terms of the SOP supply outlook, domestic demand in China has largely absorbed the increased supply coming from the newly-commissioned Luobupo project. While the premium of SOP over MOP has decreased slightly in China, many of the high-cost furnaces used in the Mannheim Process that convert MOP to SOP (**Mannheim Furnaces**) remain competitive due to their location. 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 major new source of SOP has been developed since the 1990s. Several high cost operations using Mannheim Furnaces have curtailed operations and Great Salt Lake Minerals Corporation is the only producer planning to increase production capacity (by 100,000 tonnes). There are a handful of new primary SOP development projects currently under evaluation, although production from these projects is not expected until 2016 at the earliest.

19.1.9 Sulphuric Acid Market Overview

Domestic production of sulphuric acid in the United States is currently about 32Mtpy with domestic demand of approximately 34Mtpy. 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 United States is expected to increase to approximately 2.4Mtpy by 2015.

A significant amount of fertilizer production in the United States, primarily in the Eastern United States, requires sulphuric acid. Overall, the greatest annual demand for sulphuric acid in the United States is derived from the production of phosphoric acid at 23Mt, followed by industrial uses at 6.7Mt, ammonium sulphate production at 2.2Mt and copper production at 1.6Mt. In the Western United States, sulphuric acid is primarily consumed through copper oxide leach operations, followed by use in the fertilizer and industrial sectors.

Currently, there are seven smelters producing sulphuric acid in the United States. Of the three smelters in the Eastern United States, one is expected to shut-down by 2015, reducing annual production of sulphuric acid by 245,000 tonnes. There are four smelters in the Western United States, two of which are located in Arizona, which together supply approximately 1.1Mt of sulphuric 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 copper oxide production facilities in the region and elsewhere. The fourth smelter in the region is a small operation located in Nevada.

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		2012	2013	2014	2015	2016	2017	2018	2019	2020
Potassium Chloride (MOP)	US$/t	488.0	458.0	439.0	419.0	419.0	428.8	442.0	491.0	534.0
Potassium Sulphate (SOP)	US$/t	649.0	618.0	593.0	566.0	566.0	578.0	597.0	663.0	721.0
Sulfuric Acid	US$/t	130.0	132.6	135.3	138.0	140.7	143.5	146.4	149.3	152.3

19.2.1 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 spur 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. Potash prices still remained relatively high throughout much of 2009 as producers were reluctant to lower their prices.

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. Current spot prices of MOP were US$468 per tonne (FOB Vancouver) as of August 2012.

19.2.2 SOP Specific Price Environment

Non-chloride potash prices have generally followed the path of MOP prices in recent years although 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 2012 to 2020, CRU forecasts MOP prices in the range of US$488 and US$534 per tonne (FOB Vancouver/Portland) and SOP prices between US$649 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.

Prices for SOP must remain high enough for the conversion of MOP to SOP to be economically viable. In the event that new primary production of SOP comes online, to the extent this additional supply is not absorbed by higher demand, it will largely displace high-cost MOP conversion operations. This may reduce the price premium of SOP over MOP, however, the extent of the premium reduction will be limited, as growers currently using MOP on crops more suited to SOP would likely start shifting to SOP.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 REGULATORY ENVIRONMENT

20.1.1 State Regulations

Mining and processing operations in the United States must comply with all applicable federal and state regulations. Operations located in Utah require compliance with federal as well as state mining and environmental regulations. Utah has primacy over major environmental disciplines including mining, and air and water permitting. The project is located in southwestern Utah on State controlled land therefore most permits that will need to obtain will be issued by State agencies.

20.1.2 Federal Regulations

Vast amounts of land in Utah are managed by the BLM due to the presence of either federally-controlled surface land or federal mineral ownership. In Utah the control of large parcels of land and minerals were granted to the state to provide a source of revenue from the management of surface use or mineral development. These “state sections” and other lands obtained through additional grants or exchanges from the federal government are managed by SITLA in Utah.

When federal lands (minerals or surface) are impacted, federal approvals are required by the applicable land management agency, most commonly the BLM. Federal actions requiring permits or approvals trigger an environmental review under the National Environmental Policy Act (NEPA). The level of scrutiny a project receives is based upon the land management agencies’ discretion and the significance of impacts to the environment. When impacts are potentially more significant, an Environmental Impact Statement (EIS) may be required. This process can take several years and involves multiple agencies and extensive public input. When impacts are less significant, the federal permit or approval can be substantiated with an Environmental Assessment (EA), which is an abbreviated process that generally can be accomplished within 6-12 months. Regardless of the process that applies, environmental impacts of the project must be evaluated prior to receiving federal authorization to proceed.

The PRC project is located on SITLA controlled mineral and surface land. Because SITLA controls both surface and mineral resources on the parcel, the potential involved for federal involvement under NEPA is minimal.

20.2 ACCESS AND UTILITIES TO THE BLAWN MOUNTAIN ALUNITE PROJECT

The Blawn Mountain project area is 11,549.2 acres in southwestern Utah located on surface land and minerals controlled by SITLA.

An existing county maintained road (Revenue Basin Road) provides access to the project area. PRC will utilize this road to access the project area, bring necessary utilities to the project, and initially to transport product to a rail load-out located close to the town of Milford, Utah some 30 miles away. In its current condition, the road is not adequate to accommodate the type and amount of vehicles needed to support the project. Additionally, the road is not wide enough to include the necessary utilities, which further necessitates the need to upgrade the road. The land adjacent to the road is managed by the BLM and impacts to this land required for expansion will require a ROW from the BLM.

Revenue Basin Road also provides access to several other SITLA parcels, and provides access to areas with potential for renewable energy project development. To help encourage these multiple development opportunities, Beaver County applied for the ROW on the basis that improvement of the road for future uses will enhance economic development in their County as well as adjacent counties. While the wide environmental study corridor could include sufficient room for utilities, the County ROW requested only the road upgrade. Utilities will be addressed in subsequent requests to BLM by the specific utility service providers.

The BLM is currently preparing an EA to evaluate the potential impacts from the ROW. The ROW is expected to be granted the end of year 2012. Based upon legal precedent (Cotter Decision" Utah v. Andrus, 1979), BLM is obligated to allow "reasonable access" to facilitate development of State controlled resources.

20.3 HISTORICAL ENVIRONMENTAL STUDIES

In the 1970's ESI proposed to develop the Blawn Mountain resource as an alumina project, with alunite being the primary mineral of interest. At that time the land and minerals were managed by the BLM. At that time, the project was located on land managed by the BLM. In 1977, the BLM completed an environmental review under NEPA, EIS level study, on the proposed project in compliance with NEPA. The EIS addressed both impacted natural resources and socio-economic impacts of the project. At that time, the project was evaluated as an alumina, potassium sulphate and phosphate fertilizer project. Environmental impacts that were evaluated included air quality impacts from the mine, primarily associated with particulates, and processing plant impacts, including SO_2, NOx and fluorides, to surface and groundwater, wildlife, soils, vegetation, cultural resources and socio-economic impacts resulting from a large industrial project in a predominately agricultural community. Mitigation measures were proposed to reduce impacts and were evaluated by various agencies and deemed sufficient. Due to market conditions, the project was never launched.

Subsequently, the BLM, through a land exchange process, granted the Blawn Mountain land and minerals, and other surrounding areas to SITLA. As detailed above, PRC has an exploration agreement with SITLA and is in the process of completing the necessary studies to convert to a mineral lease. PRC has obtained an exploration permit from the Division of Oil, Gas and Mining (DOGM) to complete exploration work to further delineate and quantify the resource.

20.4 ENVIRONMENTAL SETTING

A review of the 1974 Alunite EIS, the BLM Cedar City Field Office's Resource Management Plan (RMP) (completed in 1983), and other publically available information was conducted to evaluate potential environmental concerns for the Blawn Mountain Alunite Project. Additionally, other resources of publically available data including the U.S. Fish and Wildlife Service (FWS), and the Utah Division of Wildlife Resources (UDWR) were assessed.

These documents are dated and as such physical conditions, as well as other numerous factors including project design, environmental regulation and agency policies, management objectives, etc. have changed significantly causing some of the information included in the documents to be obsolete. The review indicated that there are some environmental issues which need to be addressed prior to obtaining permit approval.

There are FWS Threatened, Endangered, Candidate or BLM Special Status plant/wildlife species identified within Beaver County. The information from these previous studies is very general and these species and/or their habitat may or may not occur within the project area. Additional evaluation needs to be conducted to determine the likelihood of these species occurring. In the event that these species or their habitat occur within the project area and are impacted by the project, FWS/BLM consultation may be required to develop mitigation plans.

In 1999 the BLM inventoried land in the Central Wah Wah Mountains for wilderness characteristics. Some of the area did possess wilderness characteristics. Some of this area is adjacent to the proposed access road to the north. The proposed access roads and lease areas do not directly affect either of these areas; however, because of the project's close proximity to these areas, special interest groups may choose to participate in commenting on permits for the site.

Table 20.1 is a summary of findings of environmental resources evaluated within the project area.

TABLE 20.1 ENVIRONMENTAL RESOURCE WITHIN PROJECT AREA SUMMARY

Resource	
Air Quality	The proposed mine is in a NAAQS attainment area. The nearest mandatory Class I airshed that require protection is Zion National Park (60 miles). Potential impacts to this area will need to be evaluated as plant processing emissions are better understood.
Cultural Resources	A current cultural resource inventory of the proposed project area will be necessary. Previous studies indicate that there no National Register Eligible sites are likely to be affected.
Fish Habitat	No streams, lakes, or ponds that provide fish habitat are located on the proposed mine area and none would be crossed by the required access roads.
Migratory Birds	Birds protected by the Migratory Bird Treaty Act (MBTA) including several species of raptors are known to occur along the proposed access road and in the mine area. Nest inventories and timing restrictions may apply to the project but are manageable issues
Paleontology	Mining could affect paleontological resources. Inventory and mitigation typical for similar projects would be required.
FWS Threatened, Endangered, Candidate or BLM Special Status Plant and Animal Species	Within the project area, there are some species on BLM sensitive species list in Beaver County. An assessment for habitat occurrence would likely be needed and if habitats are present, appropriate inventories, consultation and mitigation would be required.
Water Resources/Quality (drinking/surface/ ground)	Mining operations could affect subsurface water bearing strate. Analysis and mitigation would likely be required. Additional information in the hydrology section of this report.
Wetlands/Riparian Zones	No wetlands or waters of the U.S. are likely to occur in the project area. Small riparian areas may be present and are discussed in the ACOE jurisdictional waters section of this report.
Wilderness/WSA	There are no designated Wilderness areas in the vicinity of the project. Therefore no designated wilderness or WSA would be affected.
Wildlife Excluding USFWS Designated and BLM Sensitive Species	Mule deer, antelope and elk are game species that inhabit the project area. The project area is not within any critical or winter habitat areas. Special status species are noted above. Inventory and analysis may be required.
Vegetation Excluding FWS Designated and BLM Sensitive Species	Road construction, mining, and surface disturbing activities would impact vegetation. Unique or special plant communities have not been identified in the vicinity of the project area. Vegetation inventories would be necessary for the mining operation.

Several of the permits discussed below will require environmental baseline data to support the applications.

20.5 MAJOR OPERATING PERMIT AND AUTHORIZATIONS

The following discussion and Table 20.2 identifies the major permits that need to be obtained prior to the construction and start-up of the Blawn Mountain project.

TABLE 20.2 MAJOR REQUIRED PERMITS

Major Permits or Approvals	Issuing Agency
Mining Permit	Division of Oil, Gas and Mining
Water Appropriations	Utah Office of State Engineer
Groundwater Discharge Permit	Utah Division of Water Quality
Air Quality Permit	Utah Division of Air Quality
Storm Water and Discharge Permit	Utah Division of Water Quality
Dredge and Fill Permit	U.S. Army Corps of Engineers
County Conditional Use Permit	Beaver County
Federal Right-of-Way	U.S. Bureau of Land Management

This report is not meant to be all inclusive and covers only the major permits required. In addition to the permits discussed in this report, other ancillary environmental permits and authorizations include spill control and response plans, hazardous waste management/transportation authorization, authorization for use of low level radioactive sources, and hazardous chemical training for employees.

20.5.1 Utah Division of Oil, Gas and Mining Notice of Intent

All hardrock mining operations, such as the Blawn Mountain Alunite Project, that disturb more than 5 acres of land must file a Notice of Intent (NOI) and obtain approval from the Utah DOGM prior to beginning operations. Permit applications must contain a complete description of the environmental resources and impact analysis in the area to be mined, a description of mining methods, a comprehensive reclamation plan and a financial security instrument acceptable to DOGM to cover the costs of reclamation to be completed by an independent third-party.

PRC needs to complete the environmental baseline studies for the site, and complete the mining and reclamation plan. Preparation, submittal and approval of the NOI can typically be completed within one year assuming sufficient planning is integrated into the baseline data collection schedule, a comprehensive, well organized application is submitted, and the project is not highly controversial. Mining permit approval time is typically 9-12 months. NOI approvals can be challenged by interested parties which can delay final authorization.

20.5.2 Water Appropriations

Acquiring sufficient water is a key issue for the project. There is a strong focus on reducing water consumption for the project. High priority or senior rights allow the holder to take water at critical periods of the year such as late July and August when water is in greatest demand. A defensible appropriations application was filed with the State Engineer’s Office. The State Engineer is currently reviewing the application. The SEO office does not have a fixed timeline to respond to applications. However, a permit decision is expected by mid-2013.

20.5.3 Groundwater Discharge Permit

Mining projects that include a potential source of contamination to groundwater resources must complete a groundwater discharge permit application. The Utah Division of Water Quality (DWQ) will review the application and determine if a permit is required. Groundwater discharge permit applications will require PRC to complete sufficient groundwater investigations to be able to evaluate potential impacts to the resource, and if necessary provide sufficient mitigation. As part of the current exploration drilling program, 5 groundwater wells were completed to help characterize the hydrologic conditions of the lease area. The hydrologic interpretation will be included in the groundwater permit application. Actual permit processing time is based upon the thoroughness and level of organization of the application and whether there are sensitive groundwater issues (source and quality) in the area where the project is located. Groundwater permit applications typically are processed in approximately 9-12 months. PRC has been proactive on this topic

20.5.4 Air Quality Permit

In Utah, all sources that emit a regulated pollutant are required to obtain an Approval Order from the Utah Division of Air Quality (UDAQ). Sources that emit more than 250tpy of any regulated pollutant are considered to be major sources under the Clean Air Act. Based upon preliminary plant design calculations, the plant is expected to exceed the 250tpy for several pollutants including SO_2, NOx, O_3, and coarse and fine particulates. Preliminary estimates of greenhouse gas emissions will exceed the 100,000tpy that trigger the need for a permit for those pollutants.

The project is located in an Attainment Area which is a designation assigned to air sheds that meet the National Ambient Air Quality Standards (NAAQS). The project is also located in relatively close proximity, approximately 100 miles, to at least one Federal Class I air shed, Zion National Park. Federal Class I air sheds include most National Parks or other pristine air sheds where air quality values that enhance the visitors experience are protected. Permitting requires sources to establish baseline meteorological data and background concentrations for regulated pollutants, and conduct modeling to predict the dispersion of emissions from the source.

PRC has been proactive in addressing the regulatory requirements to obtain an air quality permit by meeting with the UDAQ and identifying the level of investigation required to obtain a permit. A monitoring plan was prepared by PRC and approved by UDAQ. In September, an air monitoring station was installed near the Blawn Mountain Alunite Project and a particulate monitoring station was installed closer to Milford, Utah at a location approved by the UDAQ. This monitoring station will gather air quality data for approximately one year.

The air quality permit will be the longest lead-time permit based upon the requirement to collect background air quality data and drive the overall environmental permitting schedule. After gathering the one year of baseline data the application preparation and approval can take between 12 and 15 months. The timeline for permit approval can be reduced by submitting a thoroughly complete and technically correct application, and by supporting the UDAQ efforts to coordinate approval with the Federal Land Manager for Class I airsheds. Early submittal of modeling results prior to submitting the entire application can also help expedite this process.

20.5.5 Storm Water and Point Source Discharge Permit

Project construction requires a National Pollutant Discharge Elimination System (NPDES) storm water permit to control off-site sedimentation. Utah has obtained full jurisdiction from the Environmental Protection Agency (EPA) for implementing NPDES requirements. The storm water pollution prevention plan (Plan) to control erosion will be developed from the mining and reclamation plan. The Plan must be fully developed and permit coverage granted prior to breaking ground at the site.

Once the project is on-line and point sources (sediment ponds) are required, an industrial NPDES permit for these sources will be required.

Both the storm water Plan and the NPDES design requirements will be developed as part of the drainage control plan for the operation. If impoundments are larger than 20 acre feet in capacity or exceed 20 vertical feet from base elevation on the downstream side, a Dam Safety Certification will be required from the State and Mine Safety and Health Administration. These authorizations are common to the industry and will evolve from design work associated with the project. Lead times for the NPDES and Dam Safety Certification/MSHA approvals typically take 6-9 months.

20.5.6 Army Corps of Engineer's Jurisdictional Waters

The U.S. Army Corps of Engineers (ACOE) regulate Section 404 of the Clean Water Act. Section 404 Permits under the Clean Water Act are required to fill or dredge "jurisdictional" waters or waterways of the U.S. Permits or approvals granted by the ACOE require NEPA compliance because it constitutes a federal action. Mining-related small scale impacts to jurisdictional waters, less than 0.5 acres, may qualify under a Nationwide Permit which would not require a NEPA analysis.

A preliminary assessment of ACOE jurisdictional waters for the proposed access road was completed. Though some areas along the road corridor may fall within categories for which ACOE generally will assert jurisdiction, the plan would be to avoid these areas entirely or keep the disturbance level below 0.5 acres to qualify for a Nationwide Permit.

An assessment of jurisdictional waters was completed for lands within the SITLA tract boundary. This work did not constitute a formal delineation which the ACOE is required to endorse, but was completed to establish areas with potential jurisdictional waters. This assessment evaluated a few small areas of potential jurisdiction. PRC would likely avoid these areas or maintain disturbance to less than 0.5 acres required for a Nationwide Permit.

20.5.7 County Conditional Use Permit

Beaver County, where the project is located, will require PRC to obtain a Conditional Use Permit, CUP, for the project. CUPs focus on direct impacts to the social fiber of the community and the impacts to infrastructure. The CUP will require an analysis of the impacts on social programs including schools, medical facilities, law enforcement and employee housing, as well as traffic and noise.

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 2-3 months once all the supporting studies have been completed.

20.5.8 Beaver County - BLM Right of Way (ROW)

Beaver County prepared a ROW application and Plan of Development to the BLM to upgrade the Revenue Basin Road so that it can support general economic development in the valley, including the Blawn Mountain Project. The Federal ROW is being pursued by Beaver County under a separate action under the National Environmental Policy Act (NEPA) not directly connected to the project. The BLM is in the process of preparing and EA for the ROW. The anticipated time period to complete an EA is 6-12 months. The BLM has relayed to the County that the analysis and ROW grant will be complete end of 2012.

20.6 SUMMARY

Permits and authorizations required for the Blawn Mountain Alunite Project are common to most major mining and processing operations in Utah and throughout the U.S. The schedule to obtain the permits listed above may vary somewhat. However, integration of sound design principles and some flexibility with respect to the project footprint and environmental impacts as well as the implementation of mitigation measures should result in issuance of all required permits for the project.

21 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COSTS

Capital costs for both the processing plant and mine are summarized in Table 21.1 below.

TABLE 21.1 TOTAL PROJECT CAPITAL ESTIMATE (USD M'S)

	Year -2	Year -1	Total Construction and Development Capital	Sustaining Capital	Total Life of Project Capital
Processing Plant	$547	$548	$1,095	$87	$1,182
Mining Operation		$40	$40	$65	$105
Contingency	$80	$85	$165	$10	$175
Total	**$627**	**$673**	**$1,300**	**$162**	**$1,462**

The above table does not include capital costs for third party facilities such as the powerhouse and the water treatment facility discussed in Table 21.2 below; however, it does include capital costs for the sulfuric acid plant. The Project economics include the capital and operating costs for the powerhouse and water treatment facility, but only the operating costs for the sulfuric acid plant. The rates used are assumed to cover operating costs, return on capital for the operator, recovery of capital and operating expenses.

21.2 BASIS OF ESTIMATE FOR PROCESSING PLANT

21.2.1 Methodology

The capital cost estimate is based on the industry standard front end loading one (FEL1) conceptual engineering and design of a plant capacity of 750,000tpy of sulfate of potash.

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.

21.2.2 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%/+50%.

21.2.3 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.

21.2.4 Currency

Unless otherwise specified, the default currency is United States dollars (USD).

21.3 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 whose costs is significant. 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:

- Equipment Delivered
- Flow sheet adjustment
- Free-on-board (FOB) plant adjustment
- Installation
- Piping
- Structural steel foundations, reinforced concrete
- Electrical
- Instruments
- Battery limits, building and service
- Excavation and site preparation.

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.

21.4 INDIRECT COSTS

21.4.1 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 detail 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.

21.4.2 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.

21.5 CONTINGENCY AND TAXES

- Contingency- An allowance of 25% of the total direct cost is included to cover the unknowns. 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.
- Taxes- Utah state sales tax at 6.85% is applied to 30% of the total direct cost.

21.6 THIRD PARTY COSTS

21.6.1 AREA 1000 Powerhouse Services

The costs for the power house services are a lump sum turnkey installation by a third party. The cost for the powerhouse services includes the capital cost for the equipment needed to generate 60MW of electrical power with natural gas turbine-generator sets and heat recovery steam generators (HRSG) to produce 620,000 pounds per hour of steam required for the process. It also includes the cost to install a natural gas pipeline to connect with Kern River's 36" pipeline in Iron County via the shortest route available (approximately 26 miles). Natural gas consumption for this facility will be approximately 2M cubic feet per day. The cost includes the necessary auxiliary equipment such as feed-water pumping system, SCR emission control system, continuous emission monitoring system, turbine inlet cooling system, deaerator, electrical tie-in equipment, and associated items. The order of magnitude (-30/+50%) estimated cost for the powerhouse services is approximately $160.0M.

21.6.2 AREA 1100 Sulfuric Acid Plant

The costs for the acid plant, storage and load out are a lump sum turnkey installation by a third party. The cost for the 250tph (6,000tpd) sulfuric acid plant is approximately $140.0M. The feed to the acid plant for this capacity consists of an estimated 337,000 SCFM furnace off-gases containing **not less than** 10% by volume SO_2. For acid plant feedstock containing less than 10% SO_2, a sulfur burning plant is required at an estimated **additional cost** of approximately $100.0. The acid plant consists of a drying tower, heat exchangers, 4-pass catalytic converter, inter-pass tower and final tower with mist eliminators, and product acid cooler. The estimated cost of furnace gas cleaning train is $5.0M. Acid storage tank system is estimated at $3.5M. Acid loadout system is estimated at $31.5M. The order of magnitude (-30/+50%) estimated cost for the sulfuric acid plant is approximately $180.0M.

21.6.3 AREA 1200 Water treatment Plant

The costs for the water treatment plant are a lump sum turnkey installation by a third party. The water and wastewater treatment facilities at the alunite processing plant site include the following:

- A package plant for potable water treatment, which consists of flocculation, coagulation, filtration, and disinfection of water harvested from the well field for on-site uses.
- A boiler water treatment plant for steam generation with provisions for hardness removal, demineralization, and addition of oxygen-scavenger and corrosion inhibitors.
- A package sewage treatment plant for treatment of domestic wastewater.

- An industrial wastewater treatment plant with provisions for oil/grease removal, pH adjustment, and clarification of reclaimed water for reuse in the operations.
- The order of magnitude (-30/+50%) estimated cost for the water treatment plant is approximately $40.0M.

21.7 ESTIMATE SUMMARY

21.7.1 Cost Breakdown by Area

Table 21.2 below summarizes the capital cost estimate for the plant by area.

TABLE 21.2 750,000 MTPY SOP

DESCRIPTION	UNIT	TOTAL COST (M)
DIRECT COST SUMMARY		
AREA 100. Primary Crushing	USD	$ 16.6
AREA 200. Alunite Stockpile and Reclaim	USD	$ 24.2
AREA 300. Wet Grinding and Classification	USD	$ 76.7
AREA 400. Flotation and Flotation Concentrate. Solid/Liquid Separation	USD	$ 34.2
AREA 500. Flotation Concentrate and Tailings Dewatering	USD	$ 31.7
AREA 600. Concentrate Drying and Calcination.	USD	$ 198.5
AREA 700. Calcine Leaching and Solid/Liquid Separation.	USD	$ 21.5
AREA 800. Crystallization and SOP Product Solid/Liquid Separation	USD	$ 192.9
AREA 900. Product Drying, Compaction, Storage and Load Out	USD	$ 36.6
AUXILIARY SERVICES – Fire Protection	USD	$ 3.2
Total Direct Cost:	USD	**$ 636.0**
INDIRECT COST SUMMARY		
EPCM Cost	USD	$ 127.2
Construction Related Cost	USD	$ 139.9
Owner's Related Cost	USD	$ 0
Contingency and Tax	USD	$ 172.1
Total Indirect Cost:	USD	**$ 439.2**
Subtotal Project Cost:	USD	**$ 1,075.2**
THIRD PARTY COST SUMMARY		
AREA 1000. Third Party Power House Services	USD	$ 160.0
AREA 1100. Third Party Sulfuric Acid Plant.	USD	$ 180.0
AREA 1200. Third Party Water Treatment Plant	USD	$ 40.0
Total Third Party Cost:	USD	**$ 380.0**
TOTAL INSTALLED PROJECT COST	USD	**$ 1,455.2**

21.8 BASIS OF ESTIMATE FOR MINING OPERATION

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 21.3 below summarizes the capital cost estimate for the major mining equipment.

TABLE 21.3 CAPITAL COST ESTIMATE FOR MAJOR MINING EQUIPMENT (USD 000'S)

Primary Equipment				Unit Cost	Total
Hitachi EX3600	Backhoe	28yd^3	1	$6.2	$6.2
CAT 992K	Front-End-Loader	16yd^3	1	$2.0	$2.0
CAT 785C	Truck	148 ton	12	$2.6	$31.2
Support Equipment					
CAT 785C H_2O	Water Truck	16,000 gallons	1	$0.6	$0.6
CAT 16M	Grader	297Hp	2	$0.8	$1.6
CAT D10T	Dozer	580Hp	2	$1.1	$2.2
AC DM45	Drill	45,000Lb	1	$1.4	$1.4

Additional capital for the mining operation includes an estimate of $100,000 per year for small equipment and $7.4M for mine facilities including a shop, warehouse, office, and bathhouse.

21.9 OPERATING COSTS

Average annual operating costs for the processing plant and mining operation are shown in Table 21.x below. All costs are stated in constant 2012 dollars, there is no provision for inflation.

TABLE 21.4 AVERAGE ANNUAL PLANT AND MINE DIRECT OPERATING COSTS (USD$M)

Direct Plant and Mine Operating Costs	Life of Plant Annual Average
Mining	
Labor and Benefits	$12
Materials and Consumables	$21
Total Mining	**$33**
Processing Plant	
Labor and Benefits	$27
Materials and Consumables	
Crushing and Grinding	$15
Floatation	$11
Calcination	$47
Leaching and Crystallization	$4
Drying and Compaction	$3
Powerhouse	$23
Water Treatment Plant	$3
Acid Plant	$4
Contingency	$18
Total Processing Plant	**$155**
Total Direct Operating Costs	**$188**

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 750,000 tons of SOP and 1.8Mt of sulphuric acid.

Labor and benefits costs were provided by PRC and are based on a regional survey escalated to 2012. PRC also provided manpower requirements for the plant while Norwest developed manpower requirements for the mine as shown in Section 16.

Plant equipment operating costs, except for the powerhouse, 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. Costs for the operation of the powerhouse, 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 2012 Western Mine and Mill Cost Estimating Guide. 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 $3.00 per mmbtu and costs for electricity and water based third party operation of those facilities.

A contingency of 10% of estimated plant and mine operating costs is included. While 10% may be in the low range of contingency estimates, it is noted that the economics of the project are not very sensitive to changes in direct operating costs.

22 ECONOMIC ANALYSIS

22.1 PRINCIPAL ASSUMPTIONS

22.1.1 Production Volume and Schedule

Production volume is planned at 750,000 tons of SOP per year for the 30 year life of the project included in this analysis. As a result of the SOP production process, 1.8Mt of sulphuric acid are also produced annually. This requires an annual average of 17.1Mt of alunite which fluctuates within a small range depending on the grade being mined in a given year. Over the 30 year period, there are 22.6M tons of SOP and 54.2 M tons of sulphuric acid produced.

Future studies will focus on expanding the resource base and extending the life of the project beyond the 30 years in this report.

Estimated costs of start-up and commissioning are $80M and are included in cash flows in the year preceding full production. There will be some production of SOP and sulphuric acid during the process which is assumed to offset the cost of start-up and commissioning. Because start-up and commissioning occur in the year before production begins, full production is assumed in Year 1.

22.1.2 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 transportation costs. Transportation costs were provided by PRC.

TABLE 22.1 PRICING AND TRANSPORTATION SUMMARY USD

Pricing and Transportation Costs	Unit	Life of Plant Annual Average $M
SOP Tons Sold	750,000	
Sulphuric Acid Tons Sold	1.8M	
Average SOP Selling Price – FOB Port	$640/t	$480
Average Sulphuric Acid Selling Price – FOB Rail	$130/t	$234
Transportation from plant to rail (including loading and handling)	$6/t	$14
Transportation to port	$45/t	$34
Net revenue per ton of product – FOB Plant	**$261/t**	**$665**

22.1.3 Cash Production Costs

Direct cash production costs were summarized in Section 21. Additional cash production costs include site G&A expenses, property taxes and insurance, corporate overhead and royalties. These costs, except royalties, were provided by PRC. Royalties are based on the lease agreement which provides for a royalty of 5% and 4% of selling price for SOP and sulphuric acid respectively. Total cash production costs are shown in Table 22.2 below.

TABLE 22.2 TOTAL CASH PRODUCTION SUMMARY

Total Cash Production Costs	Unit	Life of Plant Annual Average $M
SOP Tons Sold	750,000	
Sulphuric Acid Tons Sold	1.8M	
Direct Plant and Mine Cash Production Costs	$73/t	$188
Royalties	$13/t	$33
Site G&A	$2/t	$5
Property Taxes and Insurance	$3/t	$6
Corporate Overhead	$1/t	$3
Total Cash Production Costs	**$92/t**	**$235**

22.1.4 Income Taxes

Income taxes include both U.S. Federal and State of Utah corporate taxes at a blended rate of 38.25% applied to estimated taxable income from the project. Tax depreciation is calculated based on U.S. 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	750,000
Sulphuric Acid Tons Sold	1.8M
Net Revenue FOB - Plant	**$665**
Direct Plant and Mine Cash Production Costs	$188
Royalties	$33
Site G&A	$5
Property Taxes and Insurance	$6
Corporate Overhead	$3
Total Cash Production Costs	$235
Operating Margin	$430
Income Taxes	$105
Cash Flow from Operations	**$325**

Pre-production cash outflows total $1.3B over the two year construction period. Cash flow is positive beginning in Year 1. Payback occurs at the end of Year 5 which is 7 years after the initial investment. Cash flow after payback averages $288M per year for a total net cash flow of $7.2B over the life of the project.

The summary of cash flow for the project is presented in Table 22.4.

TABLE 22.4 PROJECT CASH FLOW

Project Cash Flow Summary	**Year -2**	**Year -1**	**Year 1**	**Year 2**	**Year 3**	**Average Year 4-8**	**Average Year 9-13**	**Average Year 14-18**	**Average Year 19-23**	**Average Year 24-30**	**TOTAL**
SOP Tons Sold		0.13	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	23
Sulphuric Acid Tons Sold		0.25	1.56	1.77	1.95	1.81	1.78	1.81	1.78	1.83	54
Net Revenue FOB - Plant		86	545	575	611	670	674	678	674	680	20,059
Direct Plant and Mine Cash Production Costs		63	187	193	196	194	191	190	184	181	5,704
Royalties		5	28	29	30	34	34	34	34	34	1,005
Site G&A		2	5	5	5	5	5	5	5	5	142
Property Taxes and Insurance		3	6	6	6	6	6	6	6	6	173
Corporate Overheads		-	3	3	3	3	3	3	3	3	75
Total Cash Production Costs		73	228	235	239	241	237	237	230	228	7,099
Operating Margin		13	318	341	372	429	436	441	444	452	12,959
Income Taxes		2	58	41	57	88	94	110	126	129	3,149
Cash Flow from Operations		10	259	299	316	341	342	331	318	323	9,810
Capital Expenditure	627	671	42	20	4	2	5	11	11	1	1,520
Net Cashflow	(627)	(661)	217	279	311	339	338	320	306	323	8,291

22.3 FINANCIAL ANALYSIS

The internal rate of return for the project is 21.3%. 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	$1.9B	$1.3B	$0.9B

The PEA is preliminary in nature, and includes inferred mineral resources. Inferred mineral resources are considered too speculative geologically to have technical and economic considerations applied to them that would enable them to be categorized as mineral reserves. Mineral resources that are not mineral reserves do not have demonstrated economic viability. Accordingly, there can be no certainty that the results estimated in the PEA will be realized.

22.4 SENSITIVITY ANALYSIS

Table 22.6 below shoes the sensitivity of the project economics to changes in selling price, direct operating costs, and capital costs.

TABLE 22.6 SENSITIVITIES

Discount Rate	8%	10%	12%
Base Case	$1.9B	$1.3B	$0.9B
10% Increase in Selling Price	$2.3B	$1.7B	$1.2B
10% Decrease in Selling Price	$1.5B	$1.0B	$0.7B
10% Increase in Operating Costs	$1.8B	$1.2B	$0.8B
10% Increase in Capital Costs	$1.8B	$1.2B	$0.8B

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 present report.

24 OTHER RELEVANT DATA AND INFORMATION

There are no relevant data and information applicable to this report, other than sources referenced in Section 27.

25 INTERPRETATION AND CONCLUSIONS

The mineral tracts are controlled by PRC by the Exploration/Option Agreement through SITLA. The property has undergone past exploration which can be considered sufficient for the delineation of a mineral resource 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. Geologic modeling based on drilling data and field mapping depicts a single mass of alunite for Area 1 at approximately a mile long by 900ft wide and with an average thickness of approximately 500ft. Area 2 alunite body is approximately 1.8 miles long by 4,700ft wide and with an average thickness of approximately 500ft.

A mineral resource has been estimated and reported for a range of K_2O cut-off grades for Area 1 and Area 2. Measured plus Indicated in situ resources for Area 1 range from 159Mt of in situ resources with no K_2O cut-off to 81Mt using a 3% grade cut-off. At the preferred scenario of a 1% cut-off Measured plus Indicated K_2O is estimated to be 5.0Mt and Al_2O_3 is estimated to be 21.7Mt. At the 3% cut-off Measured plus Indicated K_2O is estimated to be 3.4Mt and Al_2O_3 is estimated to be 12.5Mt. Average K_2O grade ranges from 3.22 to 4.20% and Al_2O_3 grade ranges from 13.9 to 15.35% for the respective cut-offs.

Measured plus Indicated in situ resources for Area 2 range from 479Mt of in situ resources with no K_2O cut-off to 260Mt using a 3% grade cut-off. At the preferred scenario of a 1% cut-off Measured plus Indicated K_2O is estimated to be 14.3Mt and Al_2O_3 is estimated to be 61.1Mt. At the 3% cut-off Measured plus Indicated K_2O is estimated to be 9.7Mt and Al_2O_3 is estimated to be 36.2Mt. Average K_2O grade ranges from 3.07 to 3.74% and Al_2O_3 grade ranges from 13.16 to 13.95% for the respective cut-offs.

Much of the interpretation and mineral resource estimations were derived through a 3D block model created from drilling and field mapping data using MineSight® modeling software. The details of the methodology are described in the report text.

26 RECOMMENDATIONS

26.1 DEVELOPMENT DRILLING

The results of the 2012 drilling program should be included in the geological database and a new geological model produced for Area 1. The infill drilling will be providing additional grade data that is consistent with current laboratory standards as well as providing geotechnical data and bulk sample material for alunite processing simulations. This data will provide the necessary information for prefeasibility level mine planning and higher-level engineering.

A large number of drill holes that define the block model in Area 1 terminate in mineralized material. The geometry of the block model suggests potential to define additional resources at greater depths. Norwest recommends drilling 30 holes that will specifically attempt to identify mineralization to greater depths. Projected drill depths range from 400 to 1,000ft for a total of 21,000ft using a 700ft average hole depth. To support mine planning and design, Norwest recommends drilling ten of the proposed holes as continuous core holes from surface to collect samples for geotechnical characterization and gather detailed geochemistry and mineralogy information. This will provide vital information that will support mine planning and process design.

The exploration budget for Blawn Mountain for 2013 is presented in Table 26.1.

26.2 MINE PLANNING AND PROCESSING REFINEMENT

Additional detailed planning and study is recommended for the mine planning and processing components for Blawn Mountain. This will result in a more accurate recoverable reserve base. This work should initially be completed at the prefeasibility level of project evaluation. Norwest estimates the cost of this additional work would be in the range of $3,000,000 to $4,000,000. Assuming positive results are received from the prefeasibility study, a full feasibility study is recommended and an estimate of costs associated with that work would be between $6,000,000 and $8,000,000. Concurrent with the prefeasibility study, Norwest recommends additional metallurgical and pilot scale testing to aid in the detailed design of the processing facility and to provide better estimates of recoveries. The cost of this work is estimated at $3,100,000.

The work performed in these studies would cover all aspects of the project including optimized mine planning, infrastructure and processing facilities design as well as the capital and operating costs associated with these activities.

TABLE 26.1 EXPLORATION BUDGET FOR 2013

Description	Unit Cost ($)	Jan – May		June		July		Total
Personnel		**Units**	**Cost**	**Units**	**Cost**	**Units**	**Cost**	
Project Manager	2,100	7	14,700	14	29,400	21	44,100	88,200
Field Geologists	1,500	7	10,500	60	90,000	60	90,000	190,500
Field Technicians	1,000			20	20,000	20	20,000	40,000
SubTotal			**25,200**		**139,400**		**154,100**	**318,700**
Permitting								
Drill Planning	35,000	1	35,000					35,000
NOI Amendment	5,600	1	5,600					5,600
Cultural/Environmental Surveys	58,000	1	58,000					58,000
SubTotal			**113,600**					**98,600**
Site Preparations								
Dirt Work – Site Preparations, Roads	3,500	7	24,500	12	42,000	12	42,000	108,500
Post Abandonment	1,500			7	10,500	14	21,000	31,500
Reclamation	3,700					18	66,600	66,600
SubTotal			**24,500**		**52,500**		**129,600**	**206,600**
Area 2 Drilling								
Core	50			3,500	175,000	3,500	175,000	350,000
Reverse Circulation	25			7,000	175,000	7,000	175,000	350,000
SubTotal					**350,000**		**350,000**	**700,000**
Analyses								
Sample Preparation	9.05			1050	9,503	1050	9,503	19,005
Major Oxides ICP	29.75			1050	31,238	1050	31,238	62,475
Trace Elements ICP	14.3			350	5,005	350	5,005	10,010
Mineral XRD	125			175	21,875	175	21,875	43,750
Geotechnical	2,175.00			20	43,500	20	43,500	87,000
SubTotal					**92,259**		**94,157**	**238,797**
Down hole Geophysics	2,000			4	8,000	4	8,000	16,000
SubTotal					**8,000**		**8,000**	**16,000**
Total			**163,300**		**642,159**		**735,857**	**1,578,697**

26.3 GEOTECHNICAL STUDIES

Geotechnical sampling and detailed core logging should be conducted in conjunction with any drillcore activities to build a current rock mechanics database. Some of this work is already underway as part of the 2012 drilling programs.

Full investigations of the foundation materials around the processing and surface facilities areas as well as the stockpile and impoundment areas are required. The pitwall and dump design parameters are unknown at this point and should be adequately characterized as further studies proceed. Anecdotal information was used in this design study using best practices, although site construction will require further study.

26.4 WATER SUPPLY – HYDROLOGY

Additional work on the property pertaining to hydrological characterization should continue. This work has already begun and data from these efforts will be used in developing permit applications. Well completions and pump tests to define groundwater characteristics will be required prior to beginning further levels of project evaluation.

In addition to characterizing the hydrologic conditions of the project site, additional investigation has been completed to identify sources of water required for the project. Supply water studies completed by previous investigators were reviewed, and preliminary field studies were conducted to confirm the results of these earlier findings.

A water supply plan has been developed for the project. It includes obtaining approvals from the State for diverting groundwater. Aquifer testing needs to be completed to confirm the estimated productivity of the water bearing strata adjacent to the project in Wah Wah Valley. Groundwater supply wells can be developed within the boundaries of SITLA controlled land. Additionally, the presence of several large (16in diameter) existing groundwater wells has been confirmed. The development of new water supply wells for a project of this size can be significant. In order to minimize these costs, PRC will continue to explore approval to use the wells located off of SITLA controlled land. Permits and infrastructure to connect these wells to the project will still need to be obtained and constructed. Aquifer testing and developing new supply wells for the project was estimated to cost between $2.5 and $3.0M. Obtaining approval to use existing wells in the area would reduce this cost significantly even if additional permitting and infrastructure is required.

26.5 PERMITTING

Norwest has developed a permitting plan for PRC for the project. The plan includes a schedule to obtain all of the major permits (mining, air quality, groundwater discharge, etc.) required prior to start-up. Some preliminary field studies have been conducted, and monitoring of baseline air quality is underway. Environmental studies, including an expansion of the hydrologic investigation program required for these major permits are planned for the 2013 field season. Collection of air quality baseline data will continue through the 2013 field season as well. Once the field studies are completed, permit applications will be developed and submitted to the appropriate agencies for action. A permitting budget of $1.5-$2.0M has been submitted for the work. Ramp-up to begin this work will occur in Q1, 2013 and cover approximately 2 years to complete.

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: U.S. 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.

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.

Cunningham, C.G., Rye, R.O., Steven, T.A., and Mehnert, H.H. (1984) Origins and exploration significance of replacement and vein type Alunite deposits in the Marysvale volcanic field, west-central Utah: Economic Geology, v. 79, p. 50–72.

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.

Earth Sciences, Inc. (1975) Wah Wah Valley Alunite plant test water wells: Unpublished company report, 6 p. and appendices.

Earth Sciences, Inc. (1989) The NG Alunite Property of Earth Sciences, Inc., Beaver County, Utah: Unpublished company report, 25 p.

First Wind – Milford (2011) Welcome to Milford wind: Online http://www.milfordwing.com/milford/, accessed 02/24/2011.

Garrett, D.E. (1996) Potash Deposits, Processing, Properties, and Uses: London, UK, Chapman and Hill, 735 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.

Green, R.E. (1975) Summary report on the PFW Alunite property Iron County, Utah: Unpublished report for Earth Sciences, Inc.: 14 p., appendices and plates.

Green, R.E., and Bauer, C.W. (1973) Evaluation of the tonnage and grade potential of the SX Alunite deposit, Beaver and Iron Counties, Utah: Unpublished report for Earth Sciences, Inc., 16 p., appendices, and plates.

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.

Hintze, L.F., Grant, S.K., Weaver, C.L., and Best, M.G., 1994, Geologic map of the Blue Mountain-Lund area, Beaver and Iron Counties, Utah: U.S. Geological Survey Map I-2361, scale 1:50,000.

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.

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.

Holly Corp. (2011) UNEV Pipeline, LLC.: Online http://www.unevpipeline.com, accessed 02/24/2011.

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.

Johnson, T.C., and Green, R.E. (1975) Summary report on the BCM Alunite Property Beaver County, Utah: Unpublished report for Earth Sciences, Inc., 9 p., appendices and plates

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.

Lewis, W.C., Albrecht, S.L., and Logan, R.F. (1974a) Alunite–A Socio-economic Study: Unpublished report prepared by Lewis and Associates for Earth Sciences, Inc., 187 p.

Lewis, W.C., Albrecht, S.L., and Logan, R.F. (1974b) Alunite–A Socio-economic Study: Appendix D: Unpublished report prepared by Lewis and Associates for Earth Sciences, Inc., 31 p.

Lindsey, D.A., and Osmonson, L.M. (1978) Mineral potential of altered rocks near Blawn Mountain, Wah Wah Range, Utah: U.S. Geological Survey Open-File Report 78-114, 18 p.

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, U.S. Patent 4,031,182, filed March 24, 1976, and issued June 21, 1977.

Mason, J.L. (1998) Ground-water hydrology and simulated effects of development in the Milford area, an arid basin in southwestern Utah: U.S. Geological Survey Professional Paper 1409-G.

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.

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.

Mower, R.W., and Cordova, R.M. (1974) Water Resources of the Milford area, Utah, with Emphasis on Groundwater: Utah Department of Natural Resources Technical Publication 43.

Myertons, Ralph (May 27, 1976) Flotation of Alunite and Alumina, prepared for Alumet Company, Golden, CO.

Nackowski, M.P., Brown, S.D., and Botbol, J.M. (1962) Alunite annotated bibliography: University of Utah Engineering Experiment Station Bulletin 124, 103 p.

Native Plants, Inc. (1975a) Conceptual Revegetation Plan for the Proposed Alunite Mine/Processing Plant Complex in Western Beaver County, Utah: Unpublished report prepared for Earth Sciences, Inc., 21 p.

Native Plants, Inc. (1975b) Theoretical Approaches to Revegetation for the Proposed Alunite Mine/Processing Plant Complex in Western Beaver County, Utah: Unpublished report prepared for Earth Sciences, Inc., 12 p.

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.

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 (1974) Summary report of results of development work on the NG Alunite property, southern Wah Wah Mountains, Beaver County, Utah: Unpublished Earth Sciences, Inc. report, 33 p. and appendices.

Perry, Harry (1977_ Mining plan for NG Alunite area C – first five years: Unpublished Earth Sciences, Inc. report, 122 p.

Prud'homme, Michel, and Krukowski, S.T. (2007) Potash, in Kogel, J.E., Trivedi, N.C., Barker, J.M., Krukowski, S.T., editors, Industrial Rocks and Minerals, 7th edition: Littleton, Society of Mining, Metallurgy, and Exploration, Inc., p. 723-741.

Riley, J.M., Edlund, V.E., and Nissen, W.I. (1983) Shaft furnace reduction-oxidation roasting of pelletized Alunite dust: U.S. Bureau of Mines Report of Investigation 8821, 21 p.

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.

Rye, R.O., Bethke, P.M., and Wasserman, M.D. (1992) The stable isotope geochemistry of acid sulfate alteration: Economic Geology, v. 87, no. 2, p. 225-262.

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.

Sinnott, R. K. (2005) Coulson & Richardson, Chemical Engineering, Vol. 6: Chemical Engineering Design, 4th ed. Elsevier.

Sisselman, R. (Oct 1975) Engineering and Mining Journal (E&MJ), Vol. 176, N0. 10, pp. 79-84.

Society for Mining, Metallurgy, and Exploration, Inc. (2005) The SME guide for reporting exploration results, mineral resources, and mineral reserves: Online, http://books.smenet.org/sec_recomm_pdfs/Appendix3.pdf, accessed July 2011.

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].

Stringham, Bronson (1963) Hydrothermal alteration in the southeast part of the Frisco quadrangle, Beaver Cnty, UT: Utah Geological and Mineralogical Survey Special Studies 4, 21 p.

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.

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.

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.

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.

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) 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, 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) 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. Bureau of Land Management (2011) Cedar City Field Office Resource Management Plan, Transportation Map: Online at, https://www.blm.gov/epl-front-office/projects/lup/7100/17351/17551/CCFO_Transportation_60x70L.pdf, accessed 07/05/2011.

U.S. Bureau of Land Management (2011) 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) 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. Department of Energy (2011) West-wide Energy Corridor Programmatic Environmental Impact Statement: Online at: http://corridoreis.anl.gov/, accessed July 6, 2011.

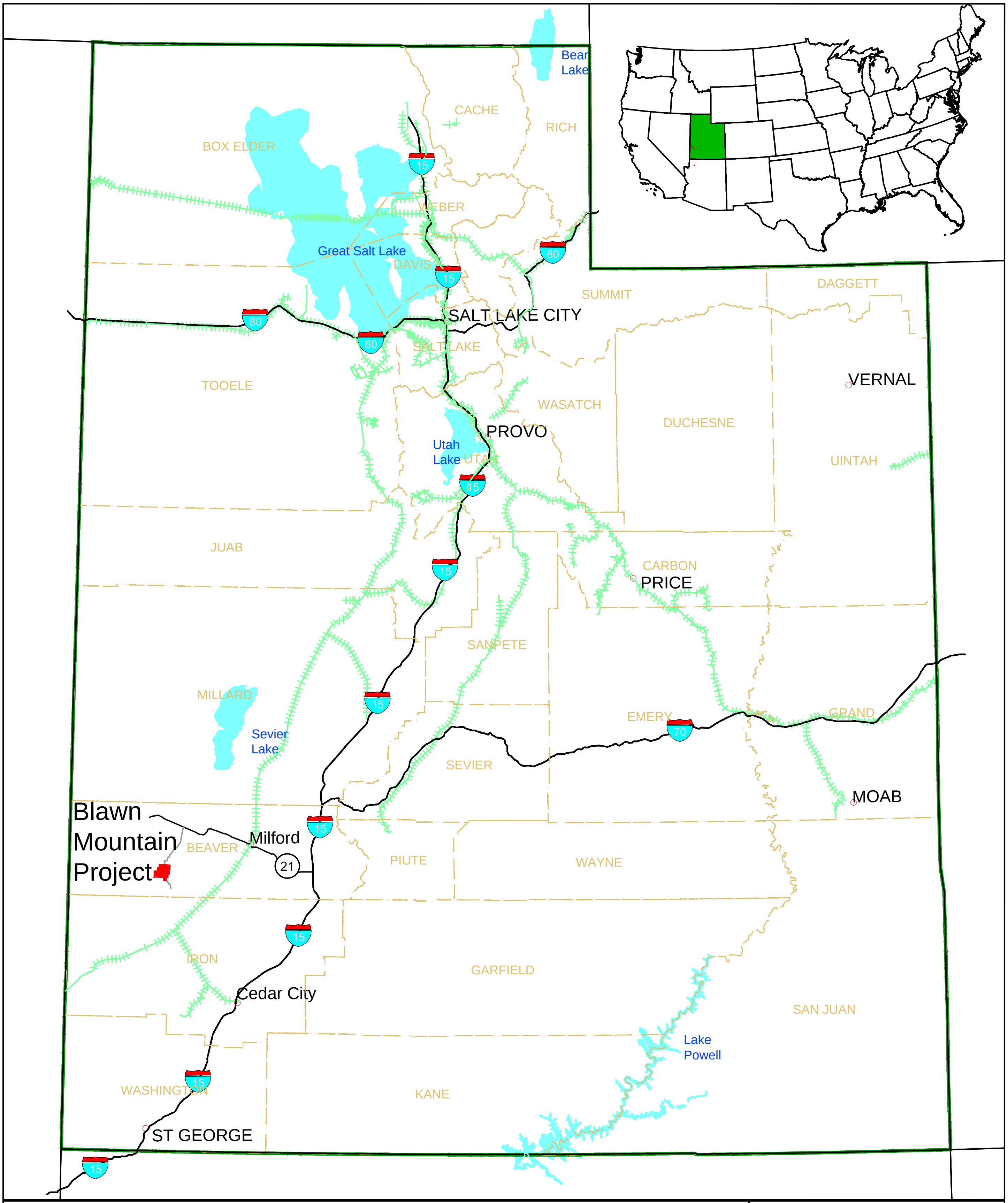
Walker, W.W. (1972) Report of results phase III exploration program on the NG Alunite property Beaver County, Utah: Unpublished report for National -Southwire Aluminum Co., and Earth Sciences, Inc., 123 p., appendices, and plates.

Walker, W.W. (1973) Evaluation of the tonnage and grade of the White Mountain Alunite deposit, Beaver County, Utah: Unpublished report for Earth Sciences, Inc., 21 p., appendices and plates.

WestConnect (2011, 2010) Draft WestConnect reports: Online, http://westconnect.com , accessed 02/25/2011.

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



LEGEND

— Freeway
○ City or Town
🟥 Blawn Mountain Project
— Railroad
— Counties
● 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: 418-3\Loc
SCALE: As Shown

NORWEST
CORPORATION

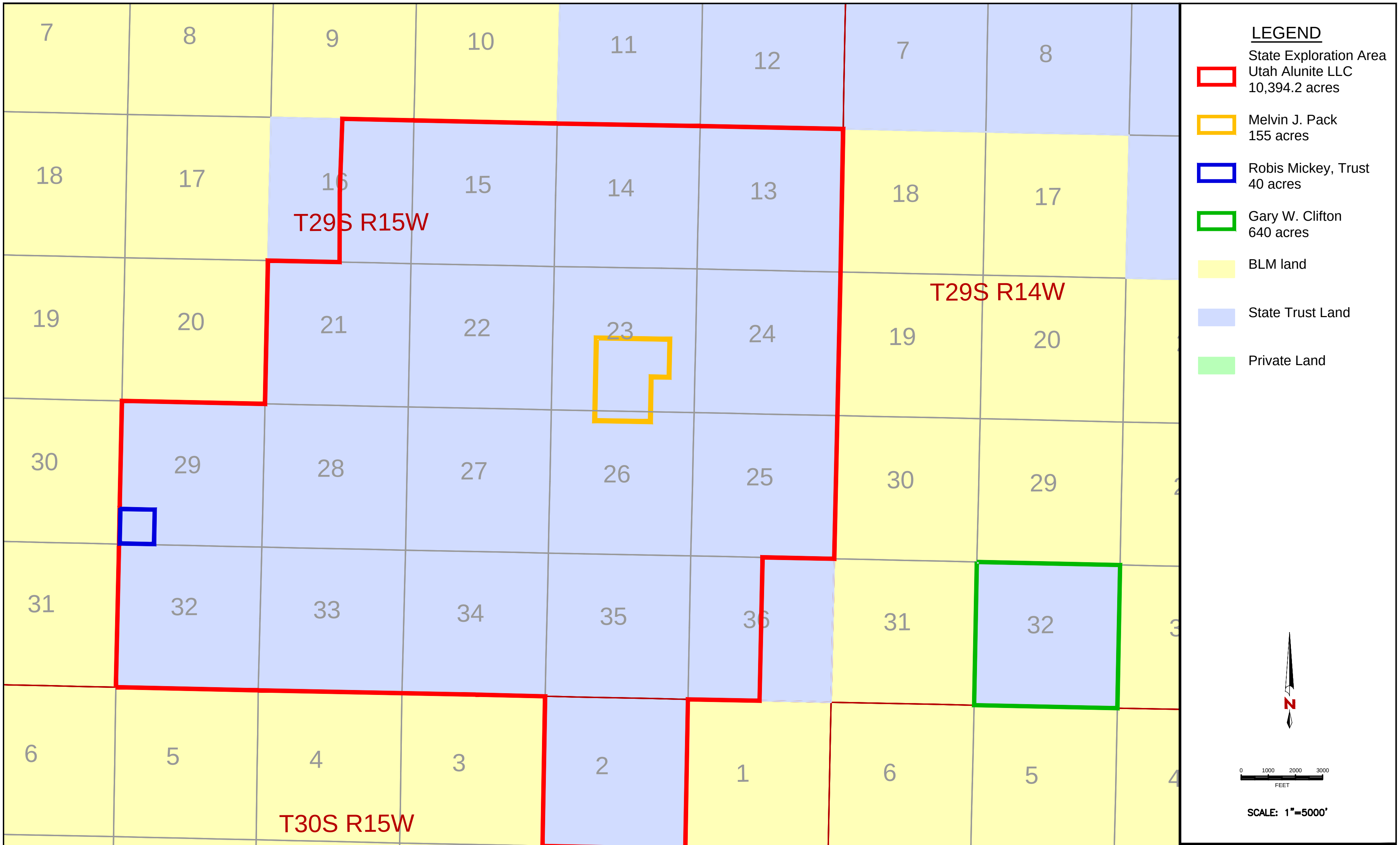


FIGURE 4.2

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

EXPLORATION/OPTION
AREA LOCATION

DATE: 06/04/2012
FILE: 418-3\owners

NORWEST CORPORATION

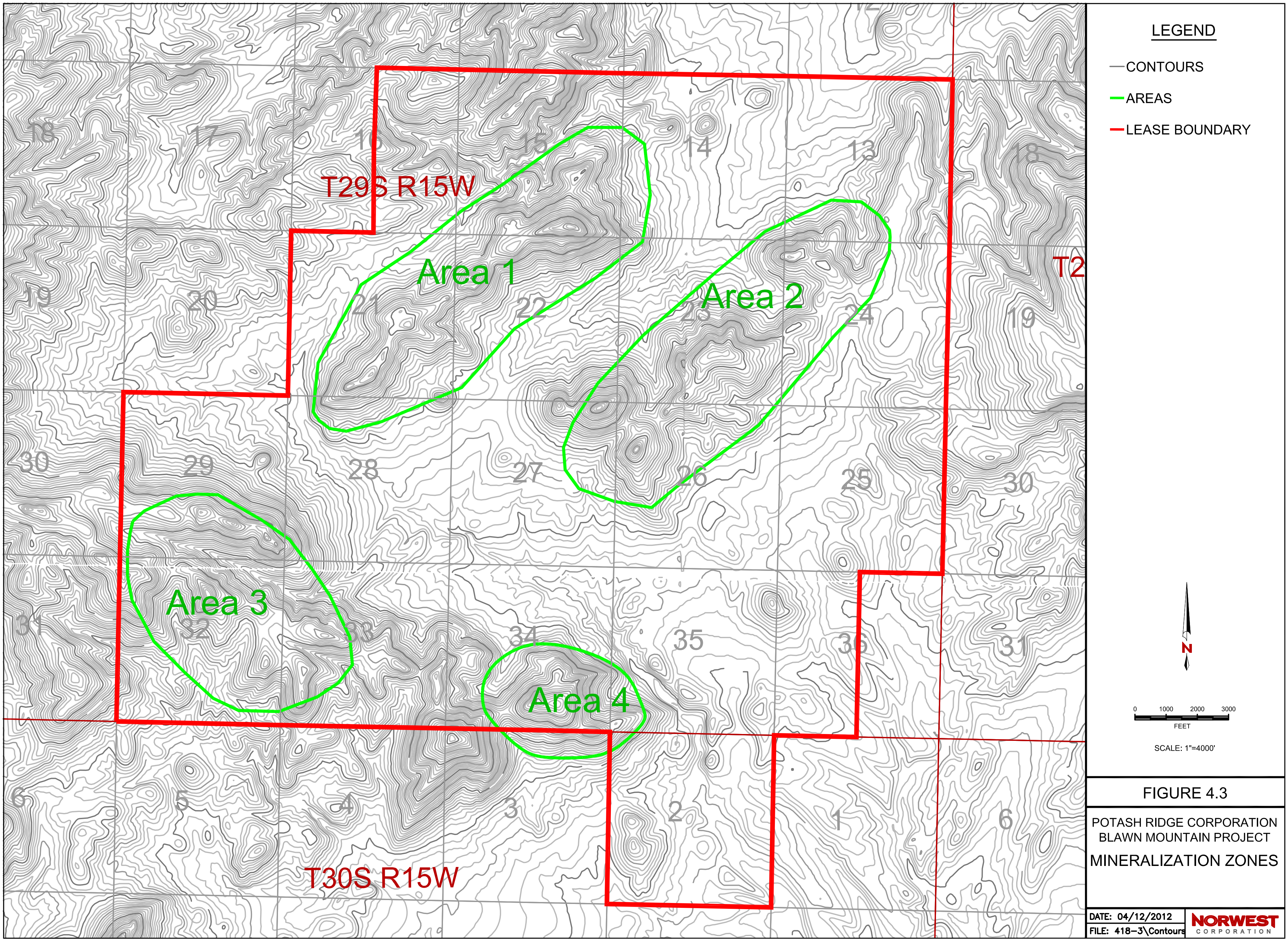


FIGURE 4.3

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT
MINERALIZATION ZONES

DATE: 04/12/2012
FILE: 418–3\Contours

NORWEST
CORPORATION

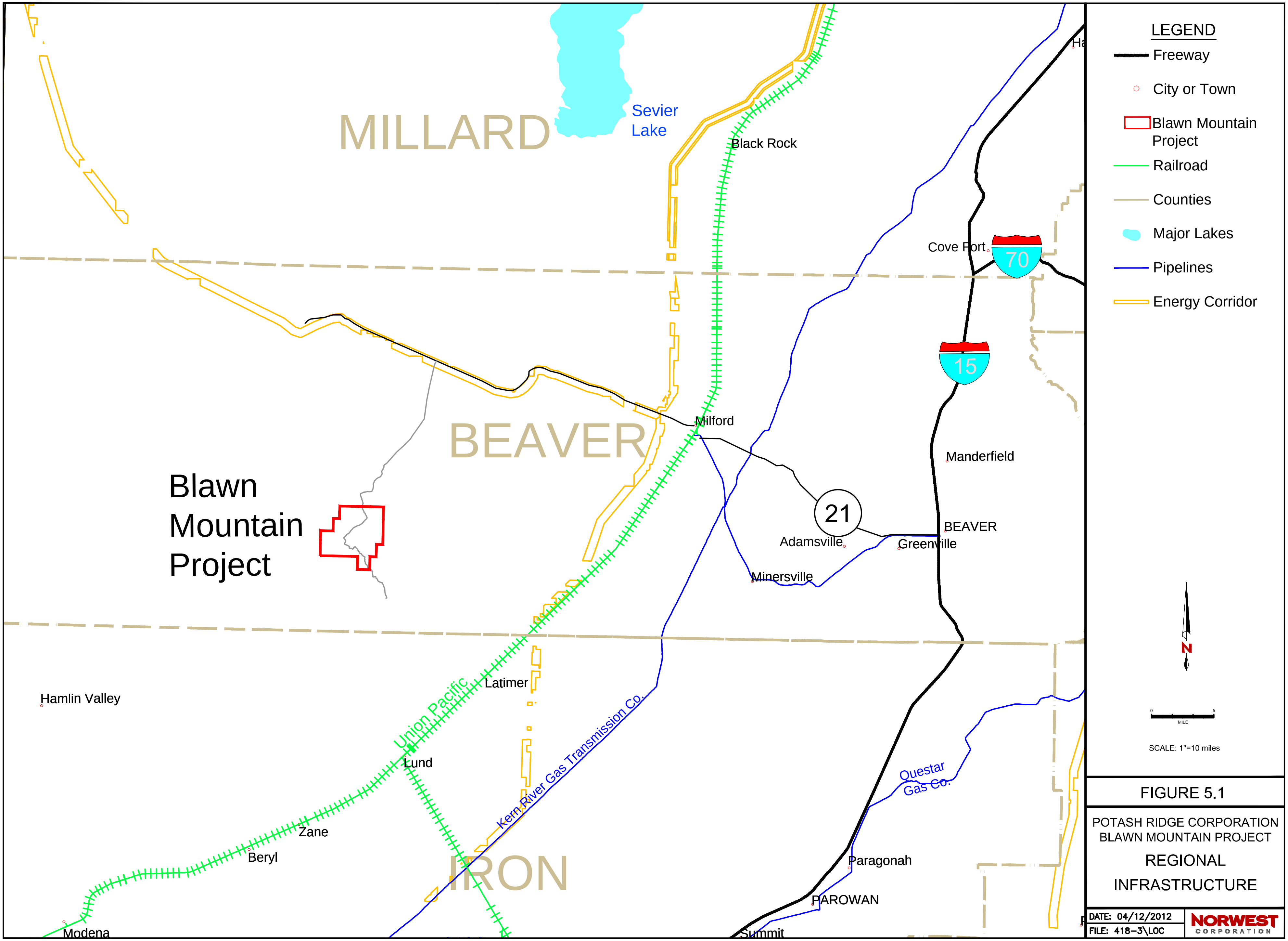


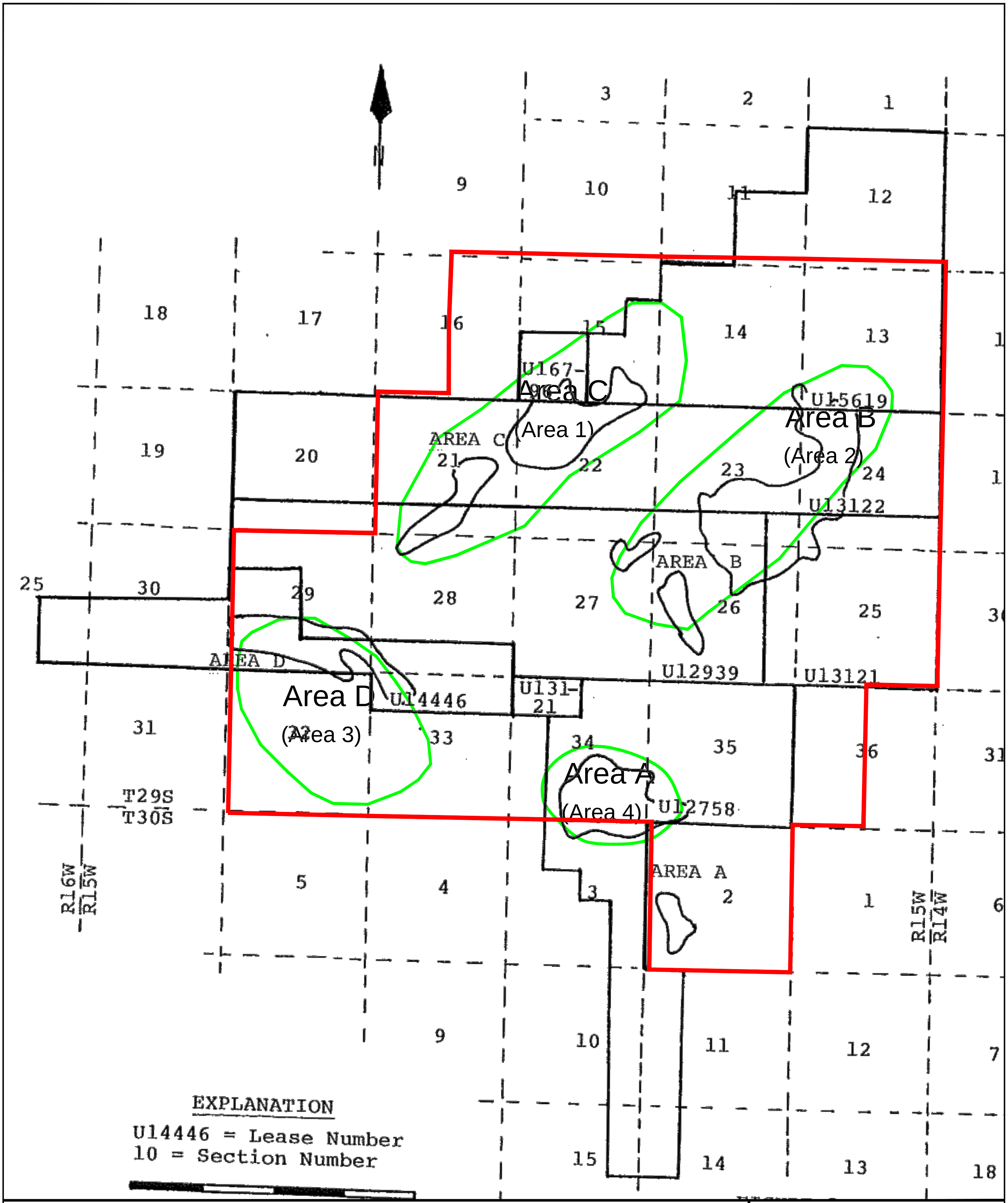
FIGURE 5.1

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

REGIONAL INFRASTRUCTURE

DATE: 04/12/2012
FILE: 418-3\LOC

NORWEST CORPORATION



LEGEND

— Current Blawn Mountain Property

— Former NG Alunite Boundary

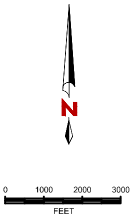


FIGURE 6.1

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

FORMER NG
ALUNITE PROJECT AREAS

DATE: 06/04/2012
FILE:418-3\Contours
SCALE: 1"=5000'
NORWEST CORPORATION

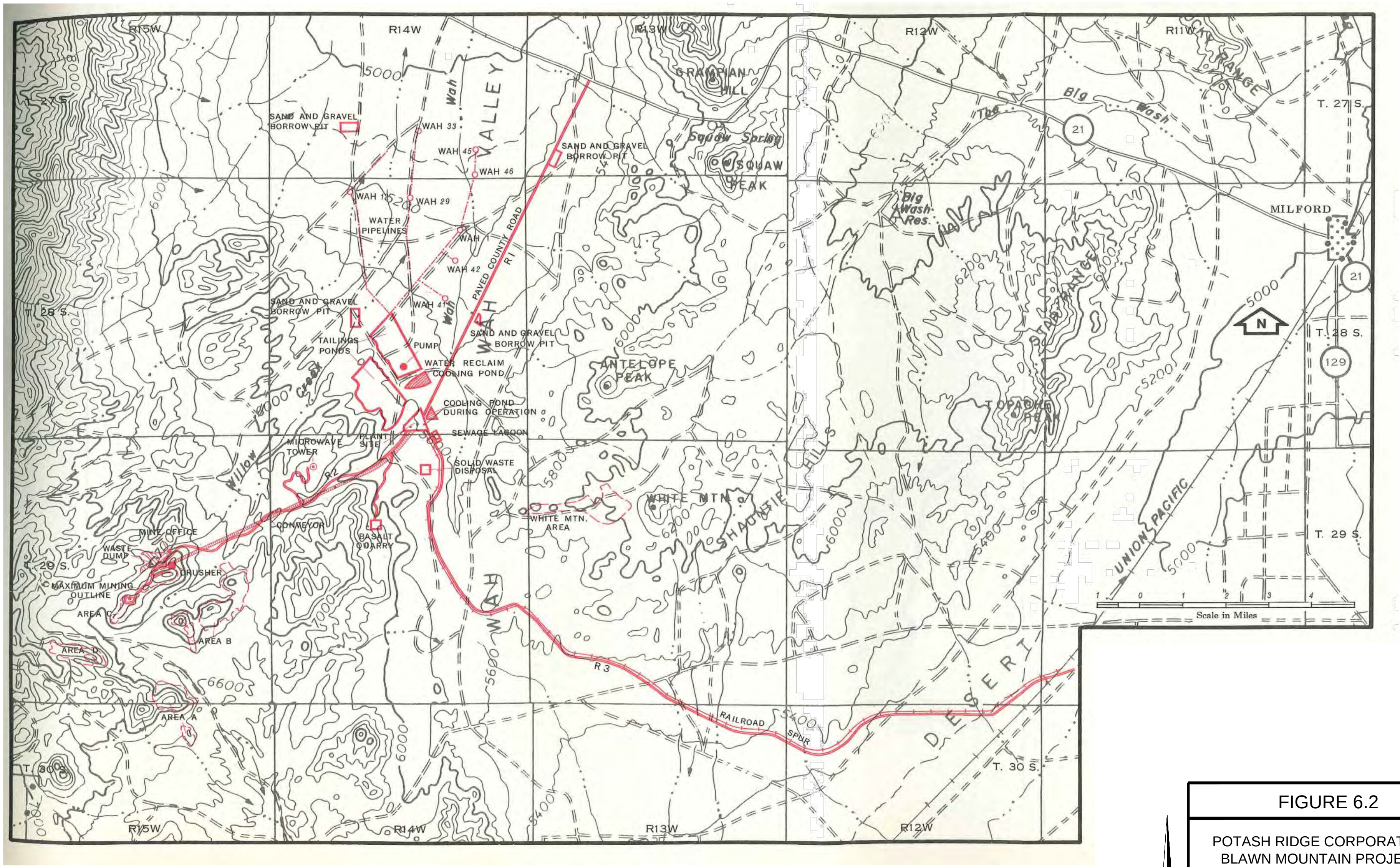
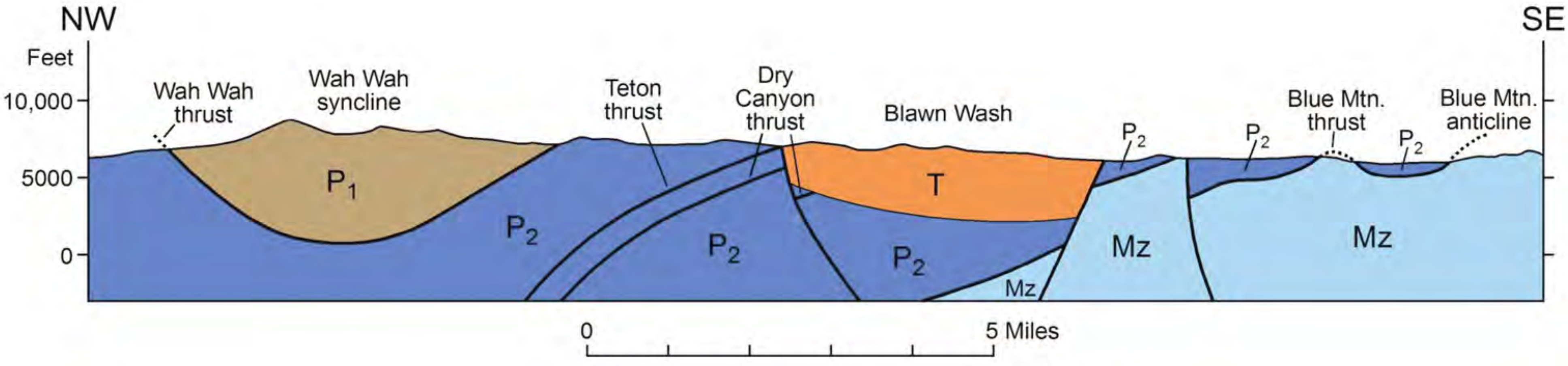


FIGURE 6.2

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

FORMER NG PROPERTY
DEVELOPMENT PLAN

DATE: 04/16/2012
FILE: 418-3FIG 6.2
PROJECT: 418-3
NORWEST CORPORATION



T = Tertiary volcanic rocks

Mz = Mesozoic block beneath the Blue Mountain thrust

P_2 = Upper Paleozoic block between the Blue Mountain and Wah Wah thrusts

P_1 = Lower Paleozoic block above the Wah Wah thrust

Modified from Abbott and others (1983) and Hintze and others (1994).

FIGURE 7.1

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

REGIONAL GEOLOGICAL CROSS SECTION

DATE: 06/04/2012
FILE: 418-3\XSEC
PROJECT: 2XXX
NORWEST CORPORATION

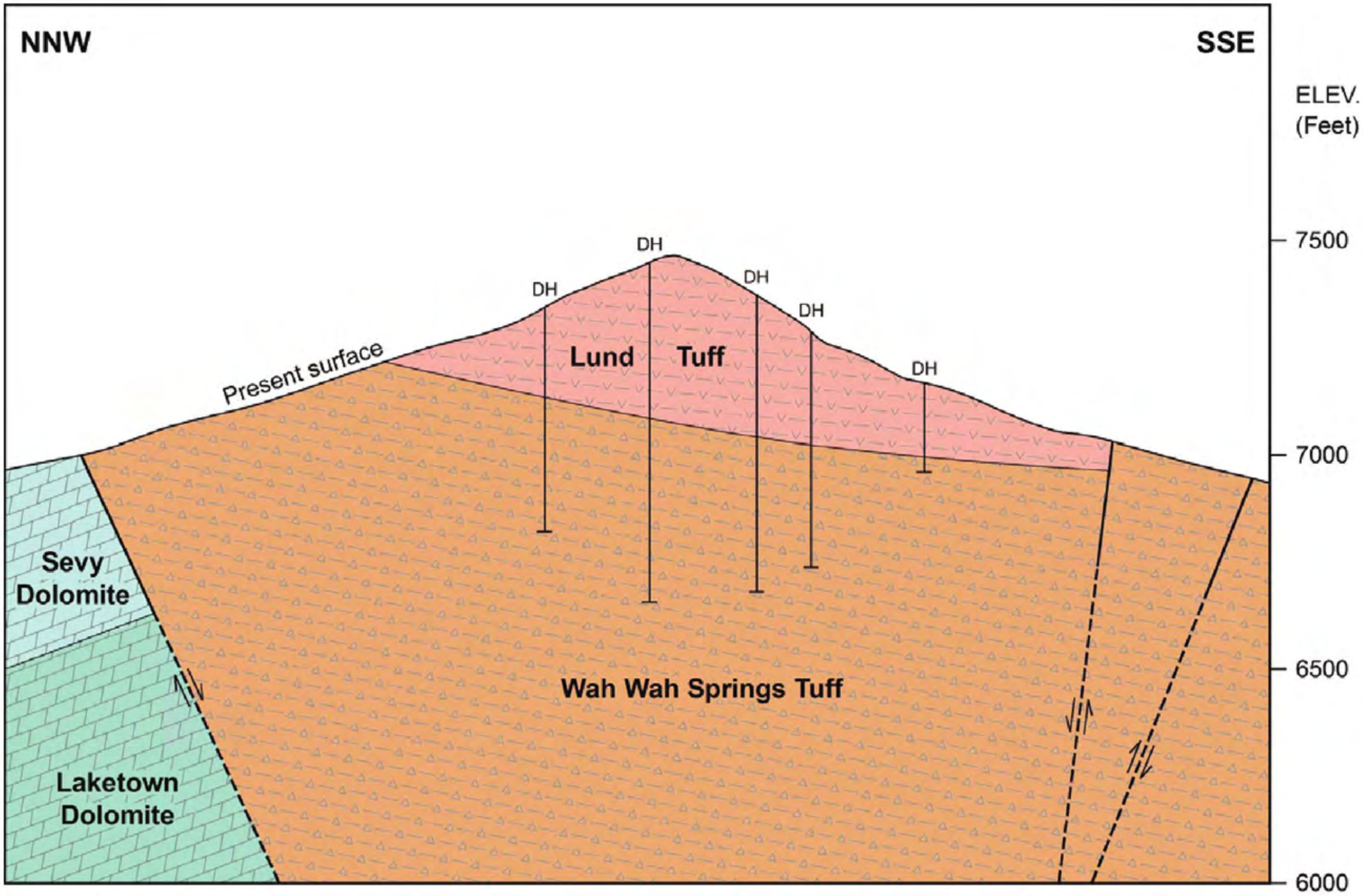


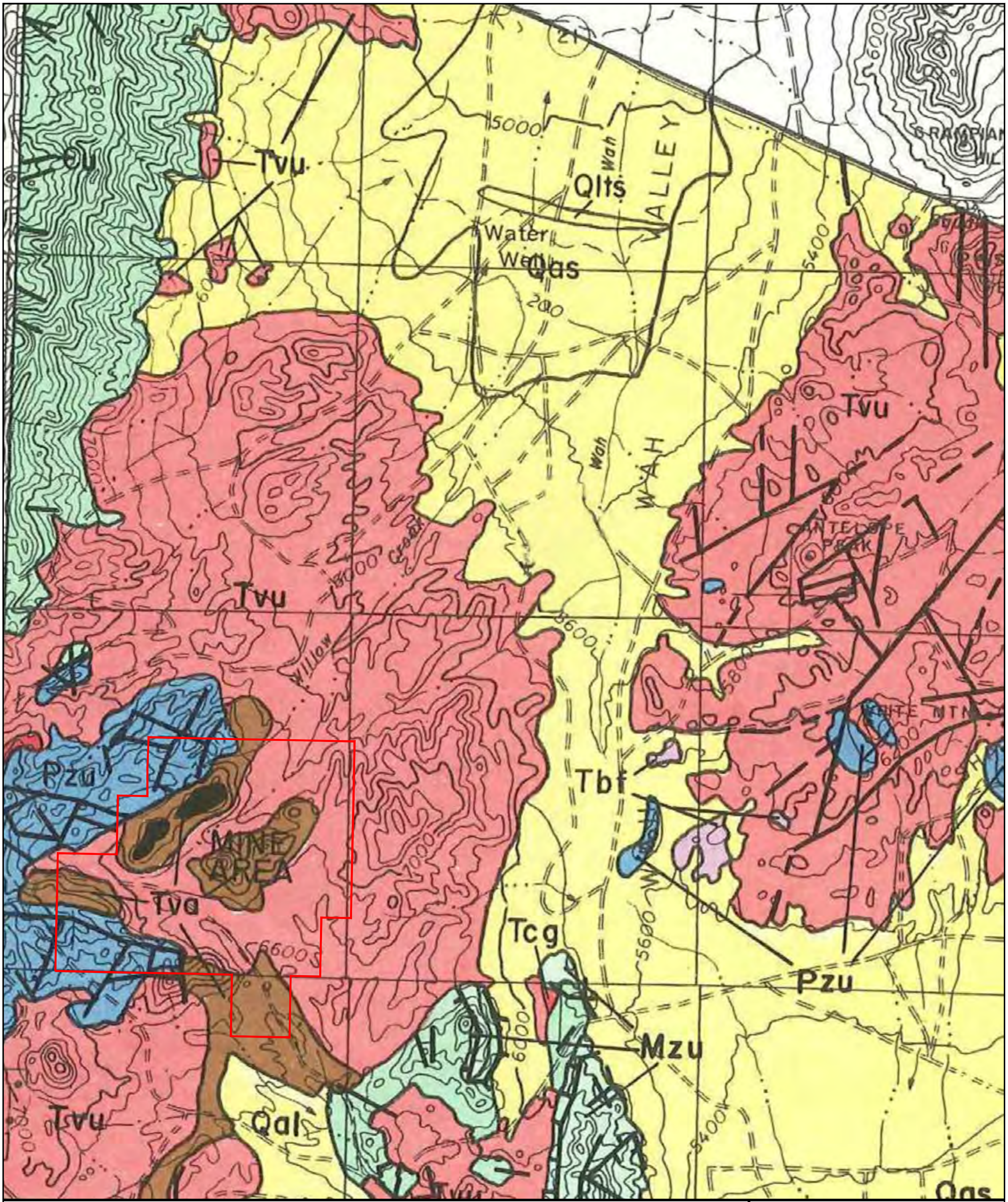
FIGURE 7.2

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

LOCAL GEOLOGICAL
CROSS SECTION

DATE: 06/04/2012
FILE: 418–3\XSEC

NORWEST CORPORATION



LEGEND

SEDIMENTARY		
🟨	Qal, Qas, Qlts	ALLUVIUM AND LAKE DEPOSITS
🟥	Tcg	CONGLOMERATE
🟩	Mzu	MESOZOIC SANDSTONE AND SHALE
🟦	Rzu	PALEOZOIC LIMESTONES, DOLOSTONES AND QUARTZITES
🟩	Cu	CAMBRIAN LIMESTONES AND QUARTZITES

VOLCANIC		
🟫	Tva	ALTERED IGNIMBRITES
🟥	Tvu	UNDIFFERENTIATED VOLCANICS
🟪	Tbf	BASALT FLOWS
—		NORMAL FAULTS
▲▲		THRUST FAULTS

0 5000 10000 15000

FEET

FIGURE 7.3

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

SURFACE
GEOLOGY MAP

N

DATE: 04/11/2012
FILE: 418-3\Loc
SCALE: 1:12000

NORWEST
CORPORATION

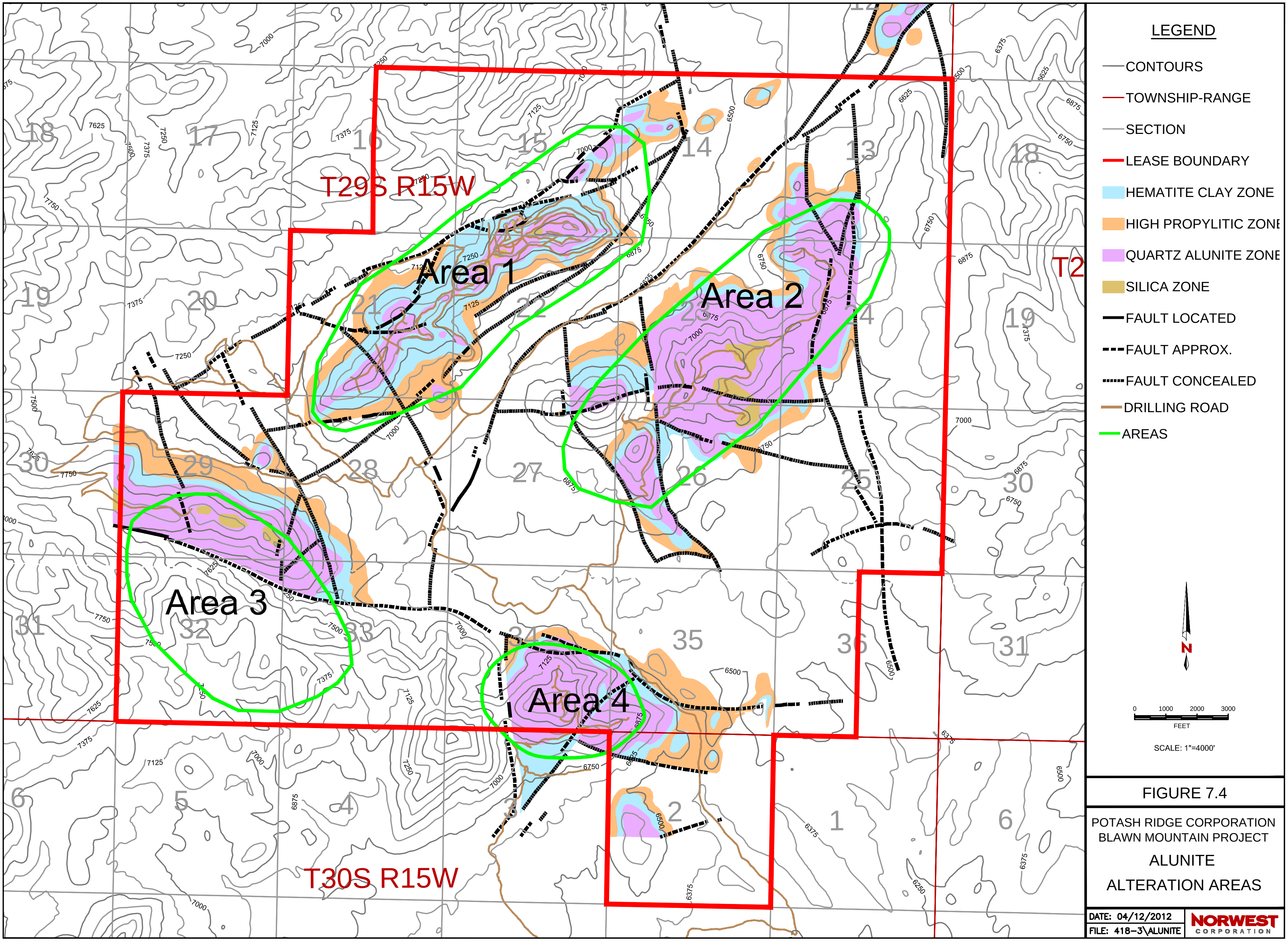


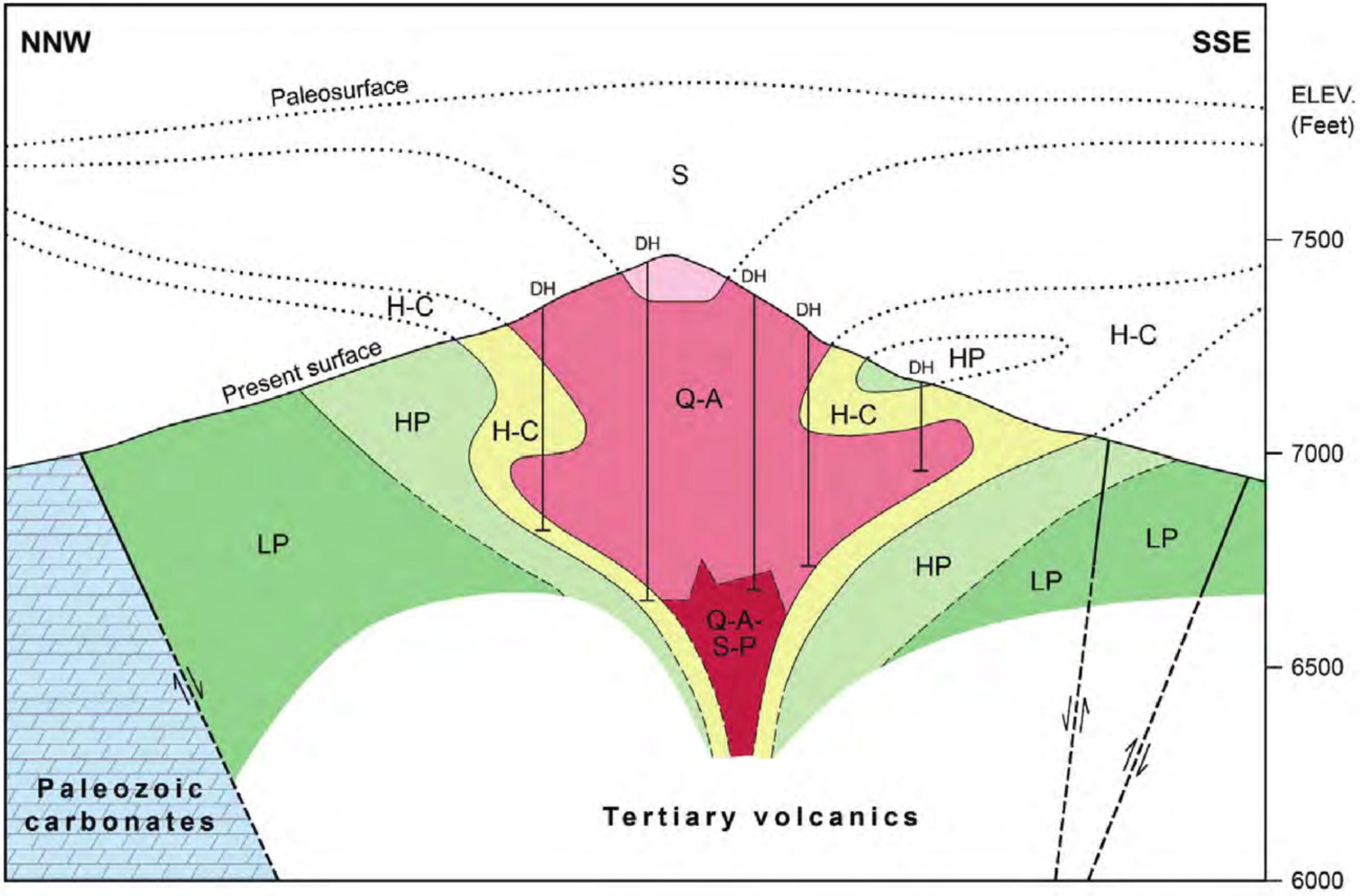
FIGURE 7.4

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

ALUNITE
ALTERATION AREAS

DATE: 04/12/2012
FILE: 418-3\ALUNITE

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

FIGURE 7.5

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

ALUNITE ALTERATION
CROSS SECTION

DATE: 06/04/2012
FILE: 418-3\XSEC

NORWEST
CORPORATION

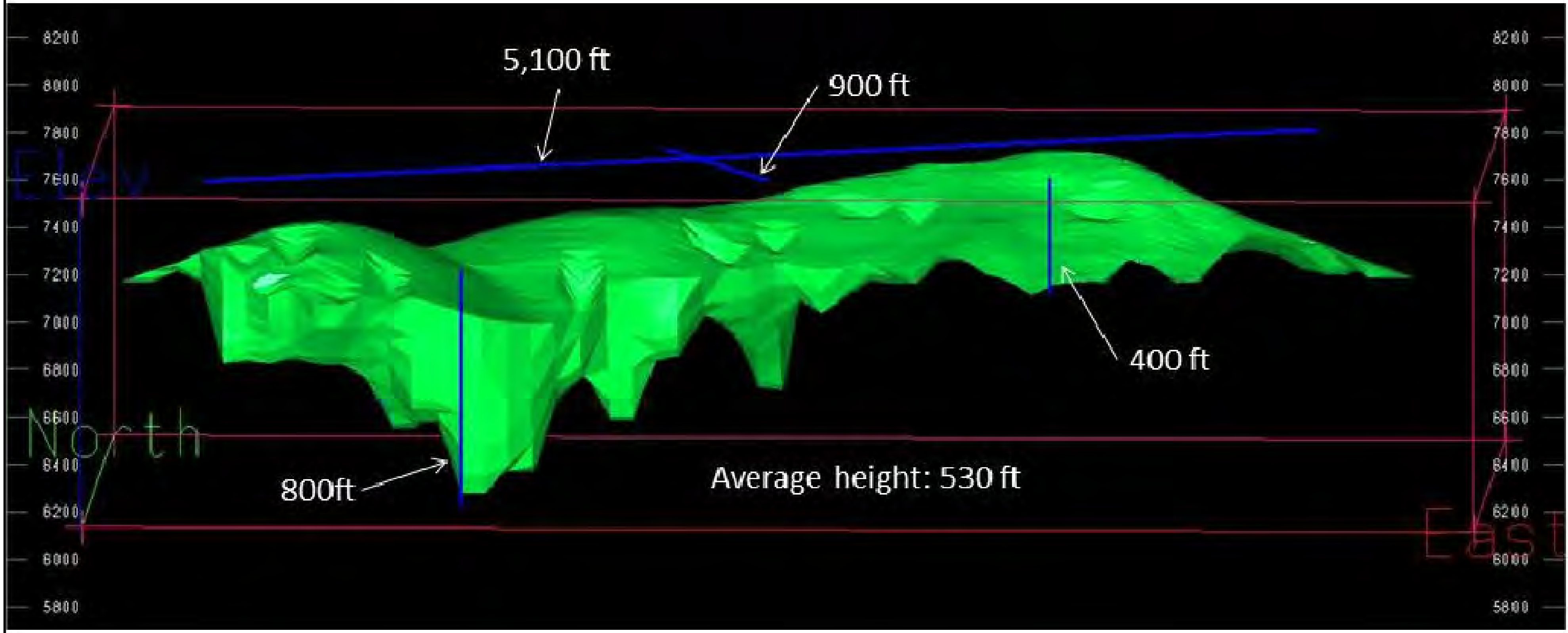


FIGURE 7.6

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

3DBM REPRESENTATION
OF AREA 1 ALUNITE ZONE

DATE: 04/4/2012
FILE: 418-3 figures
SCALE: as noted
NORWEST CORPORATION

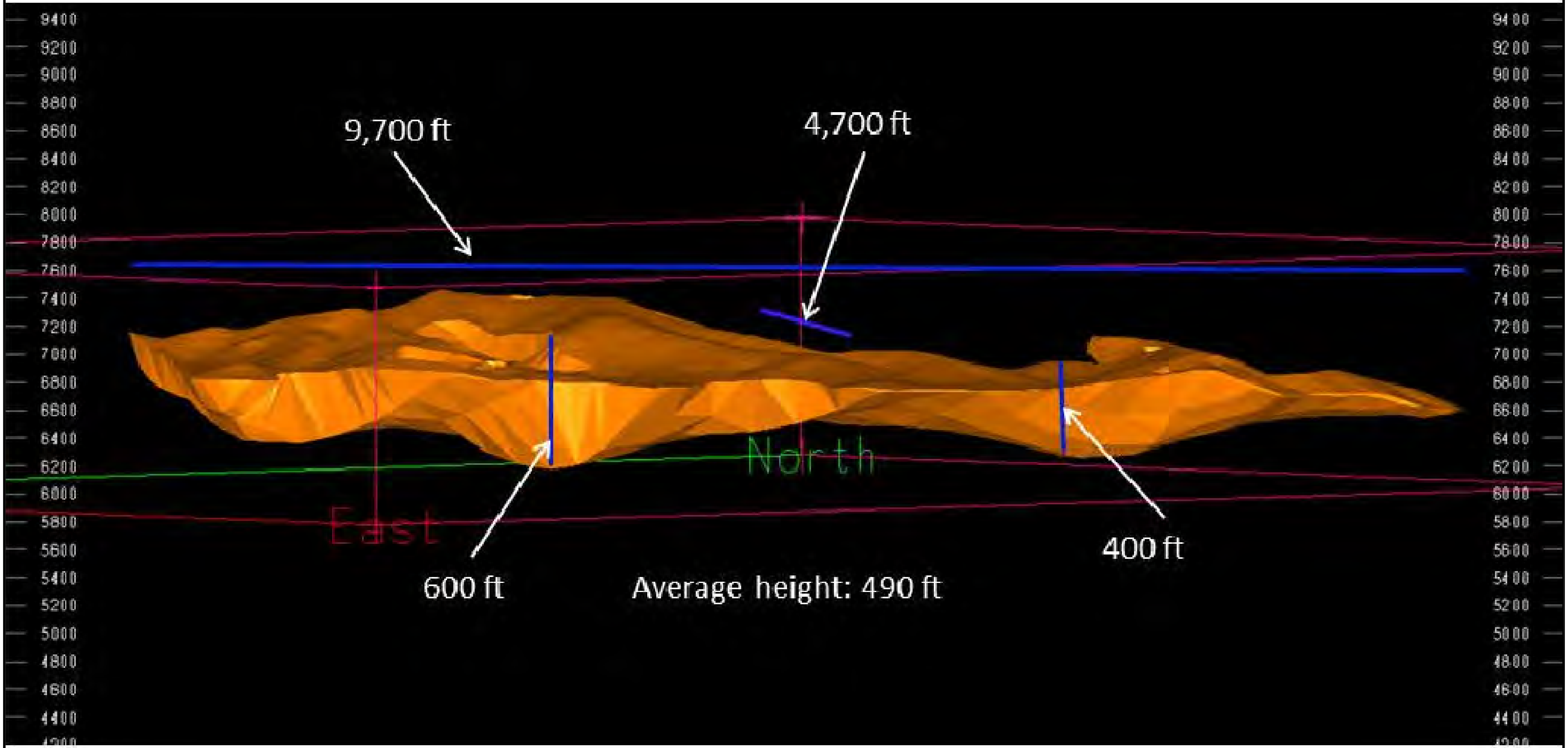


FIGURE 7.7

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

3DBM REPRESENTATION
OF AREA 2 ALUNITE ZONE

DATE: 10/30/2012
FILE: 418-4 figures
SCALE: as noted
NORWEST CORPORATION

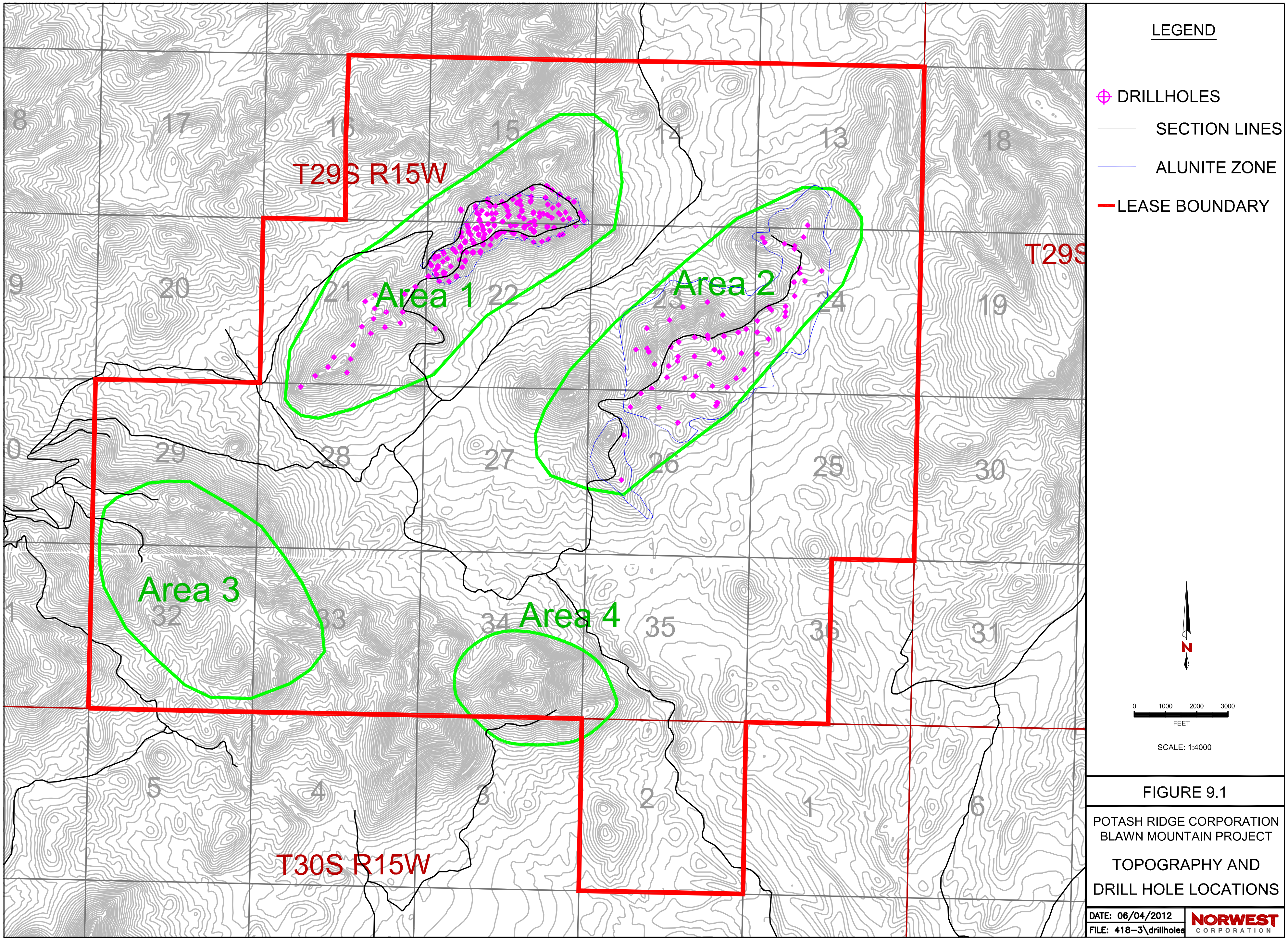


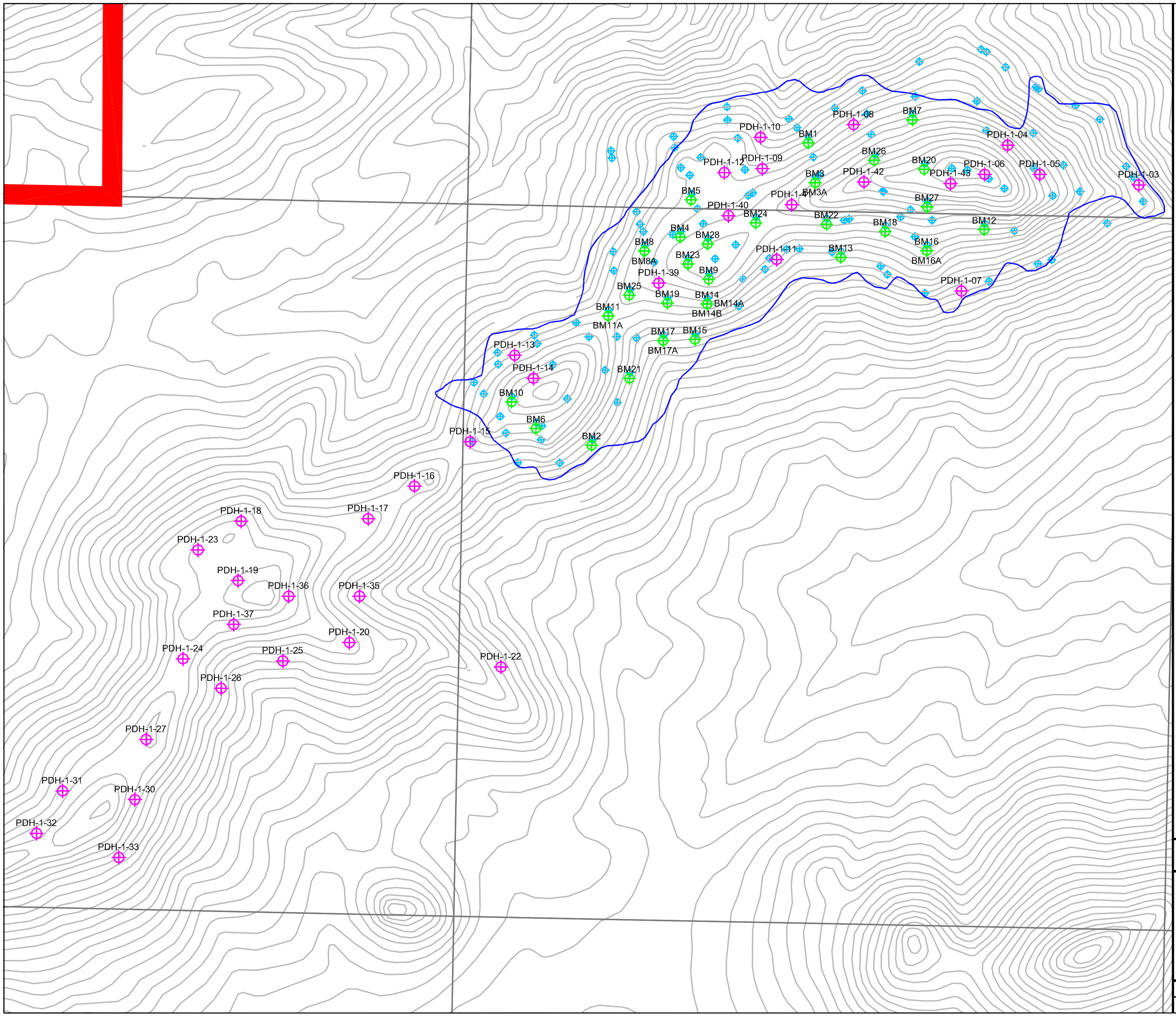
FIGURE 9.1

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

TOPOGRAPHY AND
DRILL HOLE LOCATIONS

DATE: 06/04/2012
FILE: 418-3\drillholes

NORWEST
CORPORATION



LEGEND

⊕ POTASH RIDGE DRILLHOLES (in progress)
⊕ POTASH RIDGE DRILLHOLES
⊕ ESI DRILLHOLES
— SECTION LINES
— ALUNITE ZONE

N

0 200 400 600 800
FEET

SCALE: 1:1000

FIGURE 9.2

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT
AREA 1
ALUNITE ZONE
DRILLHOLES

DATE: 04/12/2012
FILE: 418–3\drillholes

NORWEST
CORPORATION

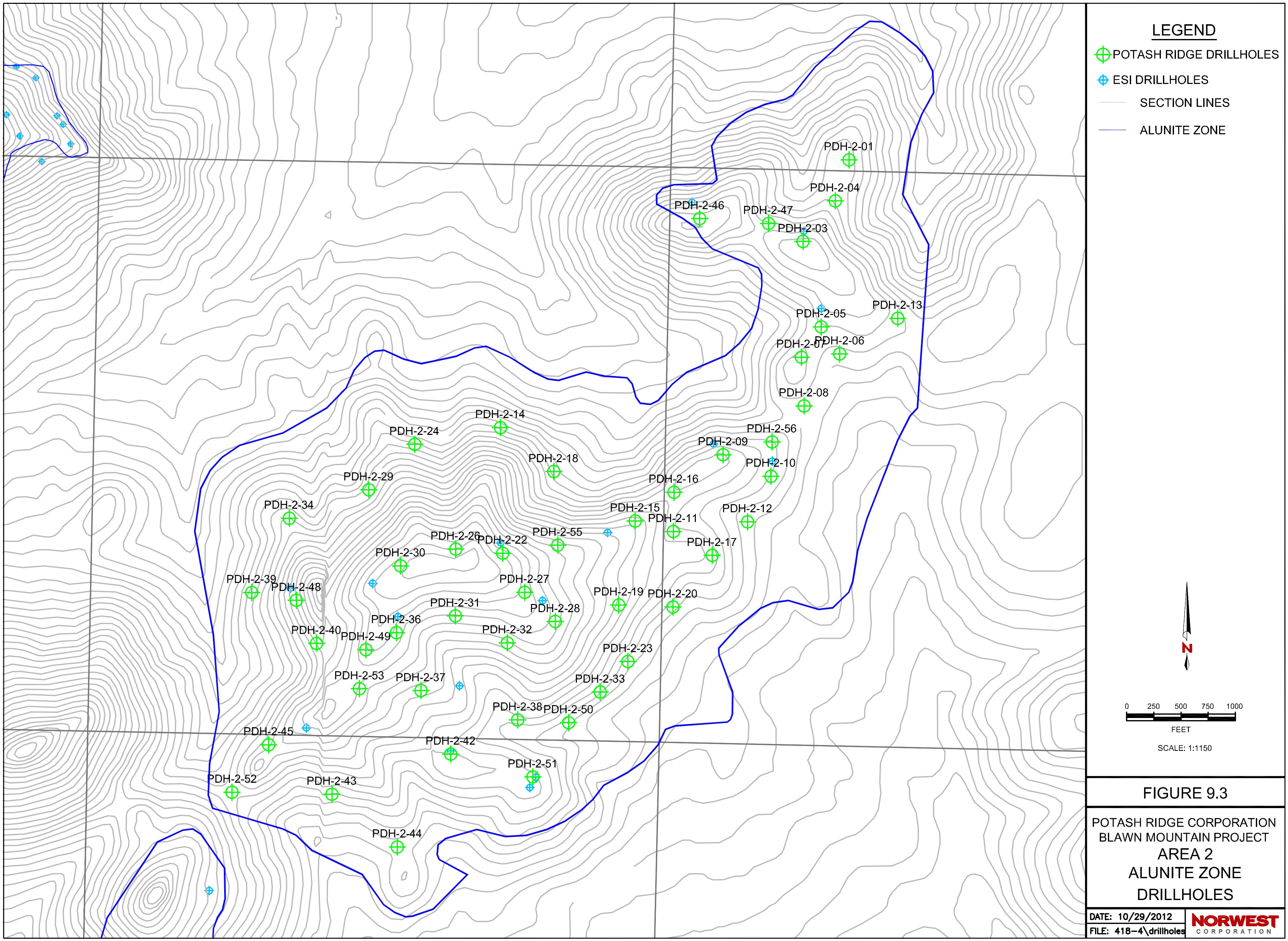
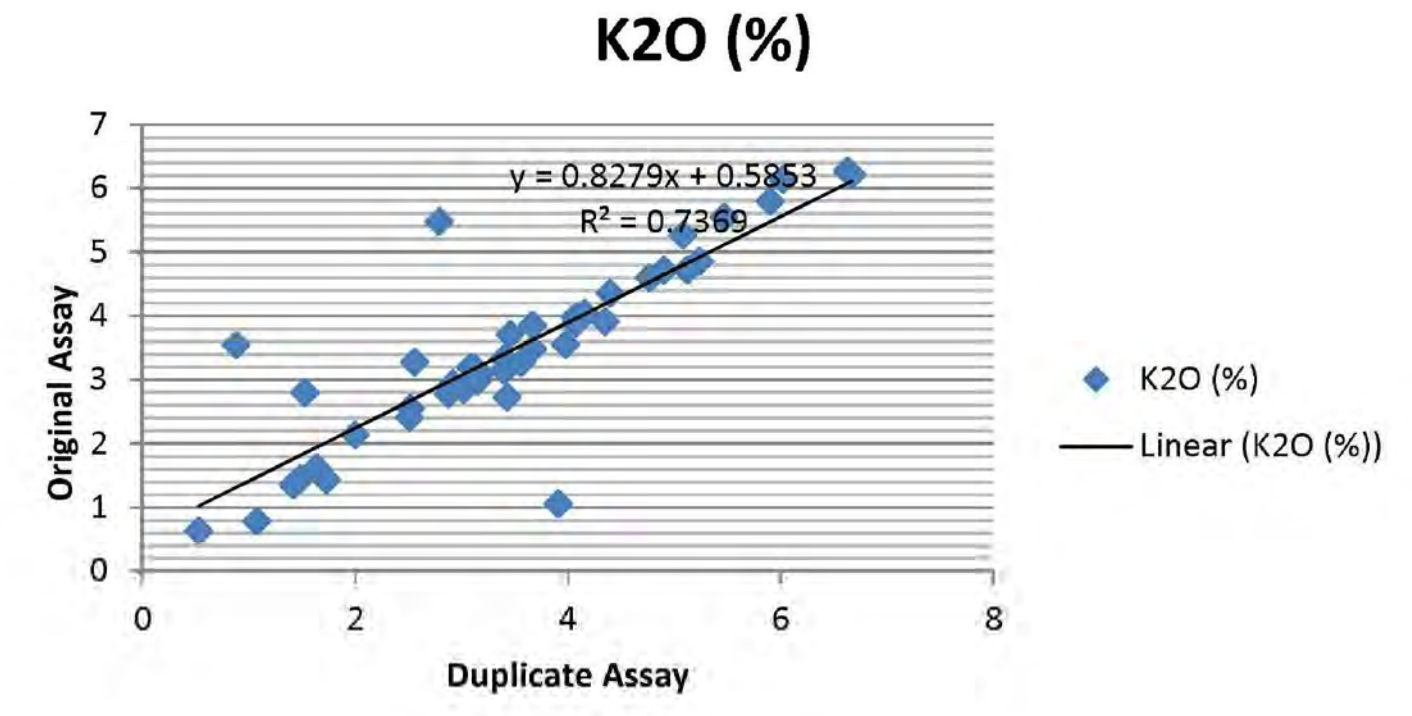


FIGURE 9.3

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT
AREA 2
ALUNITE ZONE
DRILLHOLES

DATE: 10/29/2012
FILE: 418-4\drillholes

NORWEST CORPORATION



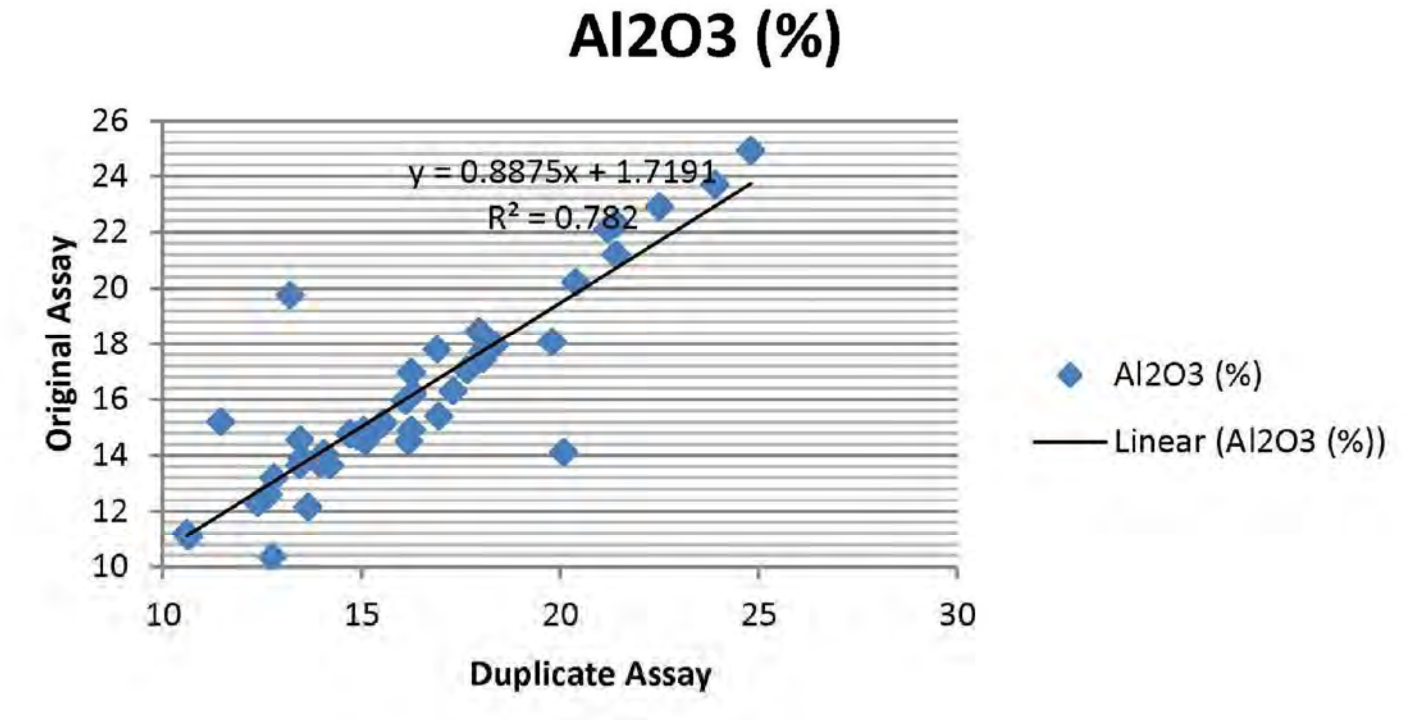


FIGURE 12.1

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

K2O and Al2O3 COMPARISONS FOR ORIGINAL and DUPLICATE ASSAYS - AREA 1

DATE: 04/13/2012
FILE: 418-3

NORWEST CORPORATION

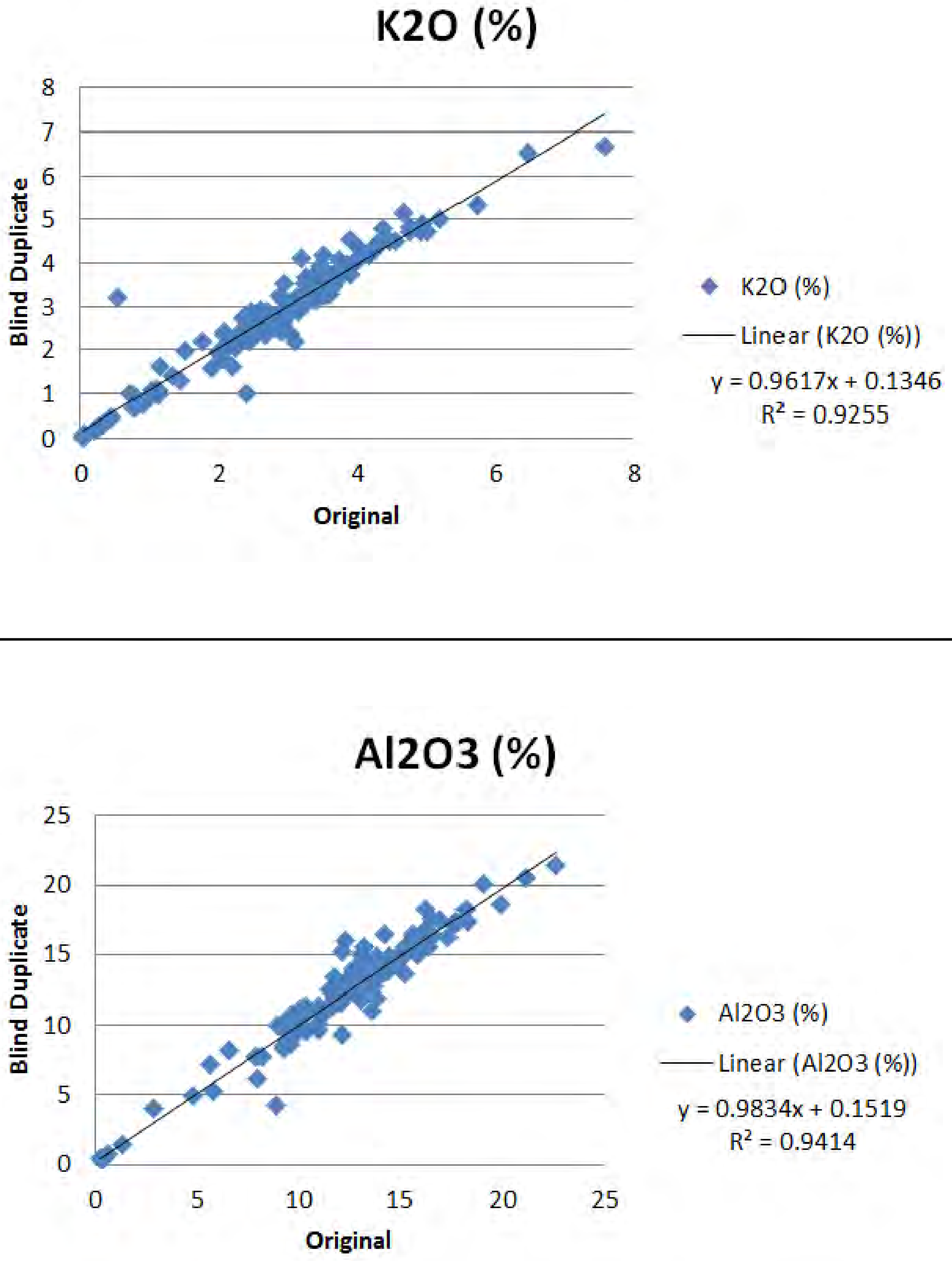


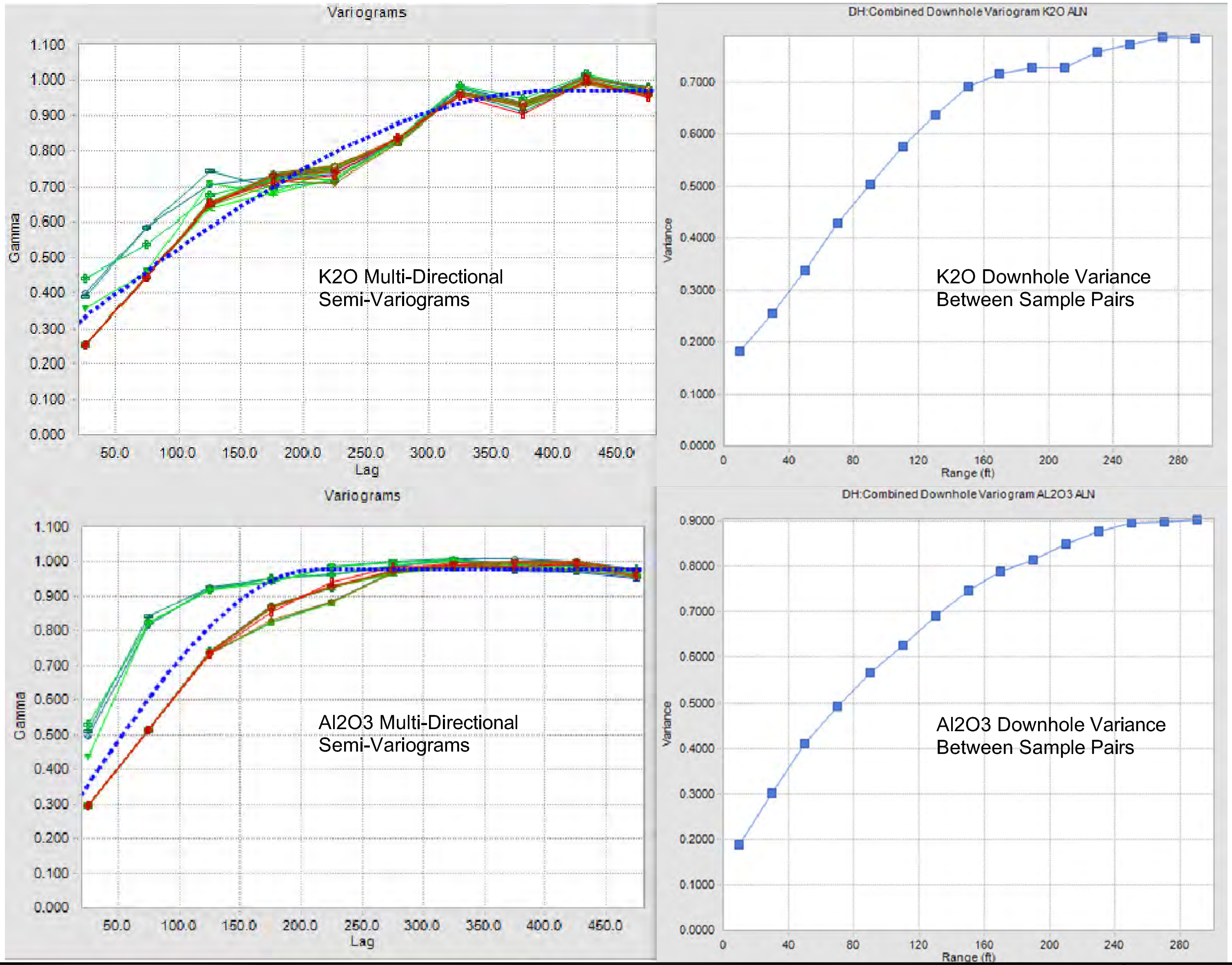
FIGURE 12.2

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

K2O and Al2O3 COMPARISONS
FOR ORIGINAL and DUPLICATE
ASSAYS - AREA 2

DATE: 11/01/2012
FILE: T:/418-4

NORWEST
CORPORATION



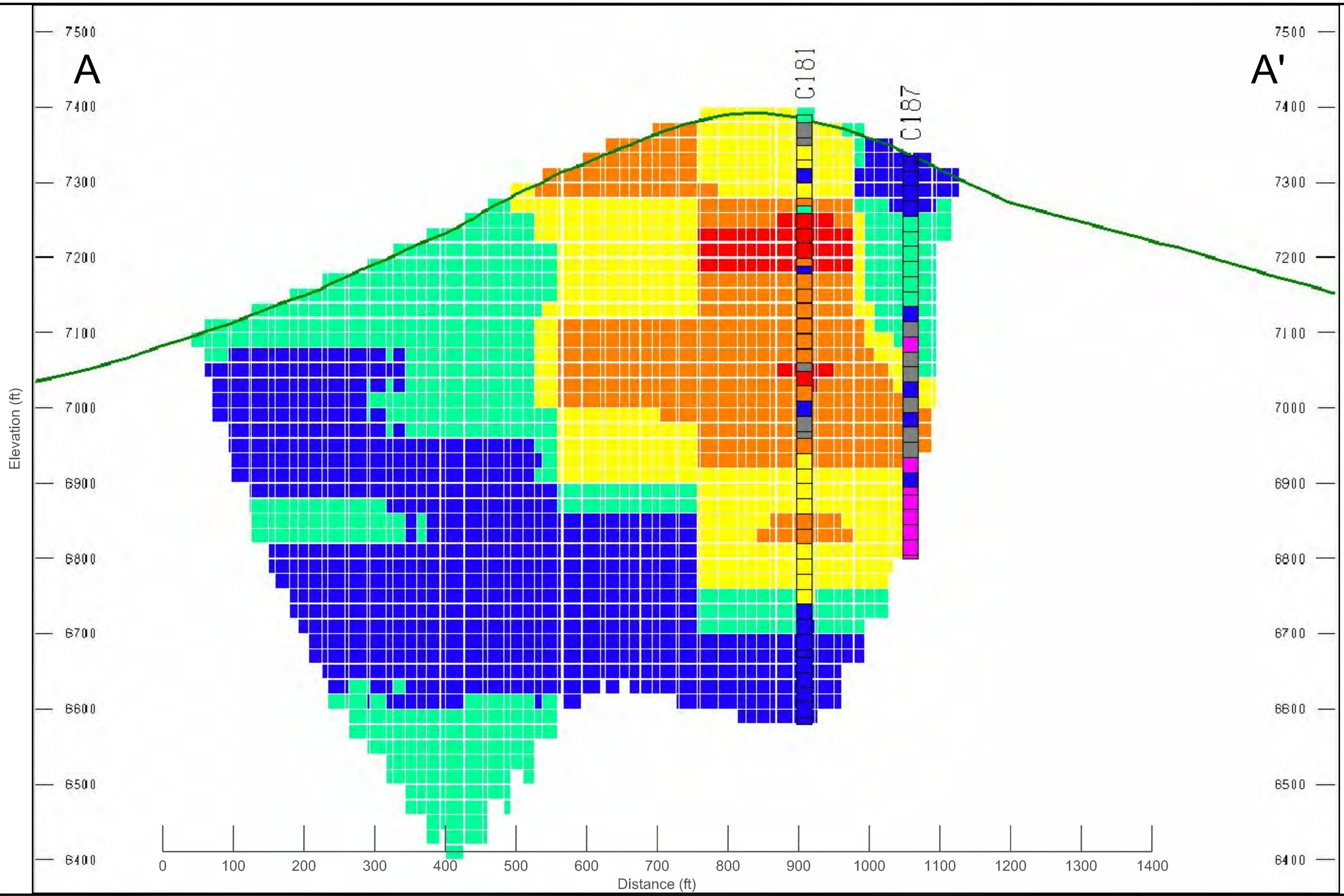
Best Fit Experimental
Semi-Variograms (Spherical Model)

FIGURE14.1

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

VARIOGRAMS OF
ANALYTICAL RESULTS

DATE: 04/4/2012	SCALE:	NORWEST CORPORATION
FILE: 418–3 figures	as noted	



K2O%

⬛	<0
🟪	0
🟦	1
🟩	2
🟨	3
🟧	4
🟥	5

Drillholes displayed can be
up to 200 feet from section

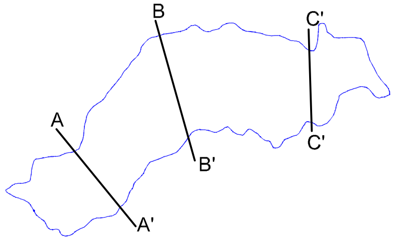


FIGURE 14.2

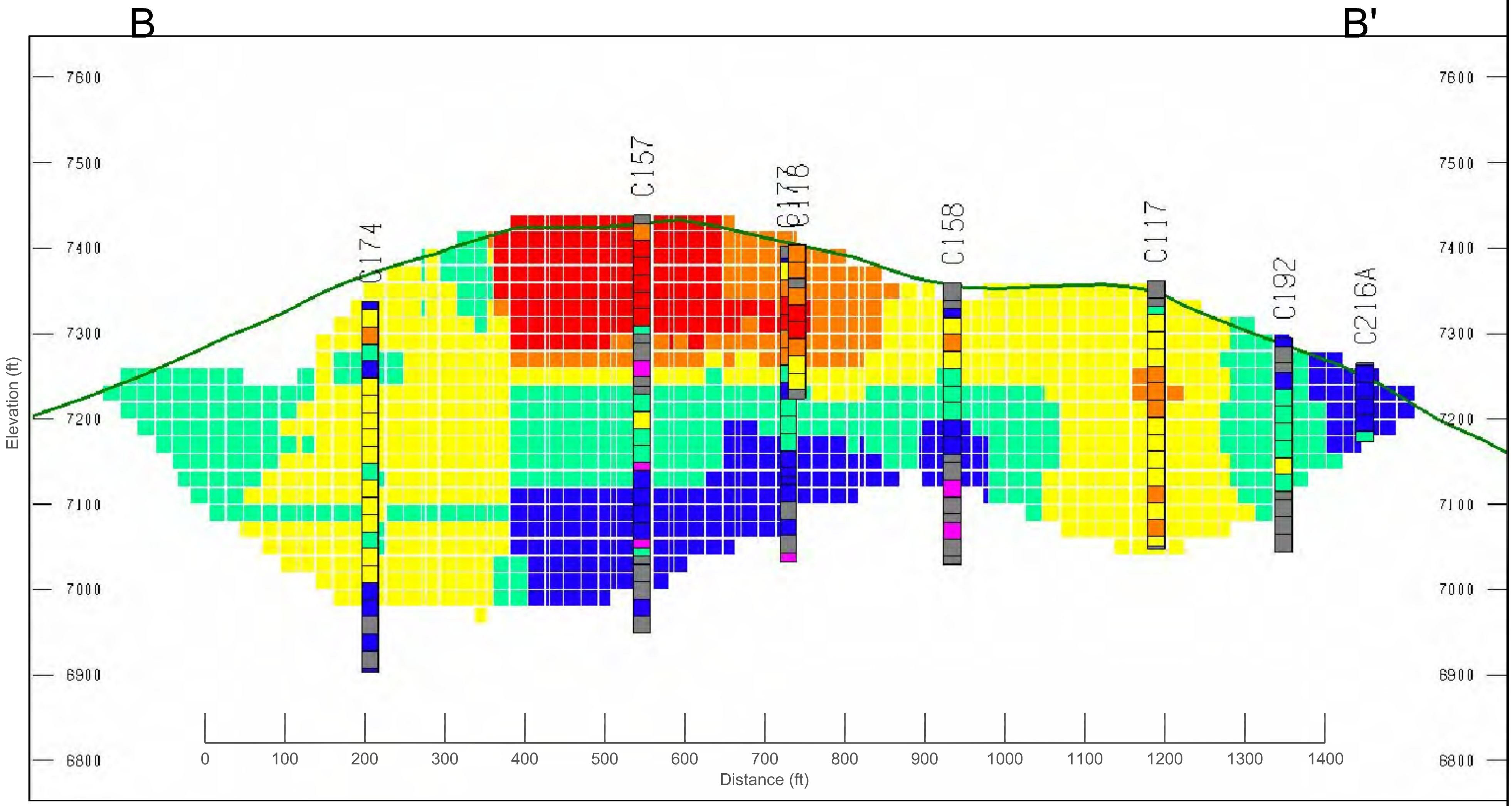
POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

AREA 1 BLOCK MODEL

CROSS-SECTION A-A'

DATE: 04/4/2012
FILE: 418-3 figures
SCALE: as noted

NORWEST
CORPORATION



K2O%

- ⬛ <0
- 🟪 0
- 🟦 1
- 🟩 2
- 🟨 3
- 🟧 4
- 🟥 5

Drillholes displayed can be up to 200 feet from section

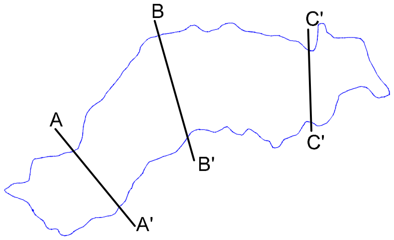


FIGURE 14.3

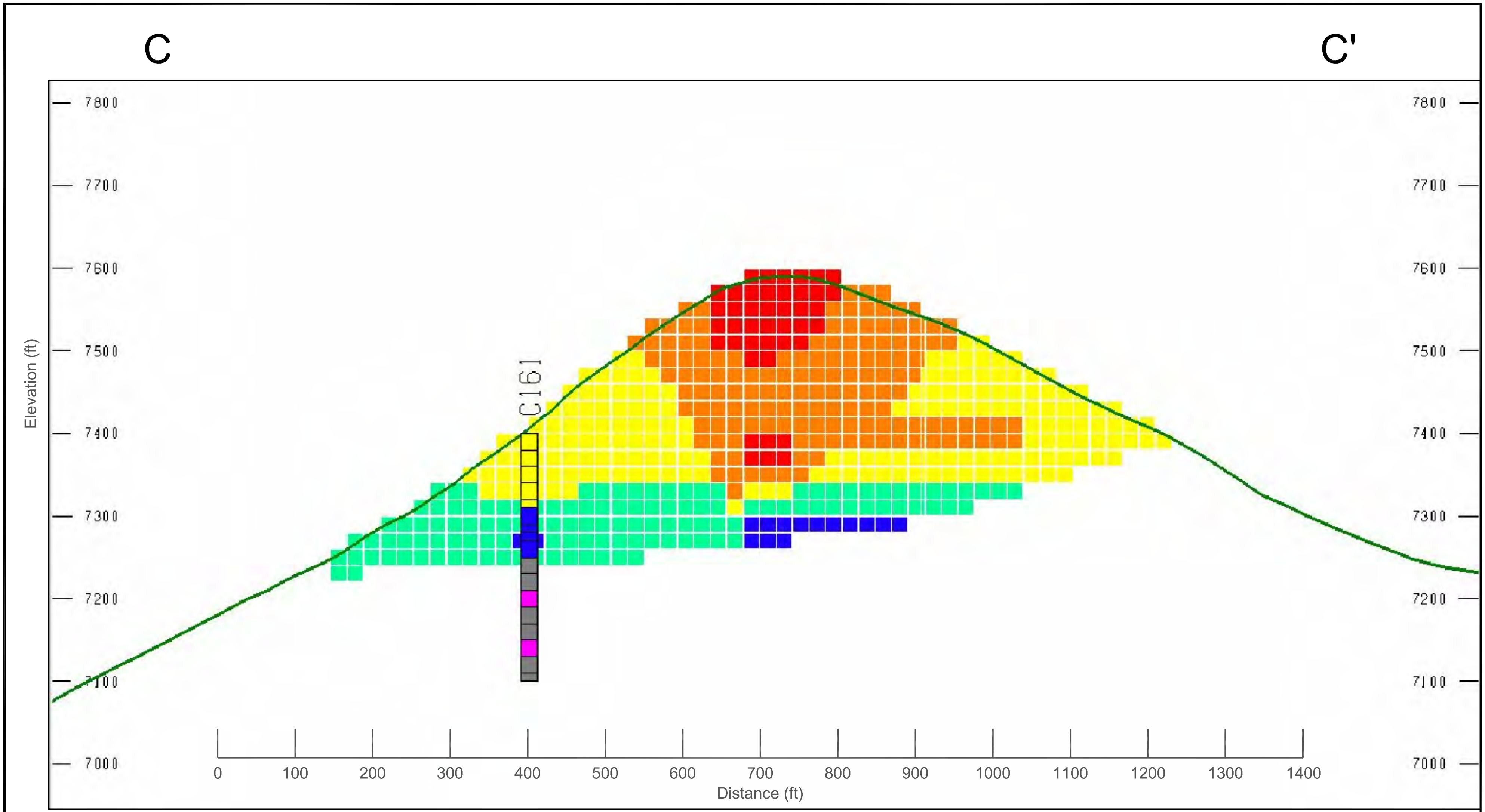
POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

AREA 1 BLOCK MODEL

CROSS-SECTION B-B'

DATE: 04/4/2012
FILE: 418-3 figures
SCALE: as noted

NORWEST
CORPORATION



K2O%

- ⬛ <0
- 🟪 0
- 🟦 1
- 🟩 2
- 🟨 3
- 🟧 4
- 🟥 5

Drillholes displayed can be
up to 200 feet from section

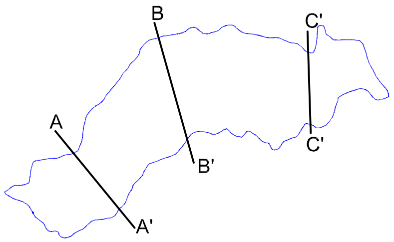


FIGURE 14.4

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

AREA 1 BLOCK MODEL

CROSS-SECTION C-C'

DATE: 04/4/2012	SCALE:	NORWEST CORPORATION
FILE: 418-3 figures	as noted	

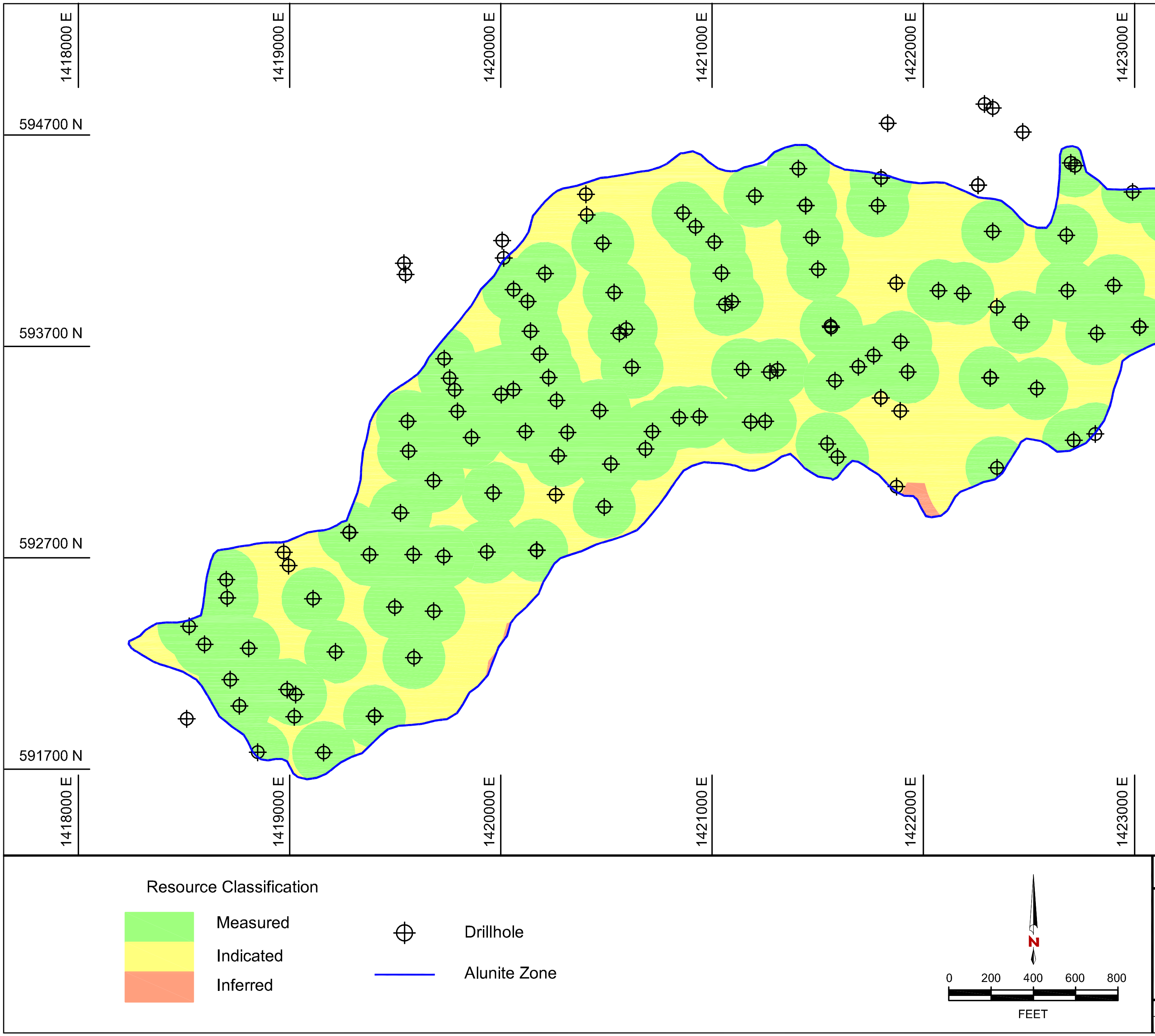
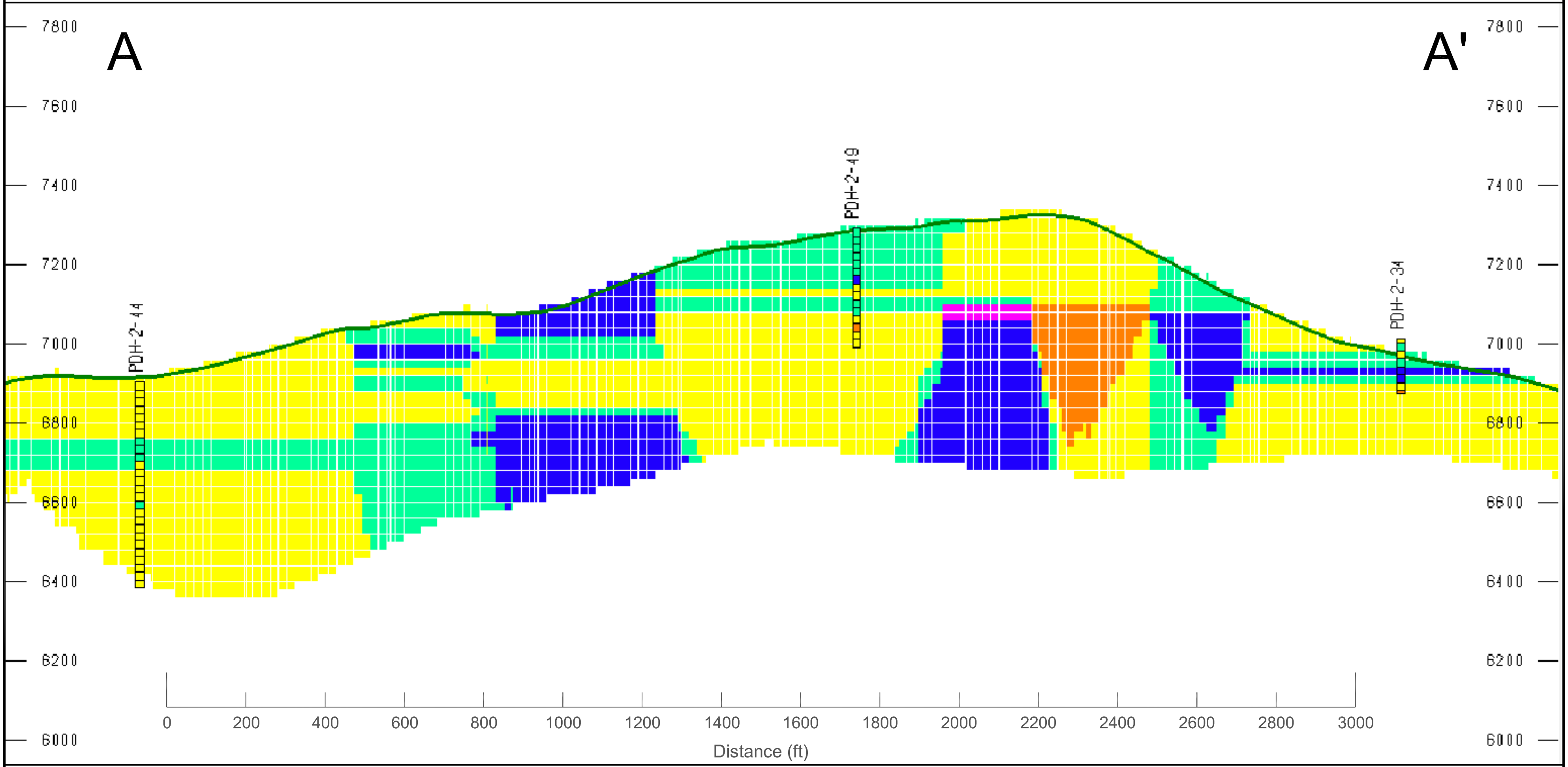


FIGURE 14.5

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

AREA 1 RESOURCE CLASSIFICATION MAP

DATE: 10/30/2012	SCALE: 1:650	NORWEST CORPORATION
FILE: 418–4resource		



K2O%

- ⬛ <0
- 🟪 0
- 🟦 1
- 🟩 2
- 🟨 3
- 🟧 4
- 🟥 5

Drillholes displayed can be
up to 200 feet from section

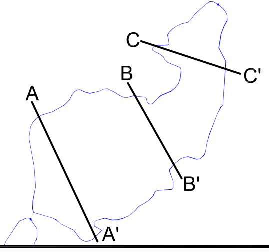


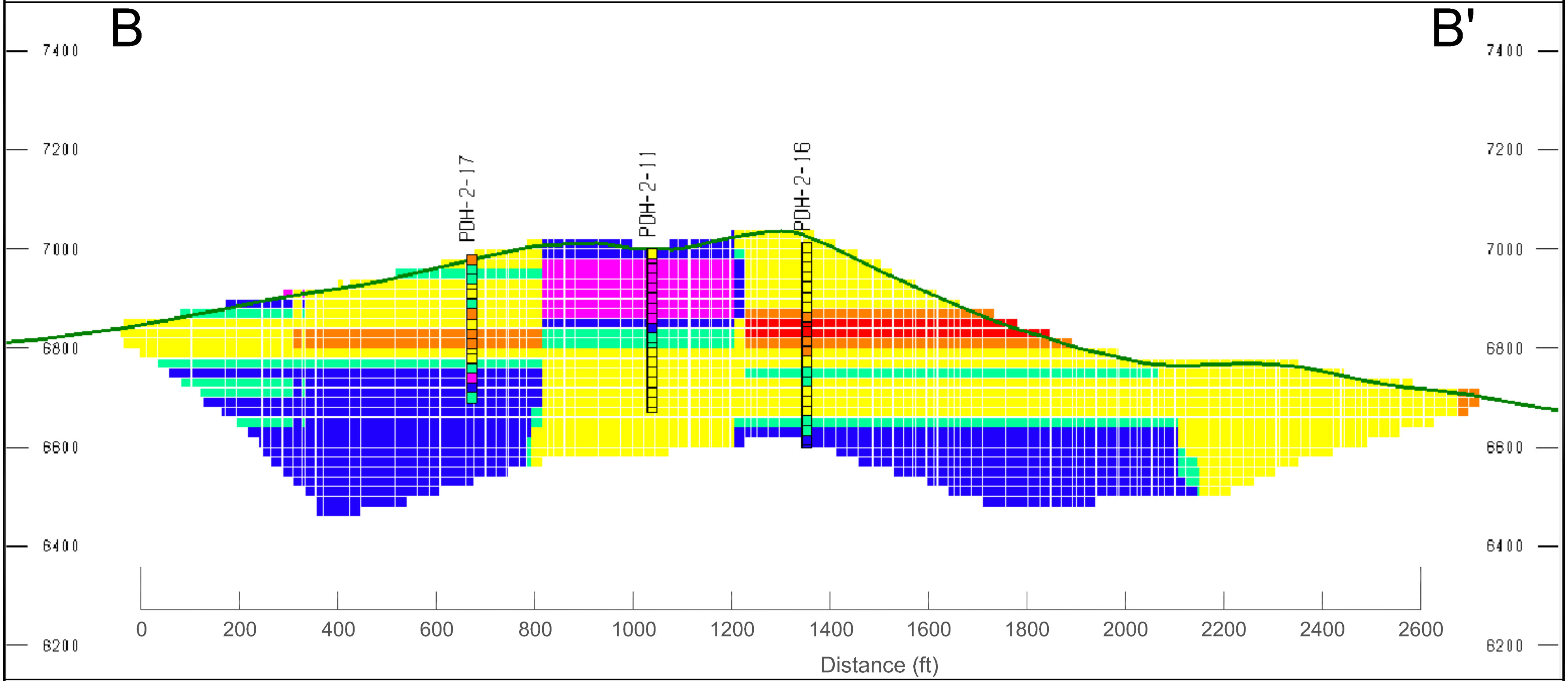
FIGURE 14.6

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

AREA 2 BLOCK MODEL

CROSS-SECTION A-A'

DATE: 10/30/2012
FILE: 418-4 figures
SCALE: as noted
NORWEST CORPORATION



K2O%

◼ <0
🟪 0
🟦 1
🟩 2
🟨 3
🟧 4
🟥 5

Drillholes displayed can be
up to 200 feet from section

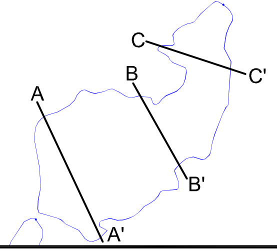


FIGURE 14.7

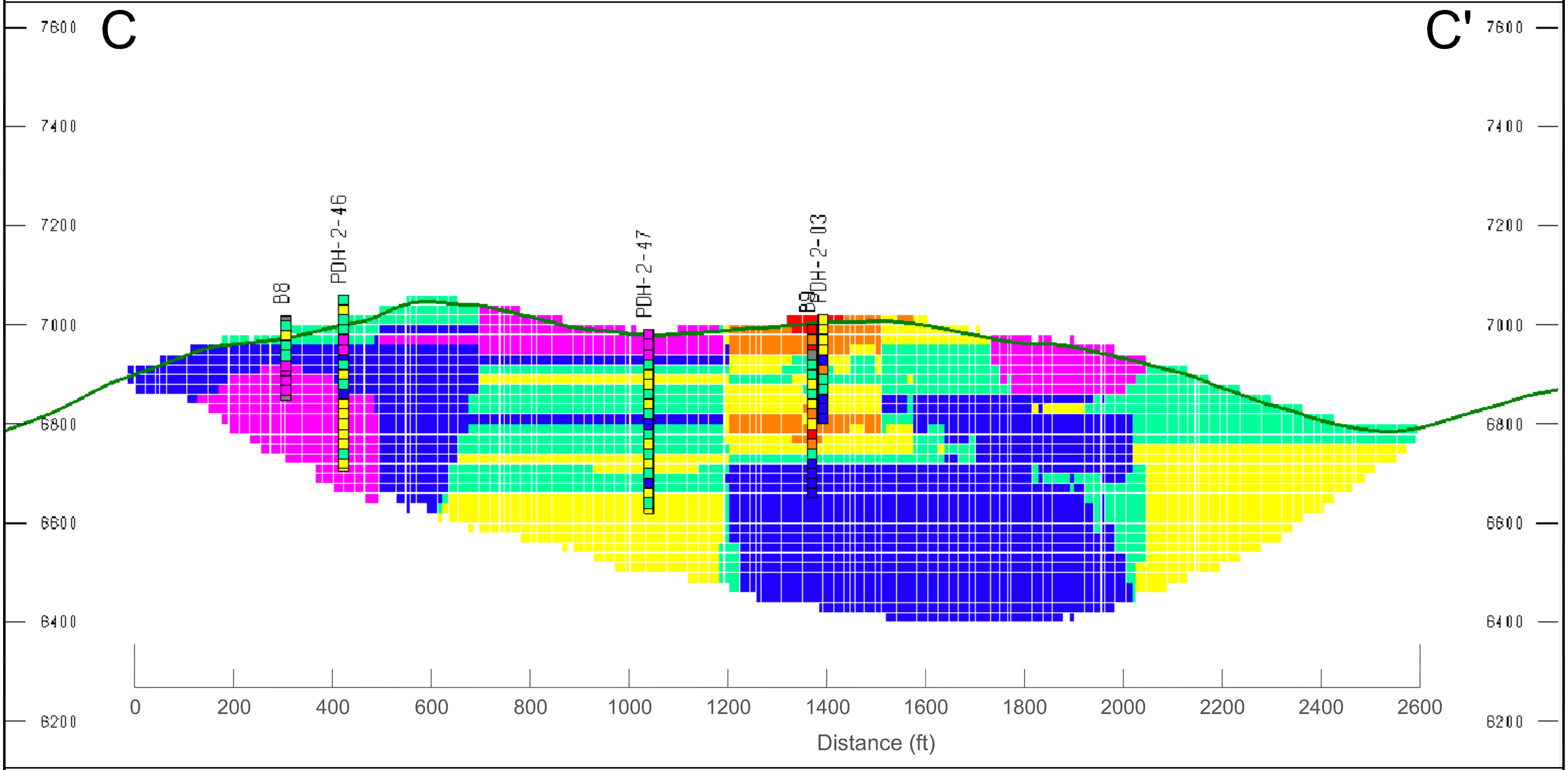
POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

AREA 2 BLOCK MODEL

CROSS-SECTION B-B'

DATE: 10/30/2012
FILE: 418-4 figures
SCALE: as noted

NORWEST
CORPORATION



K2O%

- <0
- 🟪 0
- 🟦 1
- 🟩 2
- 🟨 3
- 🟧 4
- 🟥 5

Drillholes displayed can be
up to 200 feet from section

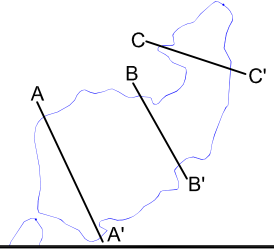


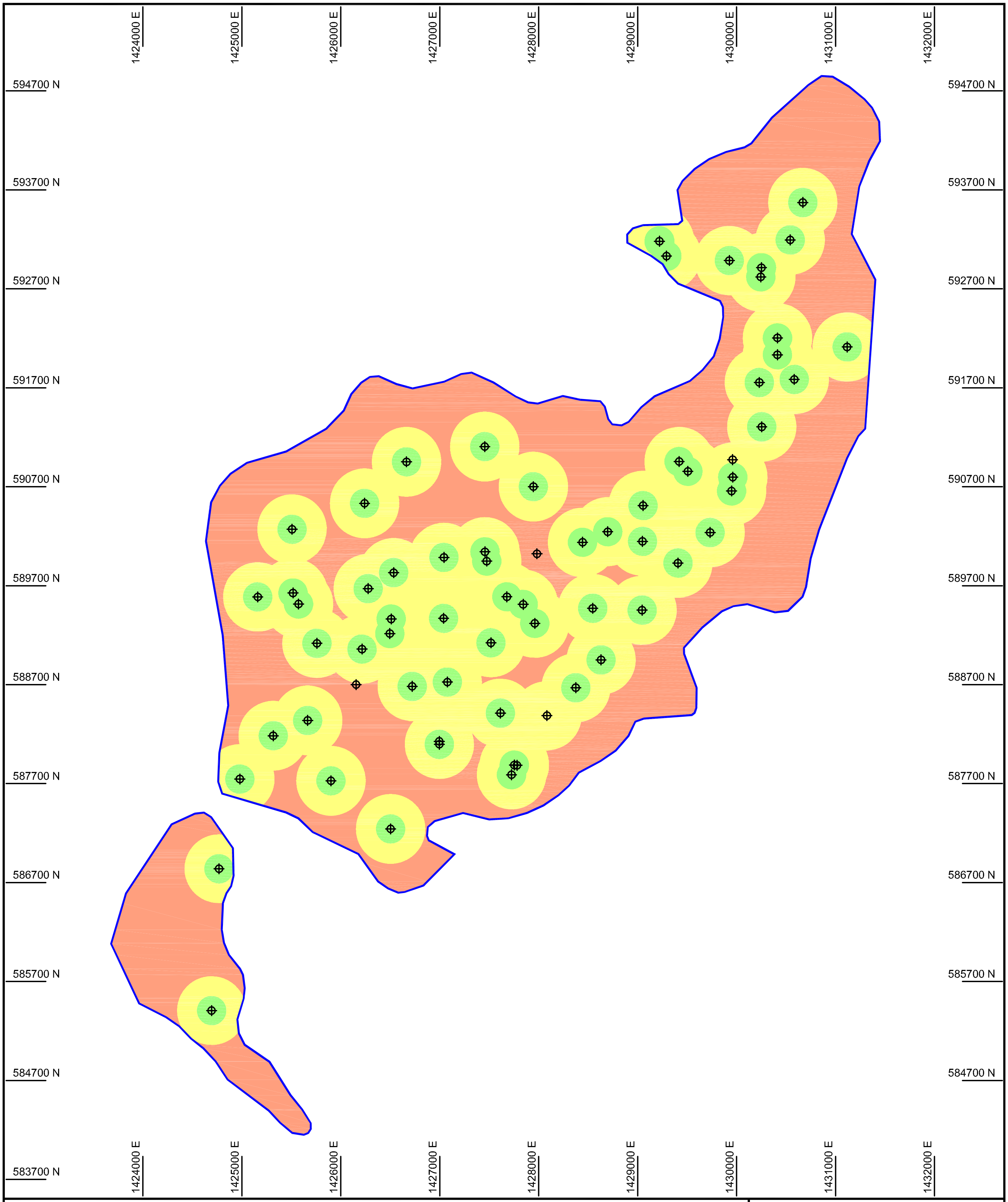
FIGURE 14.8

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

AREA 2 BLOCK MODEL
CROSS-SECTION C-C'

DATE: 10/30/2012
FILE: 418-4 figures
SCALE: as noted

NORWEST CORPORATION



Resource Classification

🟩 Measured

🟨 Indicated

🟧 Inferred

⌖ Drillhole

— Alunite Zone

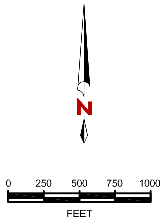


FIGURE 14.9

POTASH RIDGE CORPORATION
BLAWN MOUNTAIN PROJECT

AREA 2 RESOURCE CLASSIFICATION MAP

DATE: 10/30/2012
FILE: 418-4resource
SCALE: 1:1350

NORWEST CORPORATION

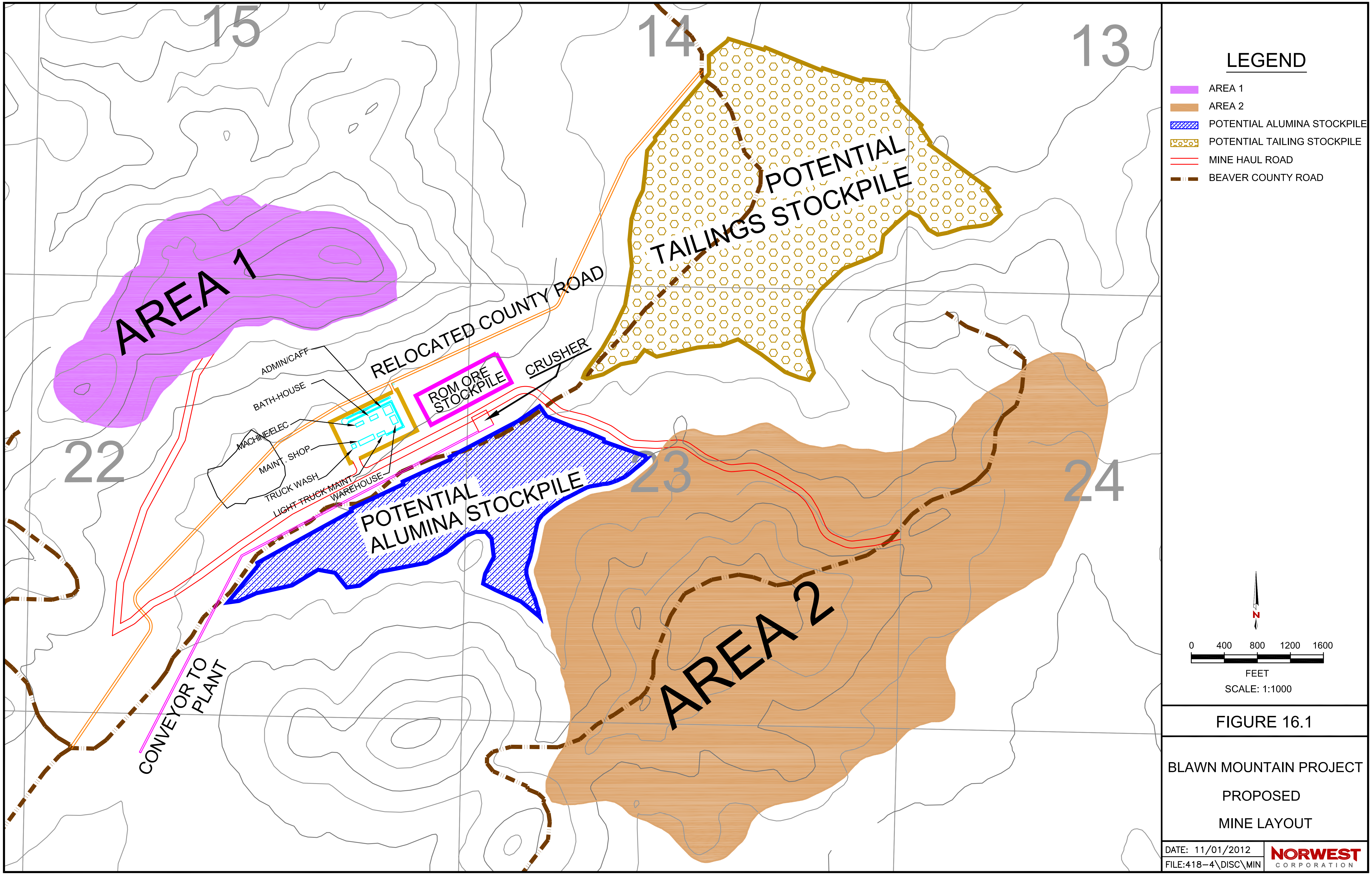
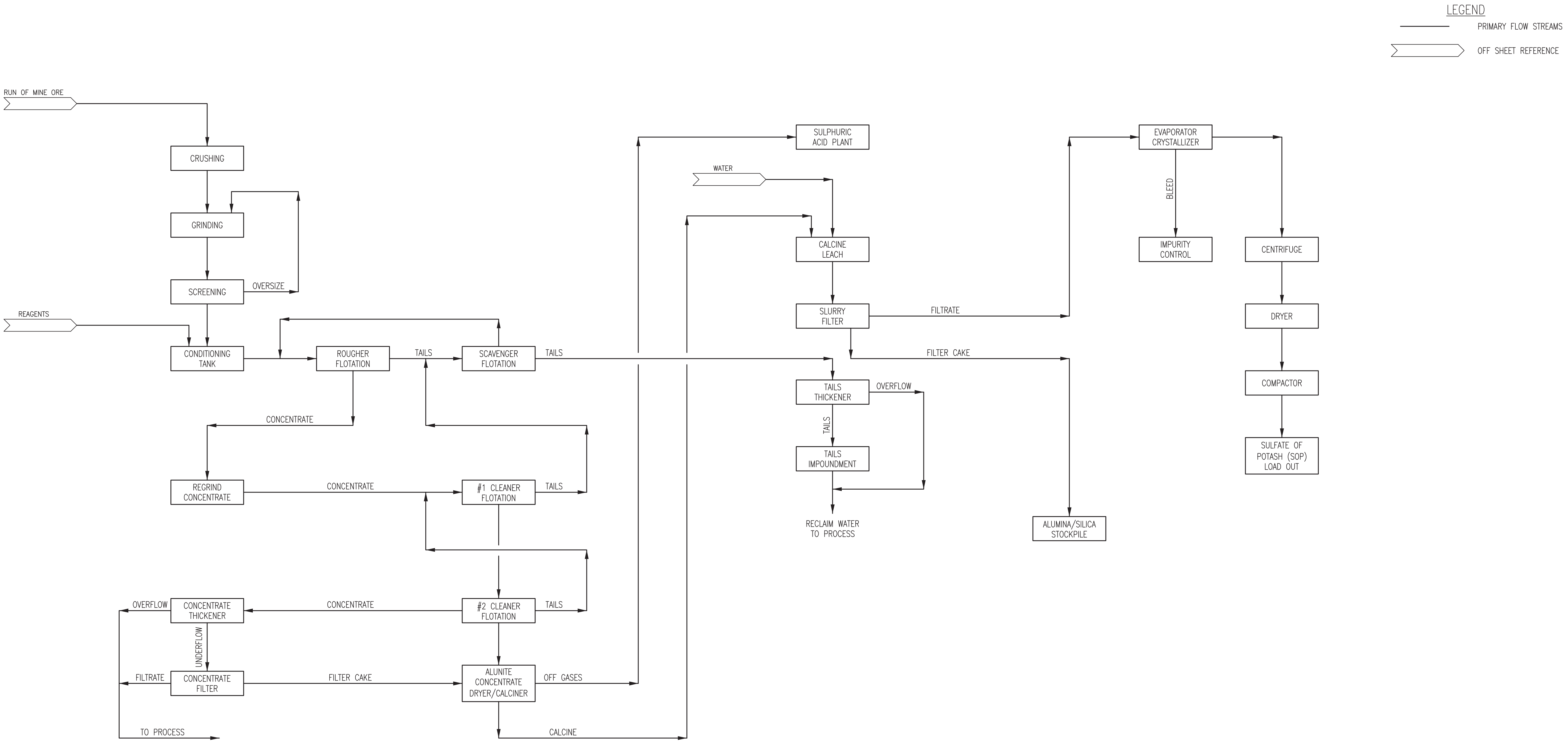


FIGURE 16.1

BLAWN MOUNTAIN PROJECT
PROPOSED
MINE LAYOUT

DATE: 11/01/2012
FILE:418-4\DISC\MIN

NORWEST CORPORATION



No.	DATE	BY	APP	REVISION	REFERENCE DRAWING	NUMBER
0	11/01/12	BES	BN	ISSUED FOR PEA REPORT		

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:	LRN	Date	10/30/12
Drawn By:	BES	Date	10/30/12
Chk'd. By:	LRN	Date	10/30/12
ENGR.	LRN	Date	10/30/12
APPD.	BN	Date	10/30/12
APPD.		Date	

POTASH RIDGE
ALUNITE PROCESSING
BLOCK FLOW DIAGRAM

SCALE NONE
Project No. 314-006
DRAWING No. FIGURE 17.1 BFD-001
REVISION 0