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June 25, 2018

Mr. Brandon Stimac
Environmental Engineer
Highland Copper Company
310 US Hwy 2 East
Wakefield, MI 49968

Re: A Phase I Archeological Survey of New Areas at the Copperwood Project in Gogebic County, Michigan
TRC Project No. 299813.0000
WIARC#218

Dear Mr. Stimac:

This report is for the above referenced Phase I Archaeological Survey. The new areas are: explosives gallery, process plant and box cut, water tank and road, sewage lagoons, topsoil stockpile, and mine ventilation and road. The entire survey covered approximately 66 acres plus 3,280 linear feet for water lines. This is the most recent archaeological work conducted for the mining operation; other surveys preceded this one between 2009 and 2012 and are covered in the brief synopsis in the next section. All of the reports were submitted to and accepted by the Michigan State Historic Preservation Office (MSHPO).

BACKGROUND

In 2009, AVD Archaeological Services, Inc., (AVD) conducted a Phase I Archaeological Survey of access roads for core drilling at the Copperwood Project in Gogebic County, Michigan (Van Dyke 2009). The project was done for Orvana Minerals Corporation who wished to establish a new underground copper mine in the western Upper Peninsula of Michigan. In September and October of 2008, Orvana Resources US Corp., entered into mineral leases covering 1,759 acres in northeast Gogebic County, Michigan. The undertaking is referred to as the Copperwood Project. Orvana retained AECOM to complete an Environmental Impact Assessment for inclusion in a Mine Permit Application under Michigan's Nonferrous Metallic Mining Regulations (Part 632 of P.A. 451, 1994 as amended). Part 632 of Michigan's Non-Ferrous Metallic Mining regulation clearly outlines the studies necessary for a mine permit application and requires, among other things, consideration of cultural, historical, or archaeological resources.

The first archaeological work was a Phase I Archaeological Survey of 60 planned spur roads to enable drill rig access within the project area. Archaeological literature and archives research was conducted at the Michigan Bureau of History prior to field work. On-going research during the project consisted of reviewing published and unpublished reports, books, and maps. No archaeological sites were known to be within or near the project area. Fieldwork consisted of shovel testing each of the spur road locations at 15 meter intervals. No artifacts or archaeological sites were found by the Phase I Archaeological Survey.

In 2010, AVD returned to the Orvana Mine area to conduct a Phase I Archaeological Survey of 40 acres for a mine site (Van Dyke 2010a). The additional Phase I archaeological survey result was presented in a letter and map as an addendum to the 2009 report. The fieldwork consisted of digging approximately 495 shovel tests in the proposed plant site at 15 meter intervals. The archaeological survey found no archaeological sites or artifacts.

AVD returned to the site again in 2010 for a Phase I Archaeological Survey of 640 Acres for Orvana Resources US Corporation at the Copperwood Project (Van Dyke 2010b). The survey was conducted between October 5 and 18, 2010 for a proposed tailings site in the south one half of section 6, and the north one half of section 7, in T49N, R46W in Ironwood Township.

Another survey of 1,140 acres was combined in the same report with the previous 640 acre survey in 2011 (Van Dyke 2011). The 640 acres survey, described above, was supplemented with another 1,140 acres of survey comprised of the north half of section 6, the south half of section 7, and the western 500 acres of section 8, also in Ironwood Township. The 1,140 acres were surveyed between May 2 and June 16, 2011 using the same archaeological field techniques described in the original report of the 60 spur roads (Van Dyke 2010a). Once again, no artifacts or historical sites or ruins were found.

Fieldwork for the two surveys consisted of digging between 6,050 - 7,915 shovel tests in the project area at 15 meter intervals covering between 73 - 96% of the area, a very reliable sample. For all that effort, no archaeological sites or artifacts were found.

Finally, in 2012, another survey was conducted, this one in sections 2, 11, and 12 of T49N, R46W, Ironwood Township, Gogebic County. Archaeological survey fieldwork took place between September 10-14, 2012. A total of approximately three miles of 100′ corridor (with three wider spots of various lengths) were shovel tested.

For much of its length, the water line route was a 100 foot wide corridor (50′ on either side of centerline of existing road) that followed old logging roads. Four additional areas of survey consisted of a 200′-x-300′ rectangle for a water inlet from lake superior; an alternate route 100′ wide and 1,100′ long with a 200′-x-300′ rectangle for a water intake; a new road spur with a 100′ wide corridor and approximately 1,300′ long corridor; and a 200′–x-300′ rectangular area for a water tank. The archaeological survey consisted of shovel testing at 15 meter intervals and discovered no archaeological sites or artifacts.

2018 Phase I Archaeological Survey

In 2018, a Phase I Archaeological Survey was conducted at new project areas shown in Figure 1: explosives magazine (7.7 acres), a new plant location (31.9 acres), sewage lagoons and stockpile (15.2 acres), box cut for mine access (10.4 acres), mine vent intakes and exhaust vents (1.0 acres), and a road from the north end of box cut to mine vent intake location (3,280 linear feet). The survey covered approximately 66 acres and 3,280 linear feet. Map 1 shows the survey area layout. Archaeological fieldwork was conducted on May 14 to May 17, 2018. Map 2 is the General Land Office surveyor's sketch map which depicts the area as predominantly forested with cedar, hemlock and fir trees and dissected by many small streams and a few ravines. Maps 3 and 4 are approximations of the shovel test patterns at the various locations for this survey.

Explosives Magazine

This wooded area was shovel tested at 15 meter intervals. Soil profiles were typical for the area with an A horizon of brown clayey silt loam to about 10 cm over a B horizon of red clay from 10-35 cm below surface. Near wetlands, the profile showed an A horizon of dark brown clay loam (0-15 cm) over a B horizon of gray clay loam.

Process Plant and Box Cut

The wooded area was shovel tested at 15 meter intervals. Soil profiles were distinctive of disturbance from use as logging/construction roads.

Water Tank and Road

This area, also wooded, was shovel tested at 15 meter intervals. Some parts of the parcel had been disturbed by mine development related activities such as construction of boring roads. Areas that had intact soils showed typical profiles for the area with an A horizon of brown clayey silt loam to about 10 cm over a B horizon of red clay from 10-35 cm below surface.

Sewage Lagoons

This wooded area was shovel tested at 15 meter intervals. Soil profiles were typical (as above) except near wetlands which showed an A horizon of dark brown clay loam (0-15 cm) over a B horizon of gray clay loam.

Topsoil Stockpile

This area was also wooded and was shovel tested at 15 meter intervals. Soil profiles were typical for the area except near wetlands which had an A horizon of dark brown clay loam (0-15 cm) over a B horizon of gray clay loam.

Mine Ventilation and Road

This area, also wooded, was shovel tested at 15 meter intervals. Like the other areas, soil profiles were typical but with a few wetlands. Profiles near the wetlands showed an A horizon of dark brown clay loam (0-15 cm) over a B horizon of gray clay loam plus disturbance from logging roads.

CONCLUSION AND RECOMMENDATION

This survey was the sixth archaeological survey for this mining project conducted since 2009. The first survey was for spur roads for drill rigs, the second covered 40 acres for a proposed plant site, the third and fourth covered 640 and 1140 acres for various proposed mining related operations, the fifth was for a waterline corridor, and the last was a new location for the plant site and various other operations locations. More than 1,900 acres and three plus miles of water lines or roads were shovel tested at 15 meter intervals. With all that, no artifacts were found.

Given the above, it is unlikely that an archaeological site that might hold information important to prehistory or history will be disturbed. If you have any questions about this survey, please do not hesitate to contact me at 262-225-5105 or by email at avandyke@trcsolutions.com.

Sincerely,

Allen P Van Dyke

Allen P. Van Dyke
Principal Archaeologist

Attachments: One Figure, Four Maps

References Cited

Van Dyke, A.P.

2009 A Phase I Archaeological Survey of Access Roads for Core Drilling at the Copperwood Project in Gogebic County, Michigan.

2010a A *Phase I Archaeological Survey of 40 Acres for Orvana Resources US Corporation Copperwood Site, Gogebic County, Michigan.*

2010b Phase I Archaeological Survey of 640 Acres for Orvana Resources US Corporation Copperwood Project, Gogebic County, Michigan.

2011 *Phase I Archaeological Survey of 640 Acres and 1140 Acres for Orvana Resources US Corporation Copperwood Project, Gogebic County, Michigan.*

2012 A *Phase I Archaeological Survey For the Gogebic Range Water Authority For the Copperwood Water Supply, Gogebic County, Michigan*

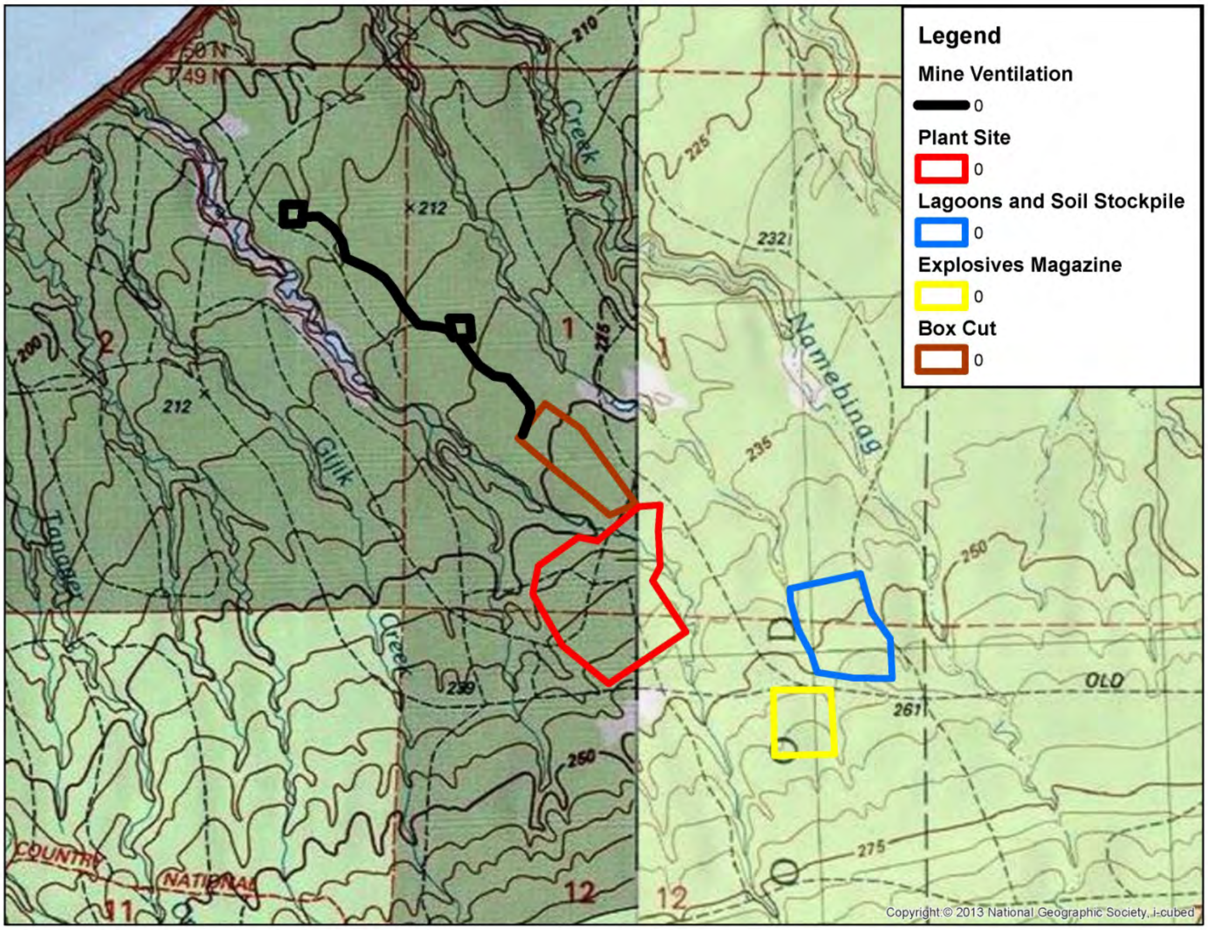
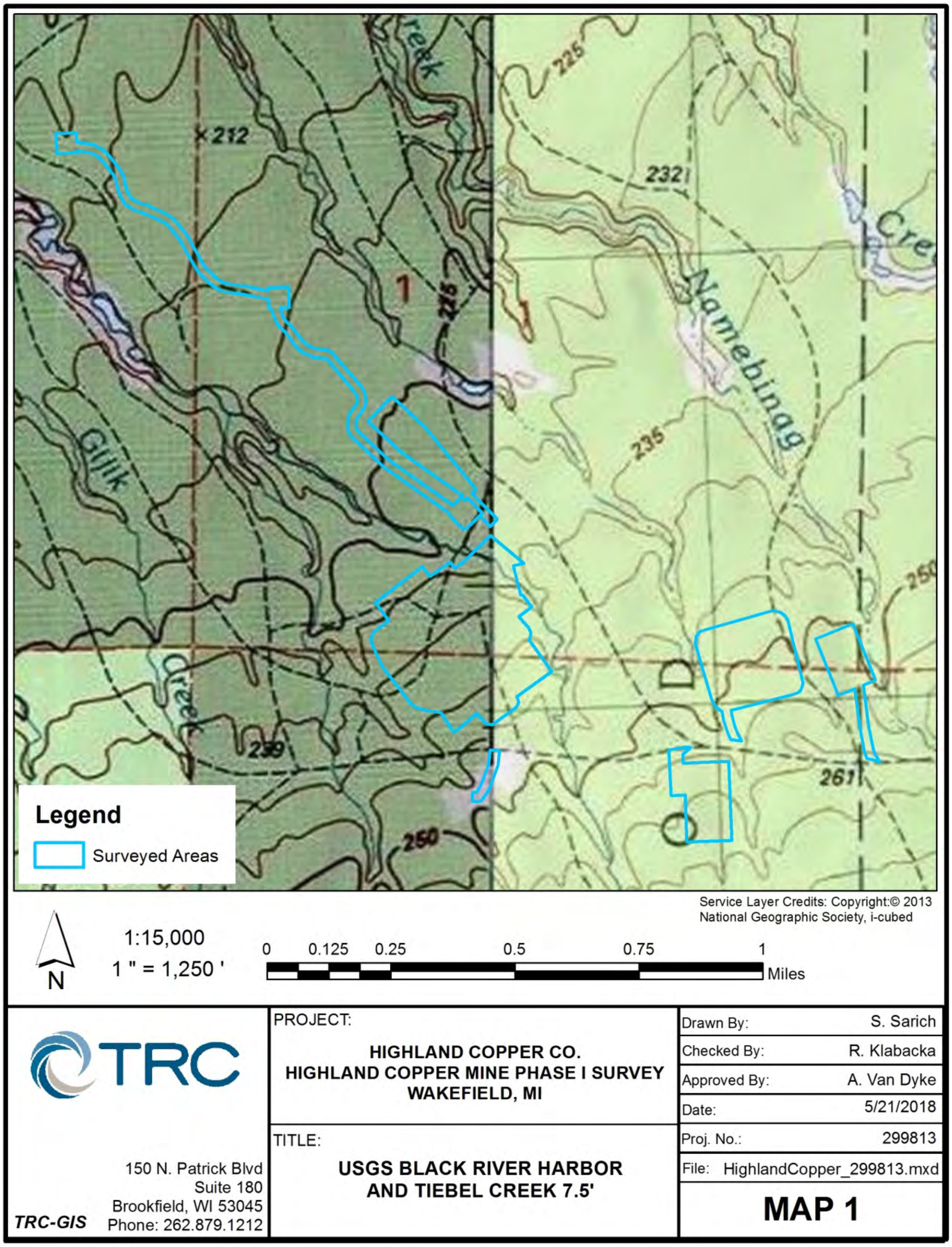


Figure 1: General Layout of Project. Source: Highland Copper Company.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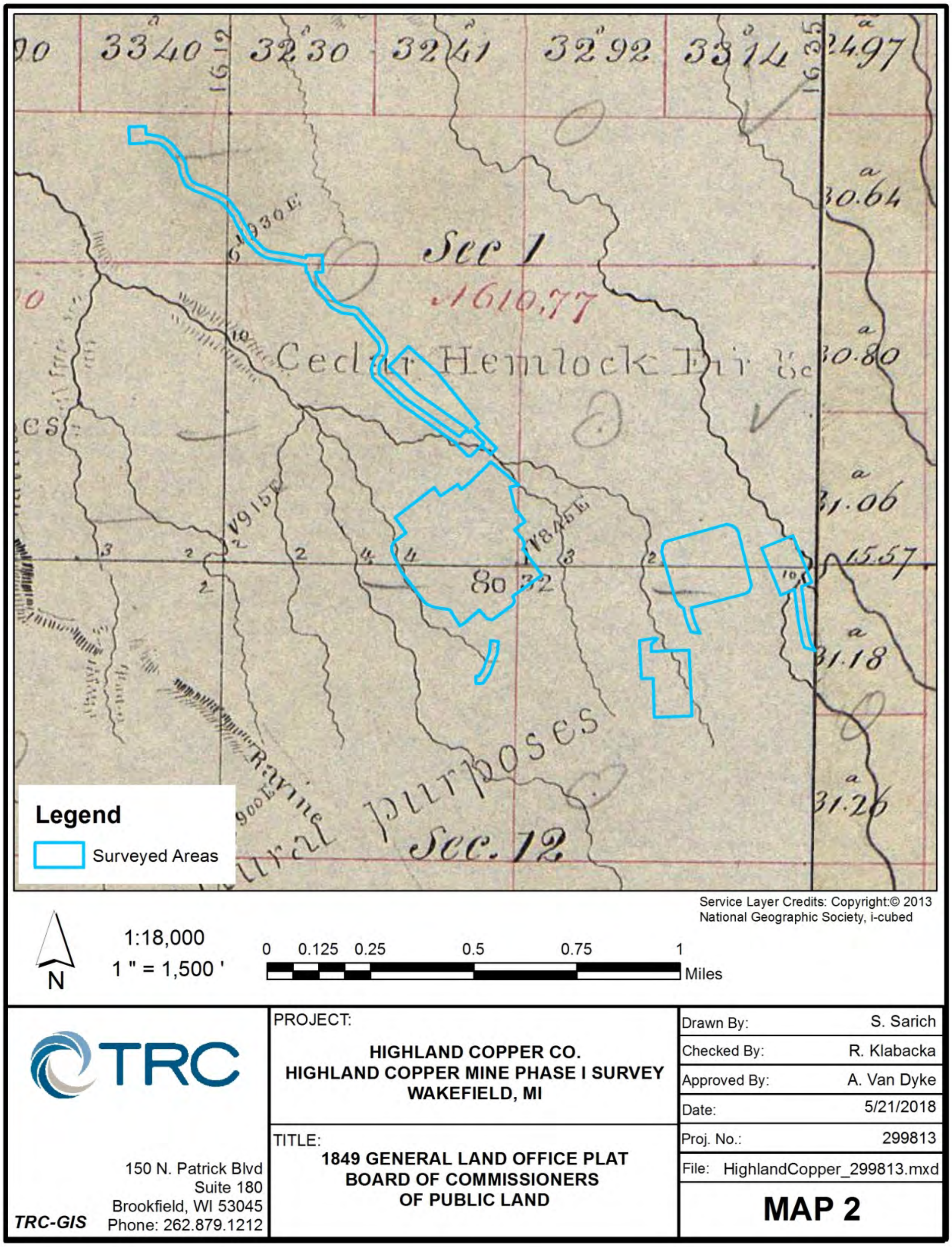
Suite 180
Brookfield, WI 53045
Phone: 262.879.1212
TRC-GIS
PROJECT:
HIGHLAND COPPER CO.
HIGHLAND COPPER MINE PHASE I SURVEY
WAKEFIELD, MI
TITLE:
USGS BLACK RIVER HARBOR
AND TIEBEL CREEK 7.5'
Drawn By: S. Sarich
Checked By: R. Klabacka
Approved By: A. Van Dyke
Date: 5/21/2018
Proj. No.: 299813
File: HighlandCopper_299813.mxd
MAP 1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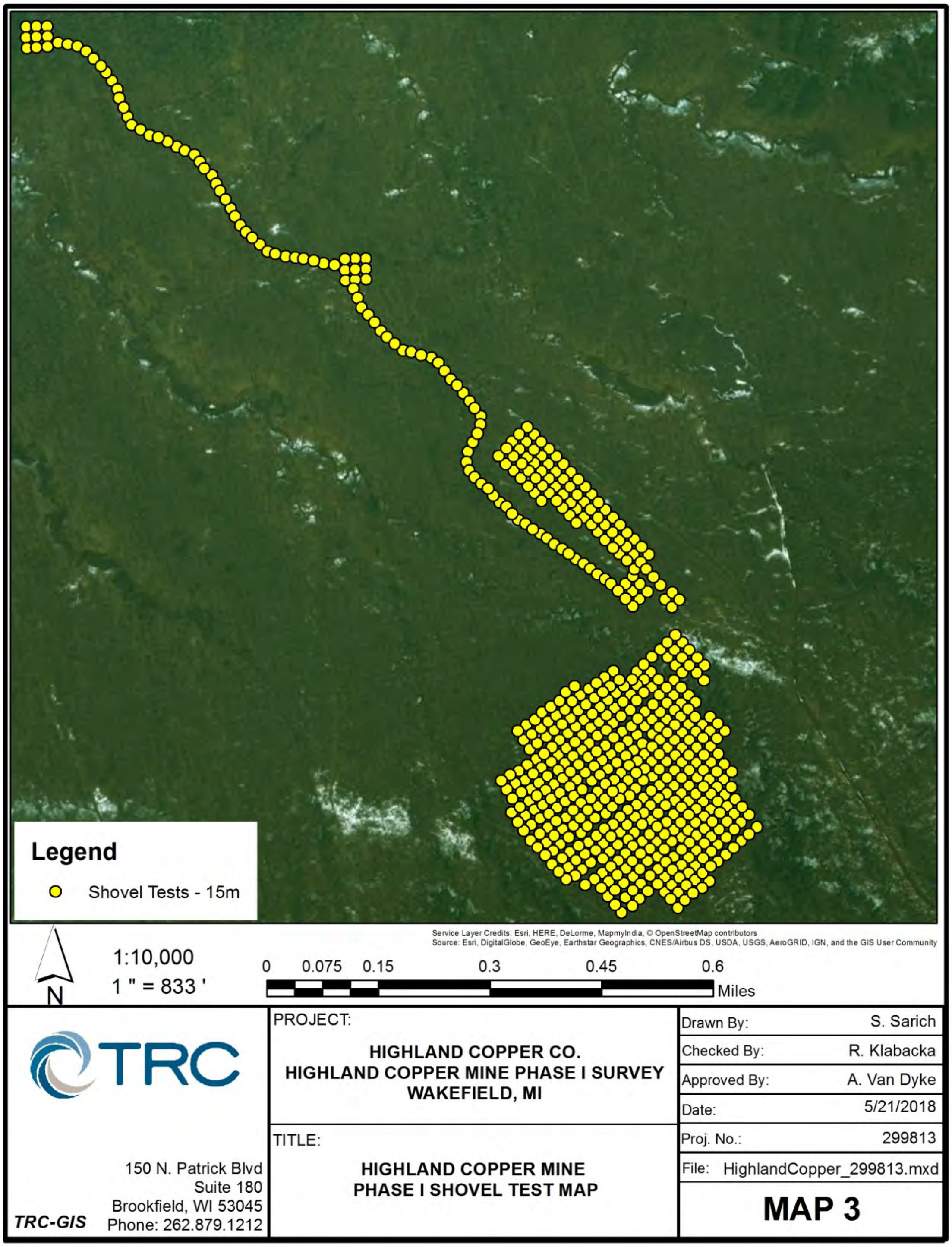
150 N. Patrick Blvd
Suite 180
Brookfield, WI 53045
Phone: 262.879.1212
TRC-GIS
PROJECT:
HIGHLAND COPPER CO.
HIGHLAND COPPER MINE PHASE I SURVEY
WAKEFIELD, MI
TITLE:
1849 GENERAL LAND OFFICE PLAT
BOARD OF COMMISSIONERS
OF PUBLIC LAND
Drawn By: S. Sarich
Checked By: R. Klabacka
Approved By: A. Van Dyke
Date: 5/21/2018
Proj. No.: 299813
File: HighlandCopper_299813.mxd
MAP 2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Service Layer Credits: Esri, HERE, DeLorme, MapmyIndia, © OpenStreetMap contributors
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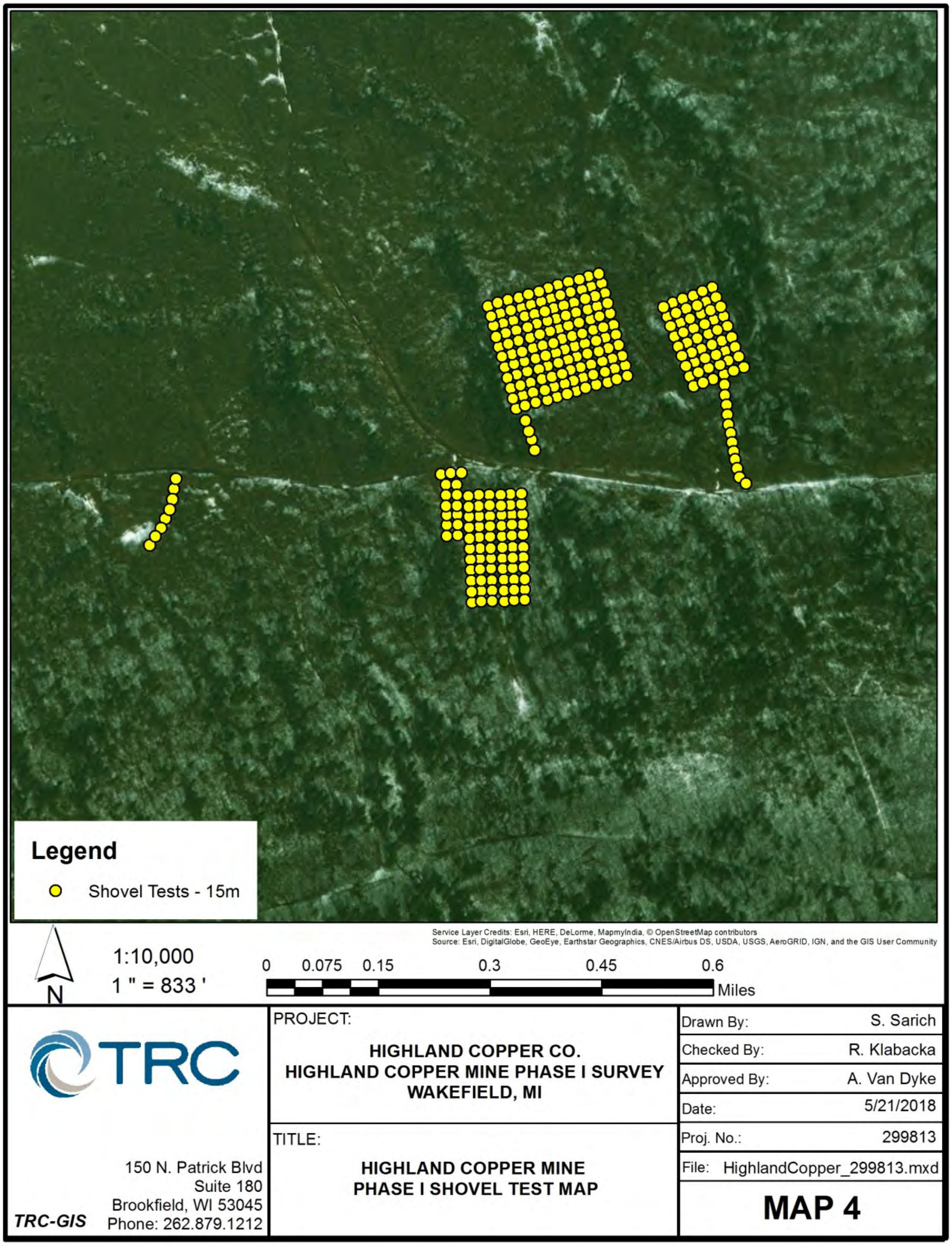
TRC
150 N. Patrick Blvd
Suite 180
Brookfield, WI 53045
Phone: 262.879.1212
TRC-GIS
PROJECT:
HIGHLAND COPPER CO.
HIGHLAND COPPER MINE PHASE I SURVEY
WAKEFIELD, MI
TITLE:
HIGHLAND COPPER MINE
PHASE I SHOVEL TEST MAP
Drawn By: S. Sarich
Checked By: R. Klabacka
Approved By: A. Van Dyke
Date: 5/21/2018
Proj. No.: 299813
File: HighlandCopper_299813.mxd
MAP 3


Legend
Shovel Tests - 15m
Service Layer Credits: Esri, HERE, DeLorme, MapmyIndia, © OpenStreetMap contributors
Source: Esri, DigitalGlobe, GeoEye, Earthstar Geographics, CNES/Airbus DS, USDA, USGS, AeroGRID, IGN, and the GIS User Community
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Suite 180
Brookfield, WI 53045
Phone: 262.879.1212
PROJECT:
HIGHLAND COPPER CO.
HIGHLAND COPPER MINE PHASE I SURVEY
WAKEFIELD, MI
TITLE:
HIGHLAND COPPER MINE
PHASE I SHOVEL TEST MAP
Drawn By: S. Sarich
Checked By: R. Klabacka
Approved By: A. Van Dyke
Date: 5/21/2018
Proj. No.: 299813
File: HighlandCopper_299813.mxd
MAP 4

MEMORANDUM

Date: September 14, 2012 **Project No.:** 12388885

To: Orvana Resources US Corporation

From: Golder Associates Inc.

RE: **COPPERWOOD PROJECT – ALTERNATIVES ANALYSIS, ALTERNATIVE 1 CLARIFICATION AND AMPLIFICATION**

Orvana Resources US Corporation ("ORUSC") has prepared this document to clarify and supplement the Alternatives Analysis which was submitted as part of the Copperwood Project revised Part 301 and Part 303 Wetland Permit application (MDEQ File No. 12-27-0001-P, ORUSC Copperwood Project, dated May 15, 2012). Specifically, ORUSC is providing additional information regarding Alternative 1 – Underground Tailings Disposal. Section 5.1 of the Alternatives Analysis presents the alternative of placing tailings back into the mine as an option for disposal. ORUSC commissioned engineers to study this option, including studies performed by a laboratory which tested the tailings to determine the suitability for use as mine backfill. The natural containment created by the void-space resulting from underground mining activities initially seemed to ORUSC to be an obvious, safe, manageable, and low-cost alternative to tailings management. Three different scenarios for backfill were considered including unaugmented whole tailings, hydraulically separated tailings, and tailings augmented with a cementation agent. Upon examination by ORUSC and their consulting engineers, however, none of the tailings disposal scenarios within the mine cavity below ground were determined to be feasible or prudent for this project. As explained in the Alternatives Analysis submitted as part of the Part 301 and Part 303 permit application, a surface tailings disposal facility is the most feasible and prudent option. ORUSC has prepared this document to describe challenges faced with below-ground disposal of tailings, thereby demonstrating that any backfilling of the tailings into the mine is neither feasible nor prudent.

1.0 BACKGROUND

As discussed in the May 15, 2012 Part 301 and 303 permit application documents, infrastructure and facilities planned to be constructed on the project site include an entrance road, a mill, a water treatment plant, a portal into the mine, a fresh water intake structure and associated pipeline (in conjunction with a local municipal water authority), various storm water management ponds, and a tailings disposal facility (TDF). The current mine plan is based on a 13-year mine life that includes the excavation of 30.3 million tons of ore. The mining plan provides for capturing groundwater seeping into the mine, and pumping it to the TDF for storage until the fourth year of mining operations. By the beginning of the fourth year, the water treatment plant (WTP) will be constructed and operational and tailings dewatering water will be routed through the WTP. Also, it is planned that at the start of the fourth year of mining operations, mechanical dewatering of the mine cavity will cease and underground mine water will be stored underground as space is made available in the mined out areas. The activities presently modeled, such as the timing of the WTP construction and operation and the pumping of groundwater have been optimized from an operations and budgetary standpoint and are reported in the Bankable Feasibility Study of the Copperwood Project, Document Q431-01-028, dated March 21, 2012 (herein referred to as the BFS).

2.0 CAVITY VOLUME

As stated in the BFS, approximately 30.3 million tons (dry weight) of ore will be extracted, and as a result of the milling processes which separate the copper and other valuable metals from the ore, approximately 28.7 million tons (dry weight) of tailings are expected to be generated on the project. The mined ore, which exists at a dry density of generally 169 pounds per cubic foot (pcf) (reference: Original Part 301 and Part 303 Wetland Permit) expands (swells) during the mining process and is ground to a fine particle size during the milling process. The total cavity left open after mining 30.3 million tons of ore at 169 pcf is approximately 358.6 million cubic feet (ft^3), however, the entire mine cavity cannot be used for backfill due to concurrent mining operations. As shown on Figure 1 (Attachment 1), backfilling the mine cavity could occur in two areas (Areas 1 and 2) of the mine as active mining proceeds. To isolate the area of the mine open for mining from those areas receiving backfill, the process of creating bulkhead dams would need to be implemented. Bulkhead dams allow for the safe separation of backfilling activities concurrent with active mining operations. Therefore, as shown on Figure 1, the total volume of the cavity after mining Years 1 through 6 is approximately 136,760,036 ft^3, and the total volume of the cavity after mining Years 10, 11 and about half of Year 12 is approximately 79,908,470 ft^3. The total mine cavity volume within Areas 1 and 2 available for backfill is 216.7 million ft^3. Additional filling beyond Area 2 has not been considered for two reasons. First, mining activities must progress safely ahead of backfilling activities and as shown in Figure 1, a necessary delay must occur for backfilling after about halfway through Year 12. Second, the only opportunity to backfill additional material would be after Year 13. However, at that time, the tailings will, by necessity, have already been deposited elsewhere and handling them a second time is costly and serves no purpose. Additional filling for Years 7, 8, 9, and 13 has not been considered due to the close proximity of those areas to the mine access portal and to allow for the mining of the bridge area connecting the two portions of the mine.

As will be explained in Section 3.0 of this document, the underground volume in the mine cavity will be less than the volume of the tailings produced. The volume of tailings that exceed 216.7 million ft^3 must be disposed of elsewhere. For cost estimation purposes, it was assumed that any tailings not disposed of underground will be disposed of in a TDF on the surface at the project site.

3.0 BACKFILL OPTIONS

Tailings produced in the milling process generally have high water content, exhibit low strengths and typically require treatment prior to underground disposal for safety reasons as described below. The tailings produced at the Copperwood Project, as tested by the consulting engineers (reference: Golder Draft Technical Memorandum 'Interpretation of Tailings Dewatering Assessment – Revision 1', dated March 30, 2012, see Attachment 2), will be comprised of very finely ground rock (P50 at approximately 22 microns, i.e. 50 weight percent passing 22 microns). Tailings slurry discharged from the milling process will be comprised of approximately 79 percent process water and 21 percent solids (by weight) (wt%), as noted in the Alternatives Analysis. Once the tailings are processed through the thickener, the tailings slurry will be dewatered to approximately 50 wt% solids and 50 wt% water.

Three backfill options are evaluated in this document including:

1. Unaugmented (Raw) Tailings Backfill
2. Hydraulic Sand Backfill
3. Augmented Tailings Backfill

These three options are compared based on feasibility and cost, however, safety, surface area required for the TDF, volume, density, and ancillary equipment needs were also considered. Each backfilling option described assumes the best possible filling scenario, not taking into account other factors that may affect disposal efficiency, such as groundwater seepage.

At this point it is prudent to define some of the terms that will be used in the following paragraphs:

- Unaugmented backfill – refers to the use of the tailings, in their unaltered state (i.e. no mechanical separation, no addition of binding agents) as backfill.
- Structural backfill – refers to the use of underground mining backfill materials augmented with a binding agent, such as normal Portland cement (NPC), which imparts an increased strength characteristic that allows the backfill to be freestanding when a vertical backfill surface is exposed.
- Backfill replacement factor – refers to the ratio of weight per unit volume of backfill deposited, to the weight per unit volume of ore extracted.
- Unconfined compressive strength (UCS) – is a laboratory determined parameter of the strength of the freestanding backfill material.

- Backfill curing period - UCS develops over time as a result of the hydration of a binder added to the backfill material. UCS values usually increase with curing time and are typically measured at 7, 28 and 56 days of curing.

- Liquefaction – occurs when the pore space between particles within a backfill mass is saturated with water and liquefies to a flowable mass in an uncontrolled fashion due to external “shock forces” such as those generated by mine production blasts or other seismic events.

Mill tailings for use as underground backfill have historically been used in two consistencies:

- As classified tailings. These have historically been called sand fill or “hydraulic fill”, in which the coarse fraction (sand) is hydraulically separated from the full tailings stream and used as backfill, while the fine fraction (slimes) are disposed of elsewhere.

- As the full, unclassified, tailings stream discharged from the mill. These may be dewatered to increase solids content and used to formulate what is commonly referred to as “paste backfill”.

In addition to the two consistencies named above and as conceived for the purpose of evaluation herein, the underground disposal of unaugmented whole “raw tailings” is also considered.

The previously mentioned particle size distribution of the Copperwood Project tailings has important implications to be considered for their use as potential backfill.

- Due to the “fineness” of the tailings, less than half of the tailings stream could be used as hydraulic sand fill. This is estimated to be at best, 40 wt% at 100 pcf of the full tailings stream (see Attachment 2). There would be detrimental implications due to the volume of “rejected fines” (approximately 60 wt% at a dry density of 50 pcf) that would still require disposal in a surface TDF. Due to the lower density of these fines, handling characteristics of such “fines” would require a larger diameter thickener, increased thickener retention times, and possibly increased TDF surface area and/or volumetric capacity to dispose of and contain the tailings in a reasonable manner. This option is considered later in this document.

- The particle size distribution and the results of the laboratory testing program conducted on the full tailings stream indicate that large quantities of binder (typically NPC) will be required to produce either “non-liquefying” or structural backfill. Binder consumption is generally related to particle size distribution, since inter-particle binding is directly related to surface area. The finer the particles, the greater the surface area to be bound, and the greater the quantity of binder required.

Unaugmented tailings as backfill or hydraulic sand without binder (which has negligible UCS), may be placed in underground workings only if it is confined and not exposed as a freestanding face. However, any contained water or water being exposed to the fill such as mine seepage must be allowed to drain or be pumped from the backfill mass in order to avoid excessive hydrostatic pressures on containment structures and the possibility of remobilizing (liquefaction) the saturated mass.

Paste backfill will only liberate small quantities of drainage water, with 80 to 90 percent remaining in the pores between particles. To this end, the backfill must be mixed with a sufficient quantity of binder to allow the binder to hydrate, thereby consuming the remaining water in the backfill mass and reducing the possibility of its remobilization, or liquefaction.

The failure to prevent the possibility of remobilization, or liquefaction of backfill can result in serious safety concerns since potentially thousands of tons of water and backfill could flow uncontrollably throughout the mine workings. Therefore, placement of tailings underground containing their full particle size distribution, without sufficient strength through binder addition to avoid liquefaction, is viewed as a significant and unacceptable safety risk. However, for the purpose of evaluation, containment of the disposed tailings susceptible to liquefaction have been considered only when disposal can occur downslope of mined out areas and when robust bulkhead dams are placed at points in the mine which could isolate liquefied tailings from active mine workings. These bulkhead dams may be designed and/or constructed in the mine workings and have been included in the cost estimates further in this document.

The design of the bulkhead dams is conceptual. Presently, mine pillars are planned at a width of 19 feet along strike across the mine. Bulkhead dams would include a barrier pillar approximately 39 feet wide which would remain in place. Access points to portions of the mine lower than these boundaries would include bulkheads which were structurally robust to behave as dams since the material contained would be filled to the area limits, would not be able to stand vertically, and may be subject to liquefaction. Accessing lower reaches of the mine through a limited number of bulkhead dams will result in higher ventilation, haulage, and pumping costs. The estimated cost of bulkhead dams include lost ore reserves, robust construction at access points, and higher operational costs due to limited access. These costs have been included in Table 5.0 for the backfill options which are subject to liquefaction.

Regarding the paste backfill and as a matter of practice, the tailings and binder must exhibit a UCS strength of 100 kilopascals (kPa) (or 14.5 pounds per square inch (psi)) at a 7-day curing time and a UCS strength of 170 kPa (25 psi) at a 28-day curing time. These values are the criteria that determine that the backfill has obtained sufficient strength and that the contained pore water has been eliminated thereby avoiding liquefaction (reference: Society for Mining, Metallurgy and Exploration, Inc. – Underground Mining Methods, Engineering Fundamentals and International Case Studies).

For paste tailings to be disposed of in the mine cavity, it would be necessary to alter/supplement the tailings. Paste used for backfill is considered an engineered backfill product and the processing plant that amends the tailings is referred to as a "paste plant".

3.1 Unaugmented Tailings Backfill

As estimated within the BFS, unaugmented tailings placed within a TDF will achieve an average density of 80 pcf. The size of the TDF presently designed for the Copperwood Project was based on that average density and the estimated 28.7 million tons of tailings produced. Based on recent (August 17, 2012) laboratory consolidation testing performed on the tailings by Golder Associates Inc. (Golder) under simulated TDF conditions, the estimated average dry density after consolidation was achieved, validating the assumption in the BFS. It is important to note, however, that tailings placed into the underground mine cavity would have a different density due to either the lack of consolidation underground (due to the limited vertical height of the mine relative to the TDF) or the addition of NPC as a binder. These tailings, as with the tailings pumped to a TDF, will have been processed through a thickener for initial dewatering. The dry density expected for unaugmented tailings placed underground is approximately 70 pcf (reference: Golder Draft Technical Memo - Orvana/Copperwood project– Consolidation Properties for Composite Tailings Sample, Dated August 17, 2012, see Attachment 3).

As shown on Figure 1, attached, the total volume of the mining cavity after mining Years 1 through 6 is approximately 136,760,036 ft^3, and the total volume of the cavity after mining Years 10, 11 and about half of Year 12 is approximately 79,908,470 ft^3. Therefore, total mine cavity volume available for backfill is 216.7 million ft^3. Based on an average dry density of 70 pcf for the unaugmented tailings, a total of 7.6 million tons of tailings could be placed into the mine cavity. As previously mentioned, the total amount of tailings is 28.7 million tons, which leaves 21.1 million tons for disposal in a surface TDF. Additional filling for Years 7, 8, 9, and 13 has not been considered due to the close proximity to the mine access portal and to allow for the mining of the bridge area connecting the two portions of the mine.

To place the tailings underground, it is theorized that mining would need to occur through about Year 10 if continuous backfilling is to occur. Starting backfilling at year 10 would allow for enough cavity volume to be available and to evacuate downslope space in Area 1 which would accommodate mine tailings. Should backfilling start earlier (after Year 4) backfilling would need to stop midway through approximately Year 7 and resume again in Year 12 to allow for enough cavity to be available to safely backfill with tailings. Tailings would have to be placed in a TDF for non-backfilling years, regardless. It should be noted that some of the mining, including the reduction in pillar size, as documented in the BFS, will be done in "retreat", thus delaying the timing that backfilling may commence. From the beginning of the project to that time, all tailings would be disposed of in the TDF since no safe space would exist underground for disposal. After approximately Year 4 (or approximately Year 10 for continuous filling), the raw, low solids and high water content, unaugmented tailings would be pumped from the thickener as they were generated into the lower portions of the cavity. As the cavity was filled, piping would be adjusted to consume the void space. Although it is not possible to fill all the void space due to irregularities in the mined surface which trap air, 100 percent of the cavity volume was used for the purpose of this comparison. Tailings placement may be staged such that as backfilling reached the constructed bulkhead dams, down-slope void space in Area 2 would available for backfill. As backfilling reached the constructed bulkhead dams in Area 2, tailings disposal would again be directed to the TDF for the balance of the mine life.

The costs associated with unaugmented tailings include: bulkhead dam costs for safety, costs for pumping the tailings directly to the mine cavity, and a surface TDF (21.1 million tons capacity, or 74-percent of the TDF cost) to accommodate the volume of tailings which cannot be placed underground due to volumetric constraints. The cost for surface disposal of the balance of the tailings which cannot be accommodated underground is estimated based on the percentage of the TDF required. Since the BFS reported a TDF which could accommodate 28.7 million tons and cost $102.3 million (as described in Alternative 5 of the Alternatives Analysis), a cost of $75.7 million (74 percent of $102.3 million) may be required to impound the balance of the tailings not disposed of underground.

Placing unaugmented tailings into the mine cavity would be an option that is not normally practiced in the mining industry. Safety issues arise when raw tailings are placed in an environment with mine workers, regardless of the efforts to bulkhead dam-off areas.

Although this practice is not commonly done, the cost associated with underground disposal, bulkheading portions of the mine, and constructing a TDF for the balance of the tailings which cannot be fit into the mine cavity is presented in Table 5.0, in Section 5.0.

3.2 Hydraulic Sand Backfill

It is estimated that the hydraulic sand fraction of the backfill will achieve an average density of approximately 100 pcf while the slimes fraction will achieve approximately 50 pcf upon deposition. The hydraulic backfill would likely be generated near the mill with the addition of cyclones for mechanical separation of the tailings. To place the hydraulic sand tailings underground, it is theorized that mining would need to occur through about Year 8 if continuous backfilling is to occur. Starting backfilling at year 8 would allow for enough cavity volume to be available and to evacuate downslope space which would accommodate mine tailings. Should backfilling start earlier, about Year 4, backfilling would need to stop midway through Year 8 and resume again in year 11 to allow for enough cavity to be available to safely backfill with tailings. Tailings would have to be placed in a TDF for non-backfilling years, regardless. At that point, the hydraulic backfill, or sand, separated from the tailings would be piped to the lowermost void space created in Area 1 and disposed of in the mine cavity. Water draining from the sand and draining into the mine cavity would be removed via temporary sumps and pumps to minimize the potential for backfill saturation and liquefaction. However, as with the unaugmented tailings, a bulkhead dam system at the upper portion of Areas 1 and 2 would be constructed for safety purposes. The remaining slimes would be processed before being piped to the TDF for disposal. The additional processing includes an estimated 50 percent larger thickener system and additional flocculant assumed to be required at a rate of 0.15 pounds (lb) per ton (reference: BFS).

As previously stated, the total mine cavity volume available for backfill is 216.7 million ft^3. Based on an average dry density of 100 pcf for the hydraulic sand backfill (see Attachment 3)), a total of 10.8 million tons of tailings could be placed into the mine cavity. As previously mentioned, the total volume of tailings is 28.7 million tons, which leaves 17.9 million tons of slimes (approximately 62-percent by weight of the total tailings) at a dry density of approximately 50 pcf remaining for disposal in a surface TDF. Due to the much lower density of the slimes (50 pcf versus 80 pcf of the unaltered tailings), no reduction of TDF volume is expected to be realized.

Additional cost would, however, be realized, above that estimated in the BFS, for the closure of the TDF containing large quantities of slimes. Since the slimes would not contain sand-sized particles which afforded the tails some minimal strength prior to hydraulic separation, additional materials and techniques would likely be required to close the TDF. These features engineered preliminarily for cost comparison include the addition of wick drains sunk into the slimes to assist in water management during closure, a geocomposite and linear gravel and pipe drains to evacuated the water liberated during placement of the cover.

Costs for the additional hydraulic separation and thickening equipment were referenced from the Feasibility Study and are shown in Table 5.0, in Section 5.0. Costs for thickening equipment and for the added TDF closure features including the wick drain, geocomposite, and gravel drains are also included in the costs presented in Table 5.0.

3.3 Paste Backfill

ORUSC commissioned Golder to characterize the tailings and to evaluate the necessary change in tailings strength (with the addition of binding augmentation agents) to allow disposal of paste backfill in the mine. The tailings samples were provided by ORUSC from a pilot plant metallurgical test commissioned by ORUSC with KD Engineering and METCON Research of Tucson, Arizona.

As previously stated, NPC is a commonly used binding agent, as is a blend of NPC and ground iron blast furnace slag (IBFS). The combination of NPC and IBFS as an augmentation binder agent is routinely used for base metal mines and was, therefore, considered by ORUSC and its consulting engineers.

Golder recently completed a laboratory evaluation for another mine backfill project also located in the Upper Peninsula of Michigan. In the course of conducting the evaluation, budgetary costs for binders delivered to the mine site were solicited from a major North American supplier. The cost to purchase the NPC is currently valued at approximately $100 per ton or more (reference: ENR.construction.com July 2012 Index Value of NPC without shipping is $107.40 per ton). The approximate cost estimate for NPC delivered to a jobsite (includes hauling) is $140 per ton, while a blend of ground IBFS and NPC (90:10 ratio) is $120 per ton, delivered (reference: Budgetary quote – Lafarge Canada).

Golder performed testing on the Copperwood Project representative tailings after adding the aforementioned binder agents: NPC and a blend of IBFS:NPC. The goal of the testing was to determine a suitable binder addition rate to yield UCS values generally considered acceptable at 7 days (14.5 psi) and 28 days (25 psi) of curing time, to avoid the liquefaction potential of paste backfill.

The binder augmentation rates selected were 3 and 5 wt% of the tailings based on dry unit weight (e.g. for every ton of tailings, this would amount to approximately 60 and 100 pounds of the binder added to paste backfill, respectively). The resulting paste backfill sample's UCS was measured using a Humbolt HM2800 digital load frame to ASTM Standard C-702.

The results of the paste backfill testing indicated that neither of the binders had cured sufficiently enough after 7 days to allow UCS testing to be carried out; and after 28 days of curing only samples containing a 5 wt.% addition rate of NPC had developed sufficient strength to allow UCS testing.

Although UCS results were obtained at 28 days of curing with the 5 wt% addition rate of NPC, the results were approximately 17 psi, which is insufficient for a 28 day curing period to avoid the potential for liquefaction.

A further evaluation was conducted which lengthened the curing period to 56 days in order to determine if there was evidence of "delayed curing". The paste augmented with the IBFS still did not develop sufficient strength to allow testing. The samples with 5 wt% NPC provided results but were still considered insufficient to avoid the potential for liquefaction due to long curing time necessary to exceed the liquefaction benchmark UCS values. In any event, the 56 day UCS of the this NPC augmented sample, was measured to be approximately 34 psi which is considered to be very poor and for most mines would not be acceptable for use as structural backfill.

Therefore, Golder concluded that neither binder type at the tested addition rates was effective in providing sufficient strength to avoid the potential for liquefaction of the tailings in an acceptable curing time, generally considered to be at 7 and 28 days of curing. Higher binder addition rates at values greater than 5 wt% would be required and the acceptable addition rates to attain the desired UCS results would require further laboratory assessment.

The additional costs associated with paste backfilling stem from three main expenditures: 1) construction of the paste plant; 2) maintenance of a paste plant and; 3) the purchase of NPC as a binding agent. The costs for these options are estimated below. The production rate or through-put of the paste plant was sized to accept the tailings produced during a year of maximum planned ore production. Based on the mine plan, the tonnage generated from development and mining will reach the maximum ore production rate at or after the third year of operation. Ore production rates planned for the Copperwood Project over the first four years are presented in Table 5-1, below, which is from the Alternatives Analysis.

Table 5-1. Copperwood Planned Development: Tons per Year 1-4

Year→	1	2	3	4
Tons per year	530,000	1,700,000	2,625,000	2,625,000
Cumulative Tons	**530,000**	**2,230,000**	**4,855,000**	**7,480,000**

Table 5-1 – From the Alternative Analysis, May 15, 2012.

Since the values in the table represent ore production, the tailings production may be estimated by multiplying the ore production by the ratio of total tailings produced divided by the total ore produced or 28.7 million tons divided by 30.3 million tons, or 0.95). As shown in Table 5-1, for each of years 3 and 4, the volume of material that the paste plant may be required to augment would be 2,486,000 tons (2,625,000 tons multiplied by 0.95) of tailings per year, prior to the addition of binding agent. Based on previous experience in the mine tailings augmentation field, the cost to design and construct a paste plant with the capacity appropriate for this project is approximately $20 million, which is based on Golder's database of similar paste plant projects. The cost to operate the plant on an annual basis, excluding the cost of the purchase and delivery of NPC, is estimated by the project engineer to be approximately $3.9 million per year.

As previously stated, the volume of tailings will exceed the void space created by mining. Since the cylinders cast for UCS testing in Golder's laboratory had an average tailings density of 107 pcf (112 pcf including the binding agent at 5 wt%), the void space created of 216.7 million ft^3 will accommodate 11.6

million tons of tailings. This will result in the remaining 17.1 million tons of tailings, 60% of the total tailing generated, being disposed of in the TDF.

Although the addition of a binding agent on a 5 wt% basis did not yield acceptable UCS values to avoid the potential for liquefaction, that ratio has been used as the basis for cost estimation purposes since the actual amount required will result in an even greater binder cost. Also, a conservative price of $100 per ton price for NPC referenced above was chosen. Therefore, the cost of the binding agent for the tailings placed into the mine is expected to exceed $58 million (with an additional $11.6 million per 1 wt% binder added above the 5 wt% estimated). If the paste plant operated at capacity, the plant life would be approximately 4.7 years (11.6 million tons divided by 2,486,000 tons per year) since the entire mine cavity available for backfilling would be filled at that time. Therefore, the cost of the backfilling operation including the cost of the plant ($20 million), the cost to operate the plant (4.7 years at $3.9 million per year results in a total of $18.3 million), and the 5 wt% of NPC binding agent ($58 million) would be an additional $96.3 million total, as compared to the option of not backfilling the mine.

Under the scenario where these expenditures are made and paste backfill is used, another supplementary disposal option for the tailings would still be required to accommodate the approximately 17.1 million tons of tailings, 60% of the total generated, that could not be placed underground due to spatial constraints. Since the total cost of surface disposal for 28.7 million tons is estimated to be $102.3 million (as described in Alternative 5 of the Alternatives Analysis), a cost of $61.4 million (60 percent of $102.3 million) may be required to impound the balance of the tailings not disposed of underground. The total cost of tailings disposal, therefore, would include the underground disposal costs for a portion of the tailings ($96.3 million) and on-surface disposal costs for the balance of the tailings ($61.4 million), for an estimated total tailings disposal expense of $157.7 million.

Also note that this cost represents a 5 wt% binder addition only, each additional 1 wt% which may be required to avoid the potential for liquefaction adds $11.6 million to the disposal costs. Table 5.0, below, presents the cost estimate summary for the paste backfill.

4.0 ESSENTIAL MINING FUNCTIONS - TDF

At the beginning of the proposed mining operation, sufficient evacuated volume will not exist in the underground mine cavity to accommodate the tailings generated. The underground workings, even if backfilled completely, would require a supplemental surface TDF alternative to accommodate tailings. As presented in the original Alternative Analysis and permit application and further explained in Section 3, a TDF is the only feasible and prudent option. In addition to tailings disposal, a TDF is required on-site to contain groundwater pumped from the mine for years 0 through 3 of mining operations. As the mine is developed, groundwater will seep into the mine at an approximate rate of 150 to 404 gallons per minute as per the BFS. This seepage will be collected through a series of sumps, pumped to the surface, and then stored within the TDF. The TDF will also accommodate runoff contact water from the ore stockpile and contact water from the mill/process plant area. The accumulated water in the TDF will be re-used in the milling process and, in later years of operation, treated by a WTP and discharged to the receiving environment.

The TDF will be used for water storage of accumulated mine groundwater inflow and precipitation prior to the WTP completion, as the construction of the WTP is delayed until the beginning of year four in the present mine plan. The timing of the construction of the WTP will decrease the amount of water withdrawal volumes needed from Lake Superior, and it will also serve to delay the required capital expenditure for construction and therefore phase some of the expenses for the Copperwood Project overall.

A TDF is also required to store mined ore prior to mill startup since the mill will need a substantial volume or feed of ore (2,486,000 tons per year) to begin processing. An enclosed facility is desirable to contain runoff from the stockpiled mined ore prior to mill startup. The TDF is ideal for this application since the space available will allow for a large accessible stockpile and the runoff from precipitation that may fall on the stockpile will be contained by the TDF.

The TDF will have additional advantages for the mine site from a water storage perspective. Temporary storage of water within the TDF will be required to limit the use of fresh water necessary to operate the mine and therefore minimize the withdrawal of water from Lake Superior. As detailed in the draft *April 12, 2012 Copperwood Underground Copper Mine, Water Treatment Feasibility Design and Cost Estimates*, the mine process water requirement will begin in year one at 883 gallons per minute (gpm) and increase to 1,107 gpm by Year Five. The process water requirement will continue at about this rate through year 13. The TDF will supply up to 622 gpm during the first year, increasing to an estimated 747 to 846 gpm for years five through thirteen. The complete water balance included in Table 7, below, from the draft *April 12, 2012 Copperwood Underground Copper Mine, Water Treatment Feasibility Design and Cost Estimates* report.

Table 7: Water Balance Results

Time	Year	Ore Production	Cumulative Ore Production	Tailings Solids to TDF	Mill Deficit	Average TDF Water to Mill	Average Makeup Water to Mill	Treated Water Discharge to Receiving Environment	Treated Water Reclaimed to Process Mill	Average Rate of Water Treatment	TDF Water to Receving Environment
		[Mtonm/yr]	[Mtonm]	[ft^3/yr]	[gpm]	[gpm]	[gpm]	[gpm]	[gpm]	[gpm]	[gal]
7/1/2014	0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
10/1/2014	1	2.12	2.12	22.74	882.71	622.10	260.61	0.00	0.00	0.00	-
2015	2	1.70	3.82	18.19	706.13	441.98	264.15	0.00	0.00	0.00	-
2016	3	2.63	6.45	28.05	1089.20	682.88	406.33	0.00	0.00	0.00	-
2017	4	2.63	9.07	28.36	1100.20	659.55	440.63	62.17	287.83	350.00	32,006,000
2018	5	2.63	11.70	28.55	1106.80	747.11	359.72	99.07	250.93	350.00	51,192,000
2019	6	2.63	14.32	28.32	1098.60	759.02	339.60	104.46	245.54	350.00	54,069,000
2020	7	2.63	16.95	28.55	1106.80	768.22	338.61	106.14	243.85	350.00	55,079,000
2021	8	2.63	19.57	28.43	1102.70	757.08	345.65	100.93	248.93	350.00	52,186,000
2022	9	2.63	22.20	28.55	1106.80	846.85	259.99	166.10	181.66	350.00	86,506,000
2023	10	2.63	24.82	28.55	1106.80	837.79	269.05	168.70	174.07	350.00	87,944,000
2024	11	2.63	27.45	28.32	1098.60	828.86	269.76	166.30	173.94	350.00	86,907,000
2025	12	2.63	30.07	29.00	1123.30	844.45	278.82	163.30	182.43	350.00	85,054,000
2026	13	2.33	32.40	25.75	997.32	654.12	343.20	149.24	115.43	286.64	57,982,000
10/1/2026	13							286.64	0.00	286.64	95,846,000
2027	14							264.30	0.00	264.30	138,250,000
2028	15							272.59	0.00	272.59	143,100,000
2029	16							283.50	0.00	283.50	148,500,000
2030								264.54	0.00	264.54	99,935,000
10/1/2030	17										

Upon initial startup of the mill, water accumulated in the TDF from mine dewatering and storm water will reduce the volume of water required to be withdrawn from Lake Superior. Also during the first three years of mine operation, the water inputs (precipitation, Lake Superior, and mine) will balance with the TDF's storage capability and return rate, requiring no water to be treated and discharged. As mining continues, tailings placed into the TDF will actually become a significant water source for the milling process. This is achieved through the consolidation and dewatering of the tailings within the TDF. Following deposition, solids within the tailings settle due to gravity and release pore water. The TDF design includes the provision to decant the excess pore water and pump it back to the mill for reuse, therefore the TDF allows for the conservation and reuse of water. This contained approach and the recycling/reuse of water will result in minimizing potential water withdrawal impacts of the mining operation on the waters of Lake Superior.

The cost for the TDF has been referenced in the Alternatives Analysis and is shown in Table 5.0.

5.0 COST SUMMARY

Table 5.0 Tailings Backfill Cost Summary

Item Description	Option 1 - Unaugmented Tailings Backfill Cost	Option 2 - Hydraulic Sand Tailings Backfill Cost	Option 3 - Augmented Tailings Backfill Cost	TDF	Notes/References
Bulkhead Costs	$48,000,000	$48,000,000	Bulkheads required but costs negligible compared to Options 1 and 2	not applicable	Lost revenue from copper left in place, ventilation, hauling, pumping, and safe access included, from Orvana
Added WTP Costs	$4,600,000	$7,900,000	8,400,000	not applicable	Costs at approximately 1 cent per gallon based on estimated seepage and backfill timing.
Additional Tailings Pumping Costs	$288,000	$288,000	Not applicable, is included in paste plant total cost.	not applicable	Additional 3,000 feet of piping, 2 additional pumps and appurtenances; costs from Bankable Feasibility Study, prepared by KD Engineering, January 2012
Additional Cyclone System	not applicable	$794,000	not applicable	not applicable	Includes cyclone feed box, feed pump, cyclone cluster, screens and sampler, and piping; costs from Bankable Feasibility Study, prepared by KD Engineering, January 2012.
Larger Thickener System	not applicable	$2,000,000	not applicable	not applicable	Includes a 50% larger tails thickener, mechanism, pumps, piping, earthworks, electrical and structural elements; costs from Bankable Feasibility Study, prepared by KD Engineering, January 2012.
Additional Flocculant	not applicable	$11,600,000	not applicable	not applicable	For slimes management cost from the Bankable Feasibility Study, prepared by KD Engineering, January 2012, rate of 0.15 lb flocculant per ton of tailings, $4.50 per pound. Based on 17.2 million tons of slimes for Option 2.
Paste Plant	not applicable	not applicable	$20,000,000	not applicable	Total
Paste Plant O&M	not applicable	not applicable	$18,330,000	not applicable	$3.9 million per year for 4.7 years
Binder Addition Minimum	not applicable	not applicable	$57,500,000	not applicable	5 wt.% minimum at $100 per ton, based on 11.5 million tons of Paste for Option 3.
TDF Disposal Required	$75,700,000	$124,460,000	$61,380,000	$102,300,000	Option 1: ~74% of Original TDF cost of $102,300,000 Option 2: Based on ~98% of Original TDF cost and includes additional reinforced closure cap costs (wick drains, geogrid) for dealing with soft tailings. Option 3: ~60% of Original TDF cost of $102,300,000
Total Cost	**$128,588,000**	**$195,042,000**	**$165,610,000**	**$102,300,000**	Conceptual Level Cost

Note: Certain detailed cost estimates have not been included for brevity.

Since backfilling in Areas 1 and 2 would consume the entire cavity from the lowest portions of the mine to the possible bulkhead dam locations, that area could not be occupied by mine seepage water. Accordingly, water which accumulated in Areas 1 and 2 of the mine after year 4 would be required to be evacuated from the mine. This water would likely be pumped to the surface and treated in the WTP. The estimated cost of treating the water seeping into mine void space has been included for each backfill option in Table 5.0. In preparation of the estimate, the length of time that the void space may remain open without backfill due to the backfill density and the expected tailings production rates, has been considered.

6.0 CONCLUSION

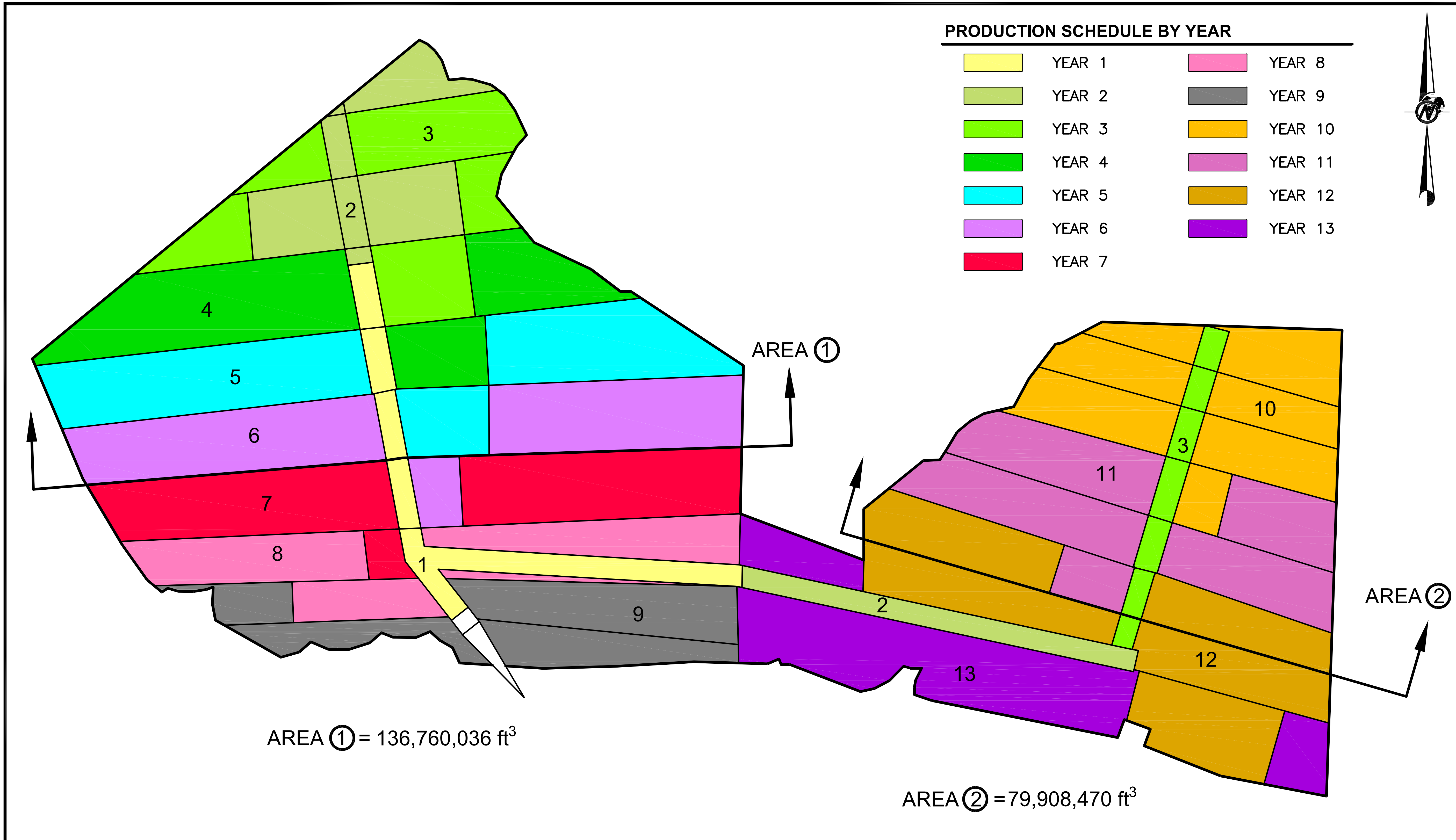
Table 5.0 presented in Section 5.0 presents a summary of the costs associated with the below ground disposal alternatives considered and includes the cost of the surface TDF for comparison. The table illustrates that none of the below ground disposal options will be as economical for the project as a TDF constructed on surface. Moreover, the least costly underground alternative evaluated, which exceeds the cost of the TDF by more than $26,000,000, is uncommon in the mining practice and may result in an unacceptable level of risk for human life. Additionally, a TDF which is at least 60% of the size of the planned TDF will be required regardless of the underground disposal option considered. For the reasons presented in this memorandum, disposal of tailings underground for the Copperwood Project is neither feasible nor prudent.

ATTACHMENTS:

Attachment 1 - Figure 1

Attachment 2 – Golder Draft Technical Memorandum ‘Interpretation of Tailings Dewatering Assessment – Revision 1’, dated March 30, 2012.

Attachment 3 - Golder Draft Technical Memo - Orvana/Copperwood project– Consolidation Properties for Composite Tailings Sample, Dated August 17, 2012.



NOTE: VOLUMES SHOWN ARE BASED ON PREDICTED ORE PRODUCTION RATES VERSUS AREAS.

Golder Associates Great Lakes

PROJECT
ORVANA RESOURCES
ALTERNATIVES ANALYSIS

TITLE
VOLUME OF MINE CAVITY AVAILABLE

PROJECT No. 113-88649		
FILE No. 11388649F001		
REV. 0	SCALE AS SHOWN	
DESIGN	TDJ	08/28/12
CADD	JJS	08/28/12
CHECK		
REVIEW		

FIGURE 1

DATE March 30, 2012 **PROJECT No.** 113-88649 / 003B Revision 1

TO Mr. Don Poulter, PE
Golder Associates Inc.

FROM Bruno Mandl, P.Eng. / Pierre Primeau, P.Eng **EMAIL** bmandl@golder.com / pprimeau@golder.com

**RE: ORVANA MINERALS CORP., COPPERWOOD PROJECT
INTERPRETATION OF TAILINGS DEWATERING ASSESSMENT**

1.0 BACKGROUND AND OBJECTIVES

The Paste Engineering and Design group of Golder Associates Ltd. was asked by Golder Associates Inc. (collectively known as Golder) to conduct laboratory testing to determine the dewatering and paste characteristics of tailings from the Orvana Minerals Corp. (Orvana) Copperwood Project in Michigan, USA.

Tailings and process water samples originating from mill process testing conducted by KD Engineering and METCON Research were received by the Golder Sudbury Laboratory at the beginning of January 2012. The tailings samples were combined and homogenized, and used with the received "Process Water" in subsequent testing.

The tests results discussed in this memo are summarized in the attached "Laboratory Assessment on Dewatering of Tailings".

2.0 HAZARD ASSESSMENT

The tailings and decant water were assessed for concentrations of: volatile organic compounds; hydrogen cyanide gas; hydrogen sulphide gas; and heavy metals.

All concentrations were found to be within acceptable levels according to Golder's laboratory Health and Safety protocols, allowing the dewatering testing to proceed.

3.0 TAILINGS MATERIAL CHARACTERIZATION

The measured pH values of the tailings solids (8.7) and decant water (7.2) of the bulk flotation slurry samples were corroborated as being in the anticipated range by Orvana. The measured pH of the process water (7.0) that accompanied the slurry samples was also in the range of expected values.

The particle size distribution (PSD) of the tailings sample with a D80 of 54 microns, as measured by Golder, compared well to the target grind D80 of 65 micron provided by METCON. The measured PSD was also corroborated by Orvana as being in the expected range.

It is interesting to note that the Copperwood tailings PSD is considered to be significantly finer than those found in Golder's database for base metal tailings. Generally speaking, tailings having a PSD with a minimum of 15 wt% passing 20 micron are amenable to produce a paste-like consistency. In the case of the Copperwood tailings, approximately 46 wt% is finer than 20 microns thus satisfying the PSD criteria.

Specific gravity (SG) tests were conducted in accordance with the ASTM D854 standard. The SG of the tailings solids was measured to be 2.8 and considered typical for tailings having the observed mineralogy.

Mineralogy and chemistry tests were carried out for the following purposes:

1) To confirm that the sample as tested falls within anticipated mineralogical variations of the ore;
2) To allow comparison should future testing occur; and
3) To identify whether based on historical data, the tailings contain mineralization that has proven problematic to dewater and/or achieve target strength.

Mineralogy and chemistry tests were conducted on the tailings solids.

The primary mineralization of the tailings is quartz (>30 wt%) which has not historically proven problematic in the formulation of paste. However, there is a large component of mica group minerals (>20 wt%) in the tailings which has the potential to be problematic in the tailings dewatering (filtration) process and can negatively impact the quantity of binder necessary to attain the desired unconfined compressive strength (UCS) for mine backfill.

The sulphur content of the tailings, as determined through the LECO furnace method, is very low at 0.14 wt%. For tailings with a few wt% sulphur, the use of a binder containing ground iron blast furnace slag (BFS) and normal Portland cement (ratio of 90:10) can prove beneficial in obtaining target UCS with lower binder content. There are however, at times, exceptions due to interactions with other chemical constituents in tailings where the use of a binder containing BFS can also be beneficial where there is no sulphur present. For this reason, this binder mixture was also tested.

The remaining minor mineralogical components are typical of other base metal tailings historically tested by Golder originating from the Pre-Cambrian Shield.

4.0 DEWATERING TESTING

4.1 Settling Tests

Dewatering/settling tests were carried out on the tailings sample. The tests were completed in an iterative fashion to determine optimum values involving: flocculant type; flocculant dosage; and feed slurry solids concentration. Different flocculant dosage and feed rates were evaluated in 500 mL, 1 L, and 4 L vessels.

The dewatering tests conducted by Golder are centered on the use of non-ionic and anionic flocculants. Cationic flocculants are not normally tested due to their potential harmful effects on fish species should they be released into the aquatic environment.

The dewatering/settling results suggest the following:

- At a typical dosage rate and lower than average feed solids concentration, all the flocculants tested in the 500 mL vessels: provided poor overflow clarity; very low initial settling rates; and average floc formation. The non-ionic flocculant was completely ineffective; therefore, two of the better performing anionic flocculants were selected for further evaluation.
- The 500 mL vessel tests were conducted once again utilizing AN 945 SH and AN 926 VHM flocculants, and a doubling of the initial flocculant dosage rate applied in one and two stages. In an attempt to improve overflow clarity, pH adjustment with lime was also examined. Overall, the best results were obtained with two stage flocculation using AN 945 SH. It provided an improved overflow clarity, an acceptable settling rate, and a potentially improved underflow density.
- Feed solids concentration, and dosage methods and rates, were refined using the AN 945 SH flocculant in 4 L setting tests. These tests provided reasonable results with a flocculant dosage rate of 50 g/tonne in a two staged approach (25 g/tonne per stage) and a feed solids concentration of 12.5 wt%.
- Settling rate was observed to be average, and the time to enter the compression zone of the settling curve was slightly longer than typically observed for most base metal mine tailings.
- Underflow densities observed in the larger 4 L cylinders were roughly 8% higher than those observed in the 1 L vessels. At a flocculant dosage rate of 50 g/tonne applied in two stages and 12.5 wt% solids feed, an underflow density of 51 wt% and 54 wt% was obtained after 2 and 24 hours of retention time, respectively.
- Centrifuge tests, which indicate the maximum solids concentration through gravitational means, provided a solids concentration of 69 wt%.

As a general summary, the tailings will dewater acceptably using available technology. A further improvement of as much as 5 wt% is often observed in production scale thickeners when compared to the bench scale 4 L tests 24 hour results.

4.2 Filtration Testing

Vacuum filtration tests, to simulate the results associated with disc filters, were conducted at a vacuum level of 20" Hg.

Typical cycle times (a full rotation) for vacuum disc filters are in the range of 90 to 120 seconds. The vacuum filtration testing program usually begins with an initial feed solids concentration of 5 wt% greater than the underflow density measured in the 4 L settling tests, followed by the observed concentration in the 4 L tests and finally 5 wt% less than the 4 L test results.

The cake loading observed at 59 wt% solids slurry concentration was very poor, even when compared to results obtained for gold tailings with a higher concentration of fines. Due to the poor cake loadings and experience indicating that even poorer results would be observed with lower slurry feed solids concentration, no further vacuum filtration testing at lower solids concentrations was undertaken.

The primary factor believed responsible for the poor vacuum filtration results is the significant presence of the platy mica minerals in the slurry that are causing rapid blinding of the filter cloth.

5.0 RHEOLOGICAL CHARACTERIZATION

5.1 Slump vs. Solids Content

Paste is generally defined as having a measurable slump within the range of 7" to 10". The solids content over this slump range was measured. The solids concentration throughout this range is anomalously high compared to most tailings having a similar PSD, SG and mineralogy.

With a difference in moisture content of nearly 2.5 wt% over the paste slump range, the tailings are considered not to be very sensitive to changes in water additions as indicated by the steep slope of the slump curve, implying that a lesser degree of control will be required to achieve the desired slump. This observation will be used in the paste preparation process design.

5.2 Static Yield Stress

Static yield stress is a measure of the amount of energy needed to initiate flow. The static yield stress results are comparable to those historically observed with base metal tailings. The nature of the curve demonstrates that above 72 wt% solids content, the yield stress increases exponentially. For instance, a yield stress value of approximately 225 Pa is observed at a 10" slump, and climbs rapidly, to 580 Pa at a 7" slump. The relatively low sensitivity of the tailings to water addition to control slump indicates that adjusting the slump could be a means of maintaining a desired pumping pressure when delivering paste underground.

5.3 Water Bleed and Plug Yield Stress

The water bleed properties and plug yield stress of the material demonstrate typical characteristics of fine grain tailings.

The water bleed over 24 hours for both the 7" and 10" slump are very low, demonstrating that the material has a high affinity for water. The values are lower than those typically measured for fine grained tailings. As with filtration, it is believed that the combined effect of platy mineralogy in the form of mica, and the fine PSD are responsible for the observed results.

Due to the small amount of water bleed, the yield stress climbed only marginally over a 24 hour period. However, the difference in yield stress between a 7" and 10" slump material is not typical, being greater than two-fold.

Plug yield stress tests were conducted to determine the degree of consolidation of paste when left at rest over time. The yield stress remains relatively constant with depth and here too may be related to the presence of platy minerals in the mica family.

The plug yield stress results suggest paste can be allowed to remain idle in pipelines for a reasonable period of time, but as a general guideline and due to presence of cement binder in the paste, this should be limited to less than two hours. Clear procedures must be put in place to allow the pipelines to be cleared effectively when unexpected shutdowns occur.

5.4 Viscosity and Dynamic Yield Stress Tests

The Bingham yield stress provides an indication as to the amount of energy required to maintain flow once in progress. With most paste tailings, the Bingham yield stress for paste at a given solids content is usually lower than the static yield stress for the same solids content. This is to say, once flowing, less force is required to maintain rather than to initiate flow. The Copperwood tailings exhibit nearly the same stress characterises for both instances; meaning the same force is needed to maintain, as needed to initiate flow.

5.5 Unconfined Compressive Strength Testing

Unconfined compressive strength (UCS) testing was carried out on paste with 7" and 10" slump consistencies, to determine the achievable strengths with differing binder types and addition rates that would support the project's underground mining operations.

Two typical binder addition rates were selected, 3 and 5 wt%, and two binder types, normal Portland cement (NPC) and a blend of ground iron blast furnace slag (BFS) and NPC in a ratio of 90:10, respectively.

BFS:NPC was selected for testing since past experience has demonstrated cost savings when this type of binder is used in base metal tailings applications where there is an adequate concentration of sulphide mineralization in the tailings.

Poor UCS results were observed:

- Neither of the binders that were tested cured sufficiently after 7 days to allow UCS determinations;
- After 28 days of curing, the cylinders cast using a 5 wt% addition rate of NPC cured sufficiently to allow UCS determination but the results, based on Golder's database of similar tailings, were very poor;
- After 56 days of curing, a doubling of the 28-day UCS results were noted for the cylinders cast with a 5 wt% addition rate of NPC, however the results at 56 days of curing are still considered poor based on previous experience with base metal tailings; and
- The cylinders cast using BFS:NPC 90:10 had not cured at all after 56 days.

6.0 CONCLUSIONS AND RECOMMENDATIONS

The following conclusions can be drawn from the laboratory assessment:

- Copperwood tailings can be initially dewatered through the use of available thickener technology;
- Filtration of tailings typically used to produce a consistent paste quality will require some form of pressure filtration rather than the more conventional vacuum filtration techniques;
- Copperwood tailings are amenable to the formulation of paste using the afore mentioned technologies;
- The UCS results are poor and require additional assessment such as increasing the NPC binder addition rate and/or the formulation of a paste/crushed waste rock paste blend to enhance UCS results and reduce binder consumption; and
- A doubling of the UCS values using NPC binder between 28 and 56 days of curing implies that delayed strength development is occurring with the Copperwood tailings and that this bears investigation for longer curing times.

It is speculated that the presence of higher than usual concentrations of platy mineral from the mica family are the source of the anomalous results noted in this assessment program.

The anomalous results, however, may be due to other contributing factors that should be investigated further such as:

- Detailed analysis of the water chemistry used in the UCS testing program;
- Detailed analysis of chemistry and mineralogy of the Copperwood tailings; and
- The use of other formulations of binders.

Golder has the ability to conduct further investigations as noted above through the use of in-house experts in the field of binder performance.

It is recommended that further investigations be conducted in a laboratory assessment program to determine the results of higher NPC binder addition rates, the addition of crushed aggregate to the paste mixture, and ultimate UCS results for curing periods greater than 56 days.

Should you have questions regarding the findings and interpretation of the testing, please contact the undersigned.

GOLDER ASSOCIATES LTD.

Bruno Mandl, P.Eng.
Senior Mining Engineer

Pierre Primeau, P.Eng.
Senior Process Engineer

BM/PP/ds

Attachment: Laboratory Assessment

March 29, 2012

LABORATORY ASSESSMENT ON DEWATERING OF TAILINGS FOR

Orvana Minerals Corp. Copperwood Project

Submitted to:
Mr. Don Poulter
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44 Union Boulevard, Suite 300
Lakewood, Colorado
USA 80228

REPORT

Report Number: 113-88649

Distribution:

1 e-Copy: Golder Associates Inc., Lakewood, Colorado

1 Copy: Golder Associates Ltd., Sudbury, Ontario

Golder Associates

Study Limitations

This report was prepared for the exclusive use of Orvana Minerals Corp. (Orvana) on the Copperwood Project. The report, which specifically includes all tables, figures and appendices, is based on measurements and observations made and data and information collected during the laboratory studies conducted by Golder Associates Ltd. (Golder) for Orvana. The test results are based solely on the ambient conditions of the laboratory at the time the measurements and tests were conducted.

The services performed, as described in this report, were conducted in a manner consistent with that level of care and skill normally exercised by other members of the engineering and science professions currently practicing under similar conditions, subject to the time limits and financial and physical constraints applicable to the services.

The sample(s) provided for the tests are assumed to be representative of material found at the site. The test data given herein pertains to the sample(s) provided, and may not be applicable to material from other production periods or zones. Assessment of the sample environmental conditions and possible hazards associated with the material composition is based on the results of chemical analysis of samples which are possibly from a limited number of locations. However, it is never possible, even with exhaustive sampling and testing, to dismiss the possibility that part of a site or a production line may remain undetected. The results found from the tests may not be reproducible under the field conditions.

The report is of a summary nature and is not intended to stand alone without reference to the instructions given to Golder by Orvana, communications between Golder and Orvana, and to any other reports prepared by Golder for Orvana relative to the specific site described in the report, tables, drawings, figures and appendices.

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The findings and conclusions of this report are valid only as of the date of this report. If new information is discovered in future work, Golder should be requested to re-evaluate the conclusions of this report, and to provide amendments as required.

Table of Contents

Table of Contents (cont'd)

TABLES

FIGURES

Table of Contents (cont'd)

1.0 INTRODUCTION

Golder Associates Inc. has retained Golder Associates Ltd., (collectively known as Golder) to carry out a laboratory assessment on the Copperwood Project tailings to assess dewatering, rheological, and strength characteristics for the purpose of evaluating options for surface disposal as well as underground mine backfill.

2.0 SAMPLE RECEIPT AND PREPARATION

2.1 Sample Receipt

Samples received by Golder's Sudbury laboratory are summarized in Table 1. All samples were received in good condition with all seals intact. The total weight of the shipment was 540 lbs. The samples were shipped via Ceva Logistics, Mississauga.

Table 1: Sample Receipt Summary

Date	Amount/Container	Label as Received	Golder Sample ID
January 06, 2012	2 – 10 gallon containers	M-785-02 -Tailing from Bulk Rougher Flotation Test - Slurry	113-88649 Copperwood Total Tails
	1 – 10 gallon container	M-785-02 -Tailing from Bulk Rougher Flotation Test - Process Water	113-88649 Process Water

All samples received by Golder are subjected to material property characterization tests to establish properties and allow for comparison should future testing be required.

2.2 Hazard Assessment

Prior to handling the samples each container was assessed separately for hazardous gases. The gas analysis results are presented in Table 2.

Table 2: Sample Hazard Assessment

Date	Label as Received	Golder Sample ID	VOC (ppm)	HCN (ppm)	H_2S (ppm)
January 06, 2012	M-785-02 -Tailing from Bulk Rougher Flotation Test - Slurry	113-88649 Copperwood Total Tails	0	0	0
	M-785-02 -Tailing from Bulk Rougher Flotation Test - Process Water	113-88649 Process Water	0	0	0

VOC: Volatile Organic Compounds
HCN: Hydrogen Cyanide gas
H_2S: Hydrogen Sulphide gas

Metals analysis using Inductively Coupled Plasma with a Mass Spectrometer detector (ICP-MS) was performed on a composite sample obtained via individual pipe samples from each pail. This testing helps to identify Health and Safety hazards such as heavy metals which may be present. The sample was sent to an external laboratory for ICP-MS analysis. Figure 1 and Appendix A present the results.

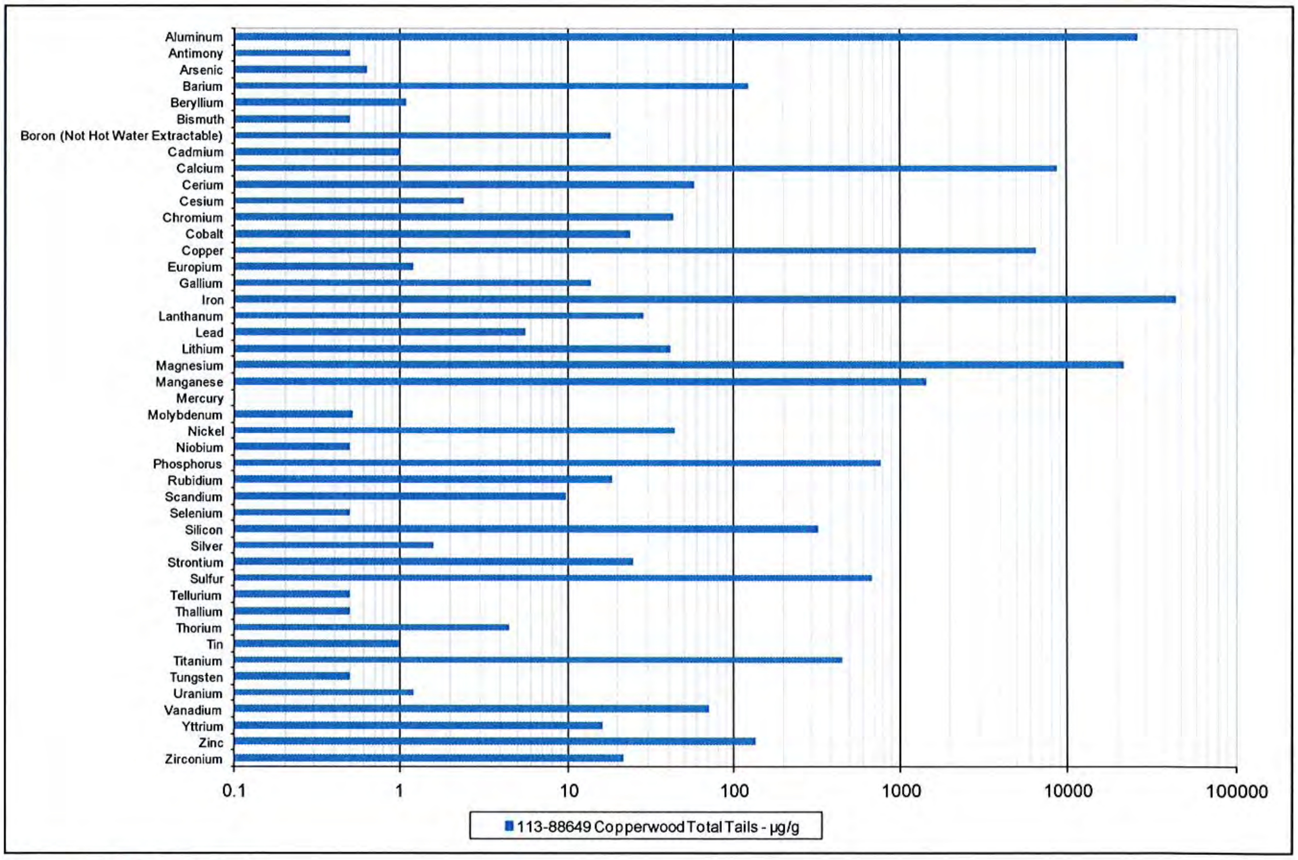

Figure 1: ICP-MS Results

No hazardous gases were detected in any of the samples. The concentrations of heavy metals present in the samples were considered to be acceptable to handle according to Golder's established protocols.

2.3 Sample Preparation

Proper sample preparation technique is a critical first step to ensure proper homogenization of solids, representative sub-sampling and reproducibility of results.

The first step was the separation of clear supernatant water from the 113-88649 Copperwood Total Tails samples. The remaining dewatered solids were combined, homogenized, and treated as one sample for the remainder of the test program.

3.0 MATERIAL CHARACTERIZATION

3.1 pH Analysis

Table 3 presents the pH of each sample and the temperature at which it was measured.

Table 3: pH Analysis

Sample	pH	Temperature (°Celsius)
113-88649 Copperwood Total Tails	8.7	21
113-88649 Copperwood Total Tails Decant Water	7.2	20
113-88649 Process Water	7.0	19

3.2 Particle Size Distribution

Particle size distribution (PSD) was determined using mechanical sieving and a Fritsch laser particle size analyzer according to ASTM D4464.

Specific values are presented in Table 4. The gradation parameter DXX, tabulated in microns, refers to the average particle diameter that XX% by weight of material is smaller than.

Table 4: Particle Size Distribution

Sample	D10 (µm)	D30 (µm)	D50 (µm)	D60 (µm)	D80 (µm)
113-88649 Copperwood Total Tails	2	9	22	29	54

Figure 2 presents the results.

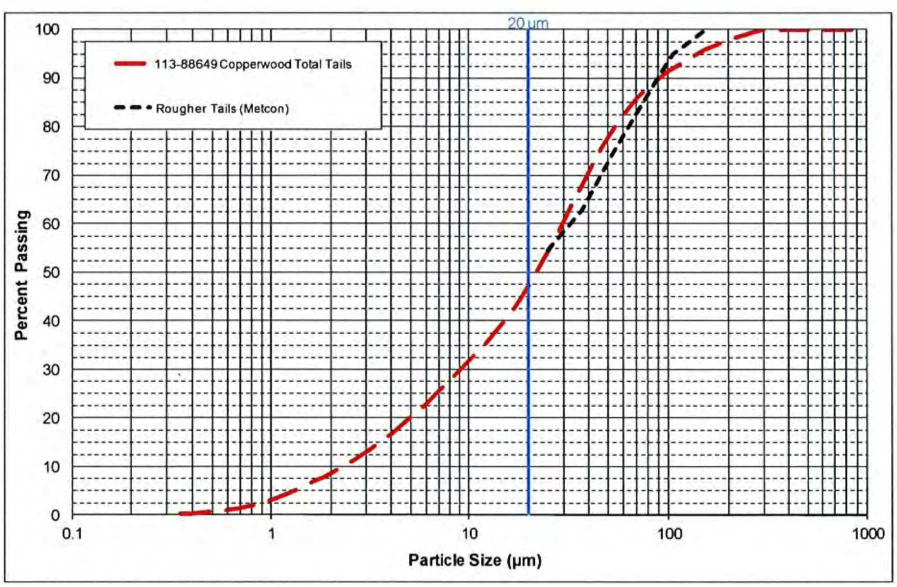

Figure 2: PSD Results

3.3 Specific Gravity

The specific gravity (SG) of the sample was determined using vacuum de-aired water. Each slurry sample was also vacuum de-aired prior to SG measurement. The results are presented in Table 5.

Table 5: Specific Gravity Results

Sample	Trial 1	Trial 2	Average
113-88649 Copperwood Total Tails	2.8	2.8	2.8

3.4 Chemistry and Mineralogy

Chemical and mineralogical analyses were performed using whole rock analysis (WRA) by inductively coupled plasma (ICP) and X-ray diffraction (XRD) via semi-quantitative analysis by Rietveld Method, respectively whereas sulphur analysis was performed by LECO method. The results are presented in Table 6 to Table 8 as well as Figure 3 and Figure 4.

Table 6: Chemical Composition (wt%)

Sample	Al_2O_3	BaO	CaO	Cr_2O_3	Fe_2O_3	K_2O	LOI	MgO	MnO	Na_2O	P_2O_5	SiO_2	SrO	TiO_2	Total
113-88649 Copperwood Total Tails	14.35	0.06	1.46	0.01	9.04	3.24	5.15	4.23	0.18	1.56	0.21	59.8	0.01	1.09	100.39

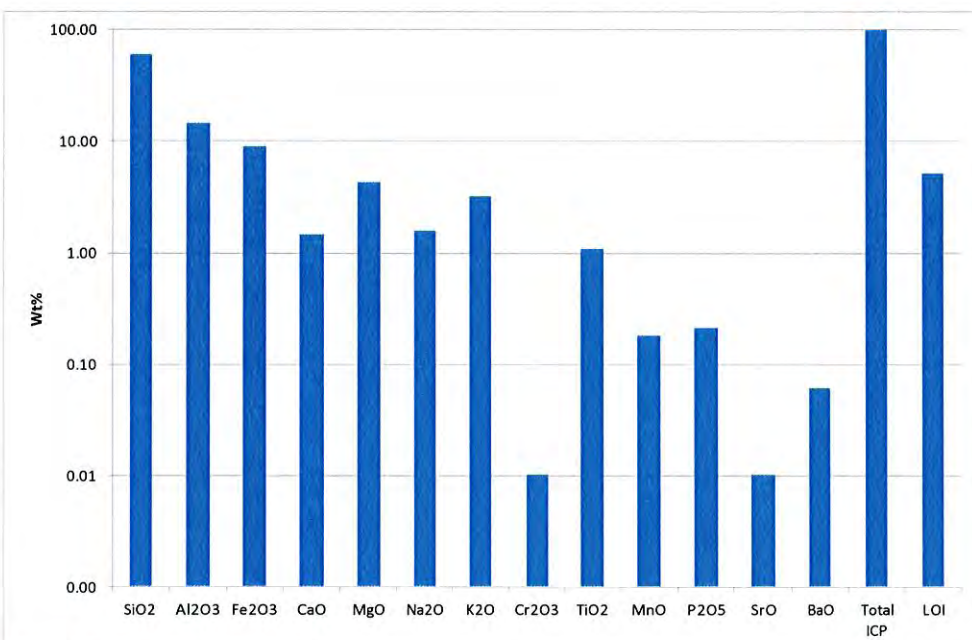

Figure 3: Chemical Composition

Table 7: Semi-quantitative Mineralogical Composition 113-88649 Copperwood Total Tails

Mineral SQ-XRD	Chemical Formula	Wt%
Quartz	SiO_2	30.53
Muscovite	$(K,Na)Al_2(Si,Al)_4O_{10}(OH)_2$	20.23
Chlorite	$(Mg,Al)_6(Si,Al)_4O_{10}(OH)_8$	18.53
Albite	$NaAlSi_3O_8$	17.37
Orthoclase	$KAlSi_3O_8$	11.22
Hematite	Fe_2O_3	2.12
Total		100.00

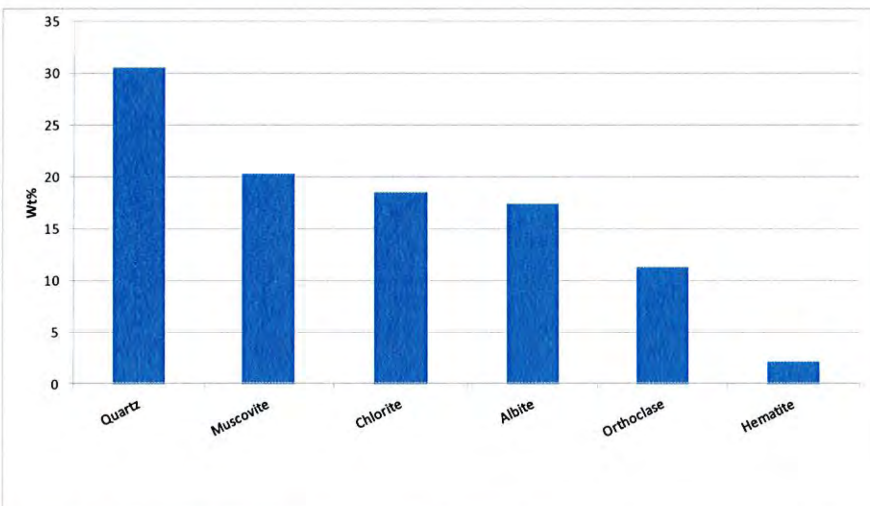

Figure 4: Semi-quantitative Mineralogical Composition

Table 8: LECO Sulphur Analysis

Sample	Sulphur (wt%)
113-88649 Copperwood Total Tails	0.14

4.0 DEWATERING TESTING

4.1 Dewatering Summary

Dewatering tests were performed using settling, centrifuge, and vacuum filtration techniques. A summary of all tests illustrating the resulting densities in weight percent solids (wt% solids) is presented on Figure 5.

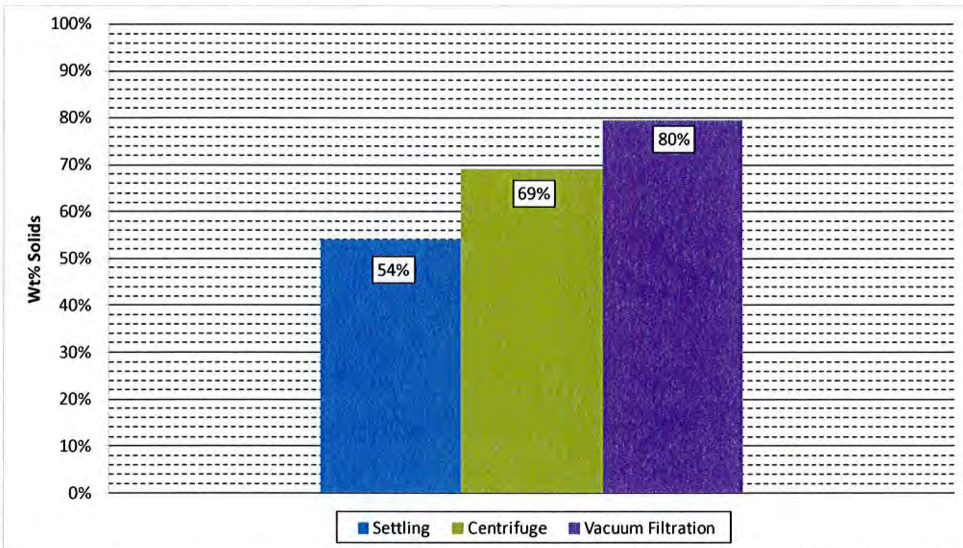

Figure 5: Dewatering Summary

4.2 Settling Tests

The first stage of settling tests usually consists of an assessment of the potential for thickening through use of synthetic polymer flocculants. Several types of flocculants were screened in order to test a range of parameters such as charge density and molecular weight. The typical types of flocculants considered are the anionic and non ionic polymers. Table 9 indicates the flocculants which were examined and their properties.

Table 9: Flocculant Properties

Flocculant	Manufacturer	Type	Ionicity (mole %)	Molecular Weight
AN 920 VHM	SNF	Non - ionic	0	Very High
AN 905 VHM	SNF	Anionic	5	Very High
AN 926 VHM	SNF	Anionic	25	Very High
AN 945 SH	SNF	Anionic	40	High
AN 977 SH	SNF	Anionic	70	High
MF 919	BASF	Anionic	46	Very High

To select the most effective flocculant, several factors were examined, such as initial settling velocity, overflow clarity, flocculant size structure and strength, as well as underflow density. The overflow clarity was measured in NTU (Nephelometric Turbidity Unit) where lower numbers indicate clearer water. The screening results are presented in Table 10 as well as Figure 6 and Figure 7, whereas Figure 8 shows a photo of the screening cylinders. All screening tests were performed in 500 mL cylinders.

Table 10: Flocculant Screening Results

Flocculant	Dosage (g/tonne)	Feed Solids (wt% solids)	Overflow Clarity after 60 minutes (NTU)	Initial Settling Rate (m/hour)	Calculated Underflow Density (wt% solids)	Floc Size	Floc Structure
AN 920 VHM	25	5	N/A	N/A	N/A	N/A	N/A
AN 905 VHM			705	8	48	Small	Normal
AN 926 VHM			555	10	53	Medium	Normal
AN 945 SH			496	7	58	Medium	Normal
AN 977 SH			525	4	55	Small	Normal
MF 919			513	6	58	Small	Normal

N/A: AN 920 VHM did not flocculate and was eliminated from further consideration

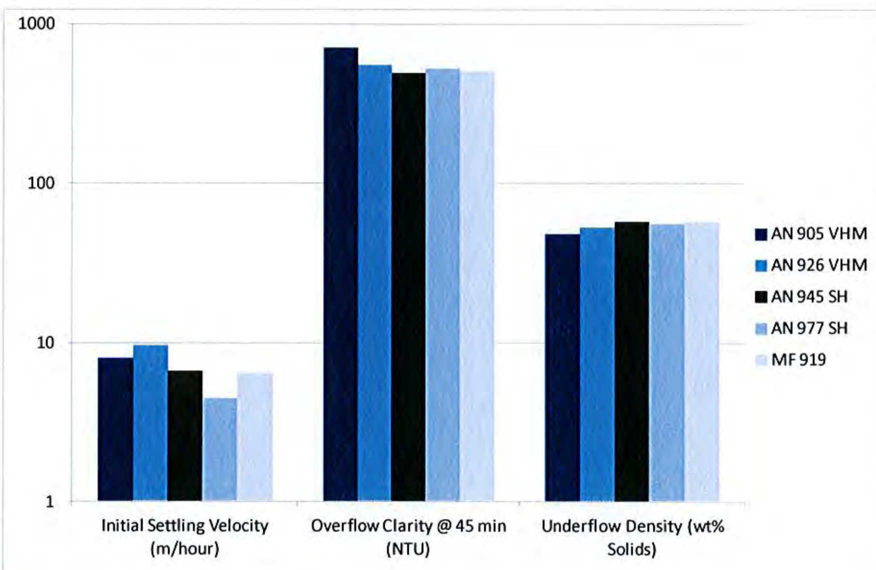

Figure 6: Flocculant Screening Results

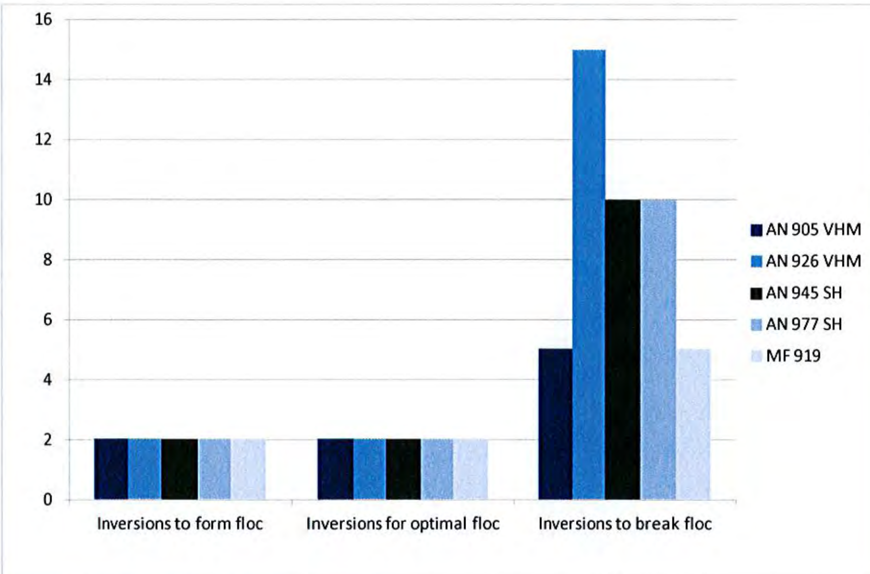

Figure 7: Flocculant Strength Results

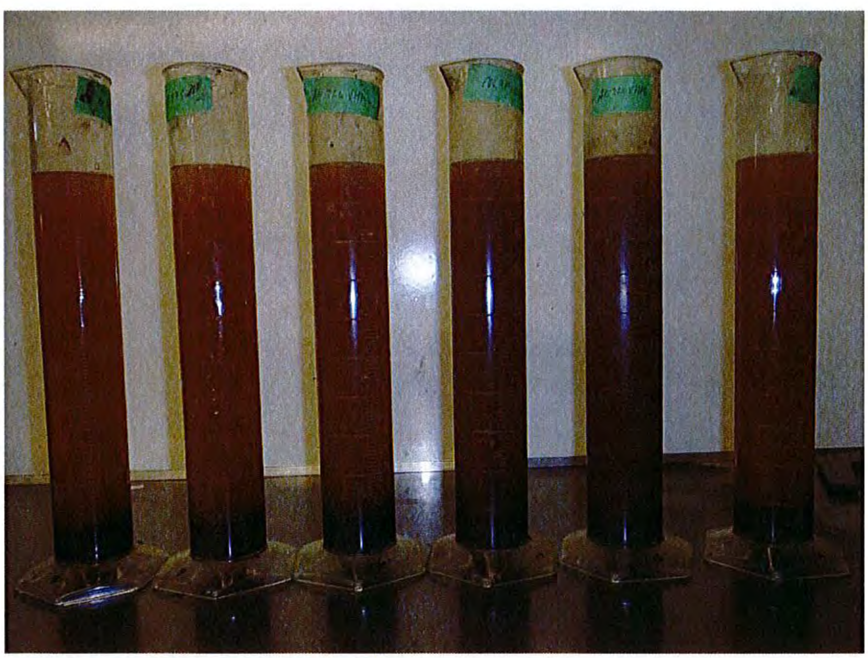

Figure 8: Flocculant Screening Cylinders

The initial screening results did not clearly identify which flocculant was the best, therefore additional screening was performed. The flocculants chosen for this stage of screening consisted of AN 945 SH and AN 926 VHM since they gave the best results for underflow solids and settling velocity respectively. The flocculant dose was increased in this stage in an attempt to increase settling velocity. In addition, two-stage flocculation as well as pH modification was investigated in order to improve the overflow clarity. The two-stage flocculation was conducted by adding half the flocculant dosage and waiting 30 seconds to add the remaining flocculant. Table 11 and Figure 9 show the second phase screening results.

Table 11: Flocculant Screening Results - Phase 2

Flocculant	Dosage (g/tonne)	Feed Solids (wt% solids)	No. of Addition Stages	pH	Overflow Clarity after 60 minutes (NTU)	Initial Settling Rate (m/hour)	Calculated Underflow Density (wt% solids)
AN 926 VHM	50	5	1	7.8	433	50	57
AN 945 SH			1	7.8	477	40	59
AN 945 SH			2	7.8	216	57	64
AN 945 SH			1	10	677	85	61
AN 945 SH			1	11	63	98	57

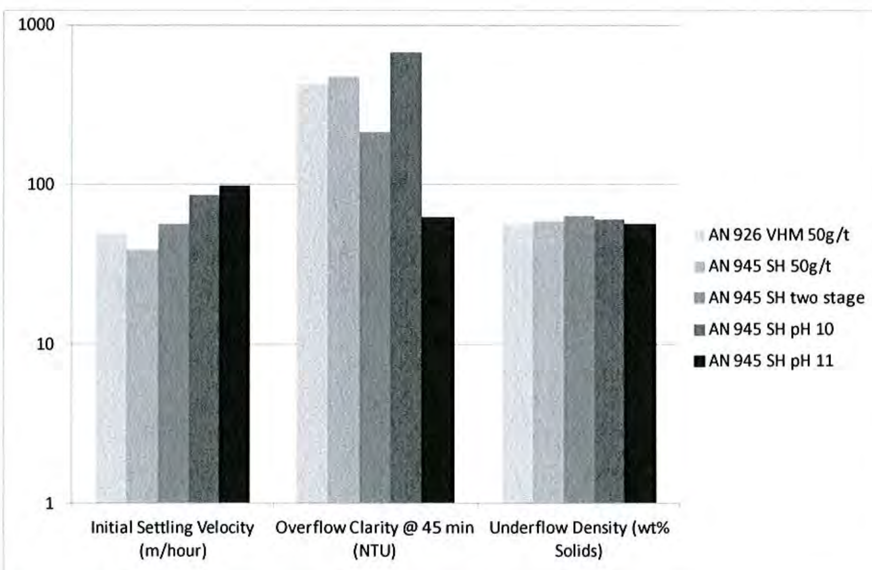

Figure 9: Flocculant Screening Results - Phase 2

Based on the screening results, AN 945 SH using two-stage flocculation was chosen as the best flocculant for dewatering 113-88649 Copperwood Total Tails. The next step in the settling tests was to optimize flocculant dosage and feed solids concentration. The dosage represents the amount of flocculant (polymer) in grams for each tonne of solids. These tests were carried out in 1L cylinders. Underflow solids were raked after settling for 60 minutes. The results are summarized on Figure 10, Figure 11, and Figure 12.

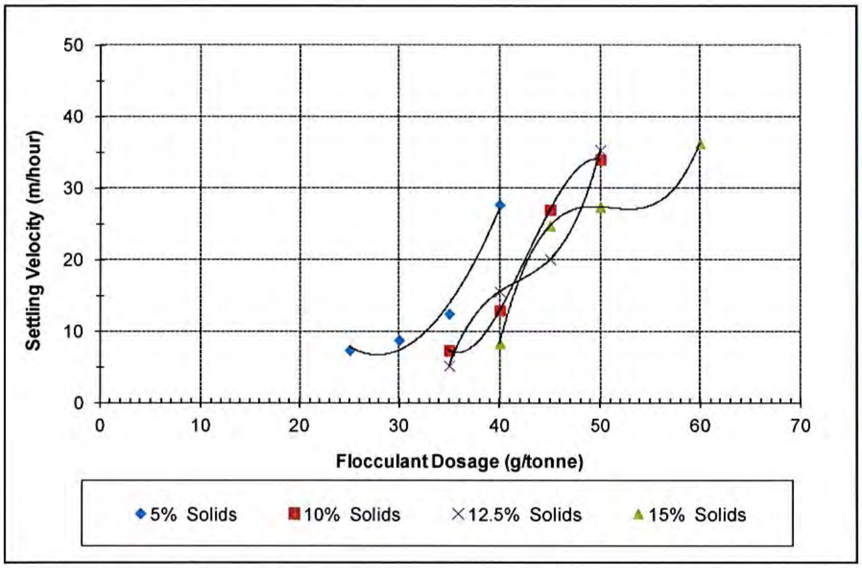

Figure 10: Settling Velocity vs. Flocculant Dosage at Varying Feed Solids Concentration

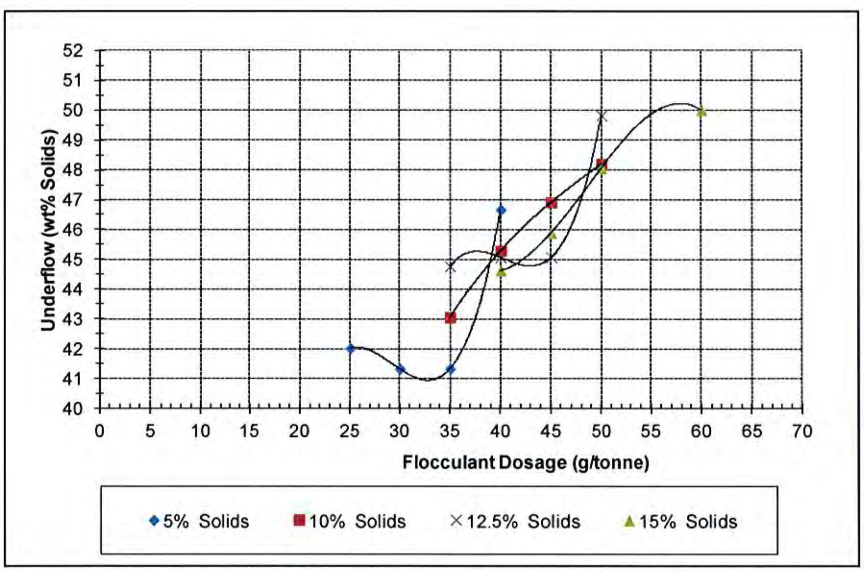

Figure 11: Underflow wt% Solids vs. Flocculant Dosage at Varying Feed Solids Concentration

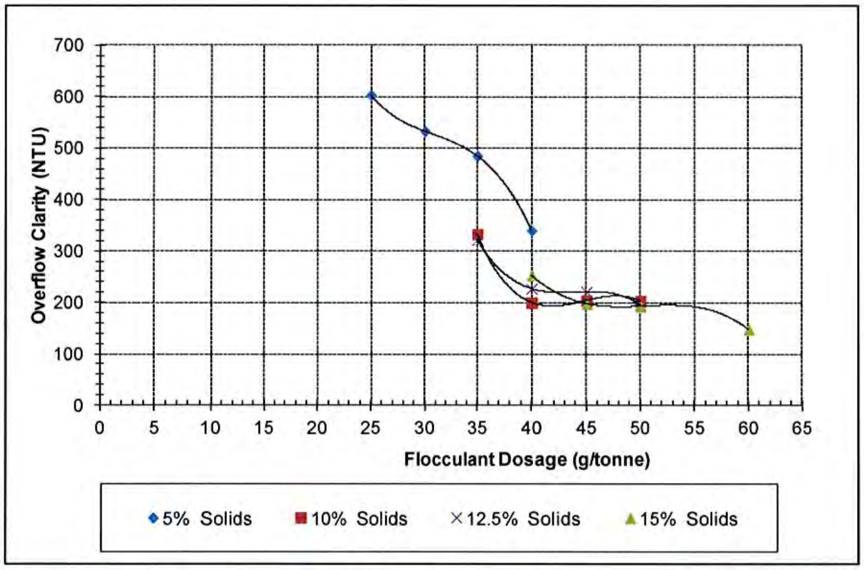

Figure 12: Overflow Clarity vs. Flocculant Dosage at Varying Feed Solids Concentration

The optimal conditions from the screening results were carried forward to larger scale 4L tests to more accurately determine the underflow density. These tests are considered static, bench scale and actual underflow solids, depending on thickener technology, may differ.

Based on the results obtained during the screening process, it was determined that a feed concentration of 12.5 wt% solids and a two-stage flocculant dosage of 50 g/tonne (25 g/tonne per stage) would provide good settling characteristics for the 113-88649 Copperwood Total Tails sample. Figure 13 presents the settling curve, whereas Table 12 provides a summary and Figure 14 shows a photo of the settling cylinder.

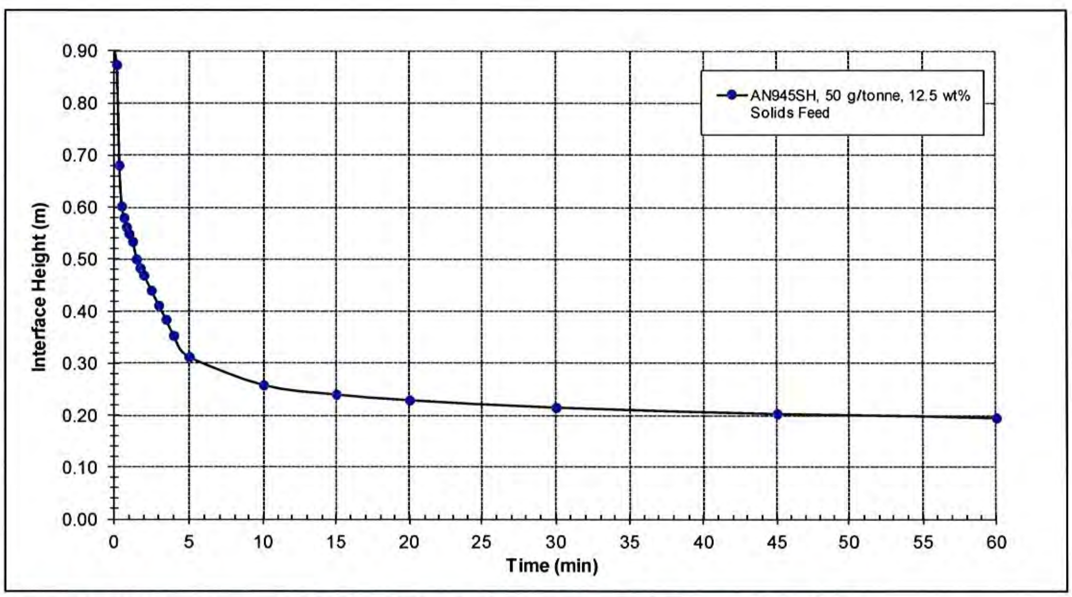

Figure 13: Settling Curve, 12.5% Solids Feed Concentration, 50 g/tonne (two-stage)

Table 12: 4L Settling Summary

Flocculant Dosage – first stage (g/tonne)	Flocculant Dosage – second stage (g/tonne)	Feed Solids (wt% solids)	Overflow Clarity after 60 minutes (NTU)	Calculated Underflow Density after 1 hour (wt% solids)	Calculated Underflow Density after 2 hours (wt% solids)	Calculated Underflow Density after 24 hours (wt% solids)	Measured Underflow Density after 24 hours (wt% solids)
25	25	12.5	260	44	51	54	54

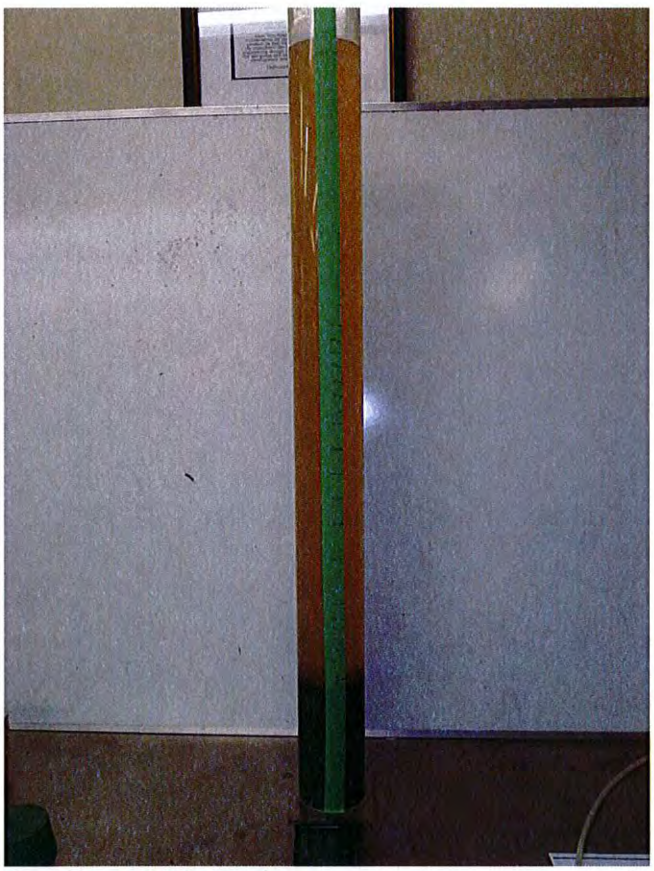

Figure 14: 4L Settling Cylinder

4.3 Centrifuge Testing

Samples of the underflow solids from the 4L settling tests were centrifuged until no further change in density was observed. This testing provided the 'phi max' value, which related to the maximum density achievable by use of high compression gravimetric settling methods. Table 13 presents the results.

Table 13: Centrifuge Results

Flocculant Dosage - first stage (g/tonne)	Flocculant Dosage - second stage (g/tonne)	Feed Solids (wt% solids)	Measured Underflow Density from 4L Settling (wt% solids)	Calculated Underflow Density from Centrifuge (wt% solids)
25	25	12.5	54	69

4.4 Filtration Testing

Tests were conducted to evaluate vacuum filtration as a possible dewatering treatment option for paste production. A range of sample consistencies are typically tested using the underflow density results from the 4L settling tests as the intermediate wt% solids to examine. Additional consistencies of sample at 5 wt% greater and 5 wt% lower are usually tested. However in this instance, filtration on the thickest sample was considered poor, therefore the consistencies corresponding to the 4L underflow as well as 5 wt% lower were not assessed.

The filter leaf was equipped with a small section of industrial grade polypropylene felt filter cloth. The leaf was immersed into the slurry and simulated production scale vacuum filters where the sectors dipped into the slurry in an agitated filter tank as the disc rotated. Proper technique and cycle times simulating continuous filters provide an estimate of cake loading, moisture, and discharge characteristics.

Since the test was performed in the laboratory, under ideal conditions, actual loading is multiplied by 0.7 to reflect variable or upset conditions which may occur in plant operations.

The following parameters were used for testing:

Elevation:	617 feet asl
Vacuum level:	20" and 26" Hg
Temperature:	68°F
Filter cloth:	Industrial grade polypropylene felt 133-03 (25-40 cfm rating)
Apparatus:	100 mm (4 inch) diameter, dip style filter head

The results are presented in Table 14 as well as on Figure 15 and Figure 16.

Table 14: Filtration Results - 59 wt% Solids

Vacuum Level ("Hg)	Cycle Time (sec)	Cake Thickness (inch)	Cake Loading ($lb/ft^2/hr$)	Cake Moisture (wt%)	Final Density (wt% Solids)
20	30	0.07	34	22	78
	45	0.06	27	20	80
	90	0.09	18	21	79
	135	0.09	13	20	80
	180	0.11	12	20	80
26	90	0.10	19	21	79

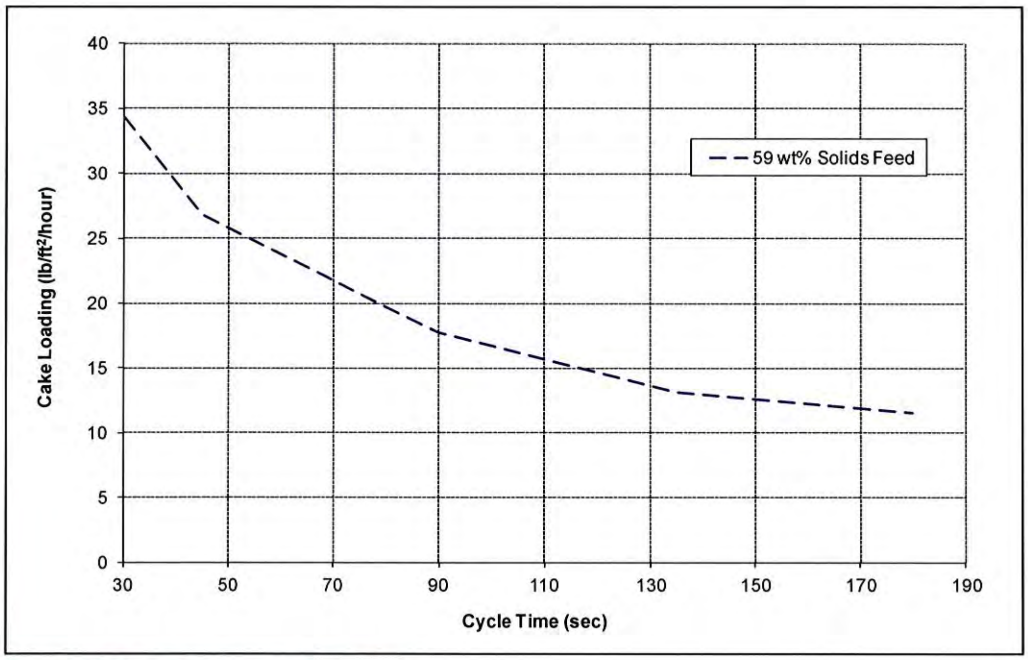

Figure 15: Cake Loading vs. Cycle Time

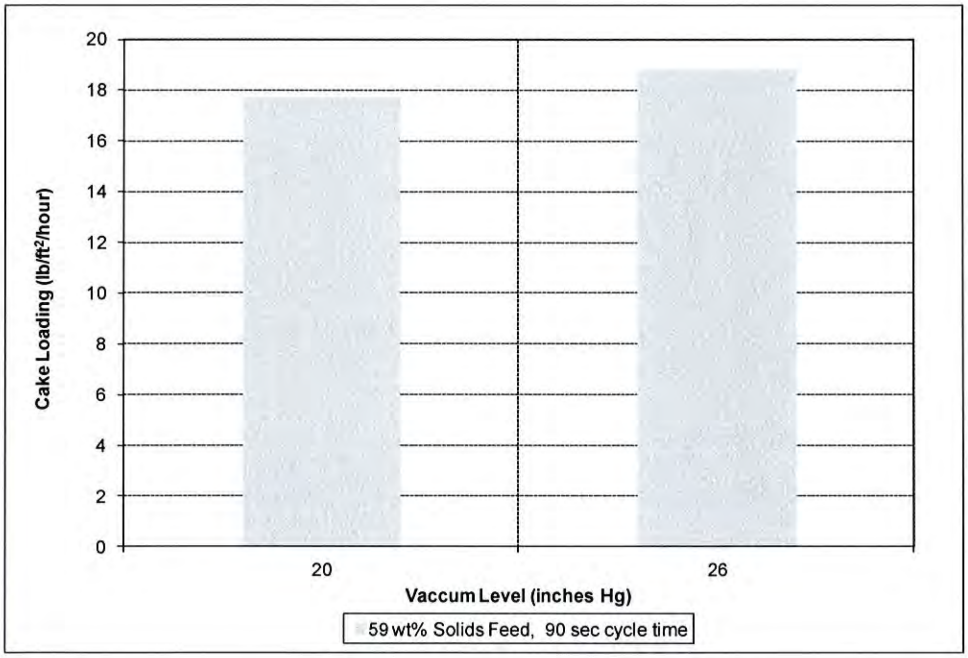

Figure 16: Cake Loading vs. Vacuum level

5.0 RHEOLOGICAL CHARACTERIZATION

Rheological testing was carried out to evaluate flow and handling properties. These tests provide an indication regarding the material's behaviour in the course of mixing, slump adjustment, pumping, flowing, and also while sitting idle. Rheological characterization provides data for the selection of process equipment such as mixers, pumps, and pipelines.

5.1 Slump vs. Solids Content

To gauge sensitivity to water additions, small increments of water were added to the bulk sample. After each addition, slump and solids content was determined. This generates a relationship between slump and solids content which is typically used to determine the degree of process control required to maintain slump control of the final product. The results are presented on Figure 17 and photos of the 7" and 10" slumps are shown on Figure 18.

Figure 17: Solids Content vs. Slump

Figure 18: Left: 7" Slump, Right: 10" Slump

5.2 Static Yield Stress Testing

Yield stress is defined as the minimum force required to initiate flow. Static yield stress was determined by using a very slow moving (0.2 RPM) vane spindle attached to a torque spring. The spindle was immersed in the sample, and measurements were taken at various solids contents. There are different test methods to determine yield stress, one termed 'static' and the other 'dynamic'. Figure 19 presents the static yield stress testing results.

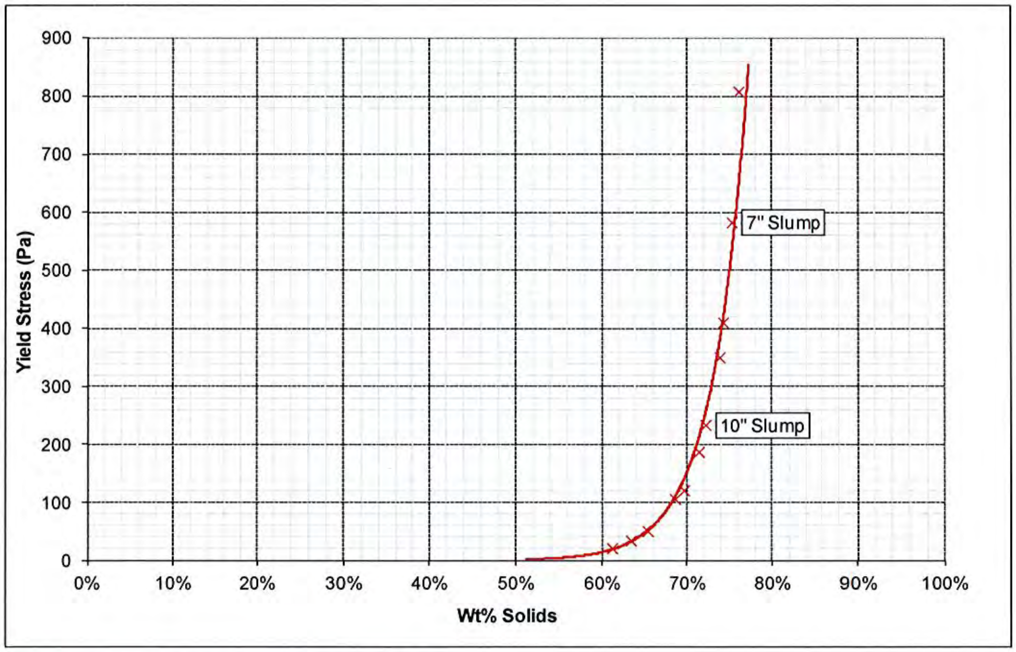

Figure 19: Static Yield Stress vs. wt% Solids

5.3 Water Bleed and Yield Stress vs. Time

Moisture retention testing was carried out to assess the water bleed properties of the paste while sitting idle in test beakers. Two slump consistencies were tested at four time intervals. At each time interval the water bleed and yield stress were measured. Figure 20 presents the results.

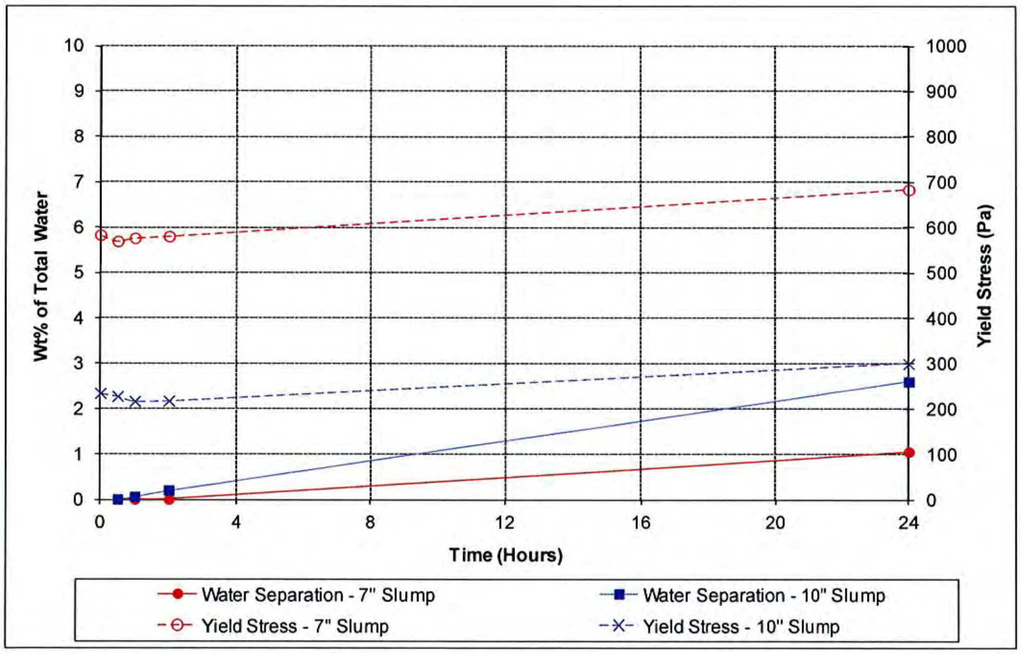

Figure 20: Water Bleed and Yield Stress vs. Time

5.4 Plug Yield Stress

Plug yield stress analysis was performed to determine if consolidation has occurred throughout a cross-section of idle paste material, as may be present in a pipeline's cross-section. Two slump consistencies of material were allowed to sit idle for two hours, and a specially designed vane spindle was immersed at three depths to measure yield stress. Figure 21 presents the results.

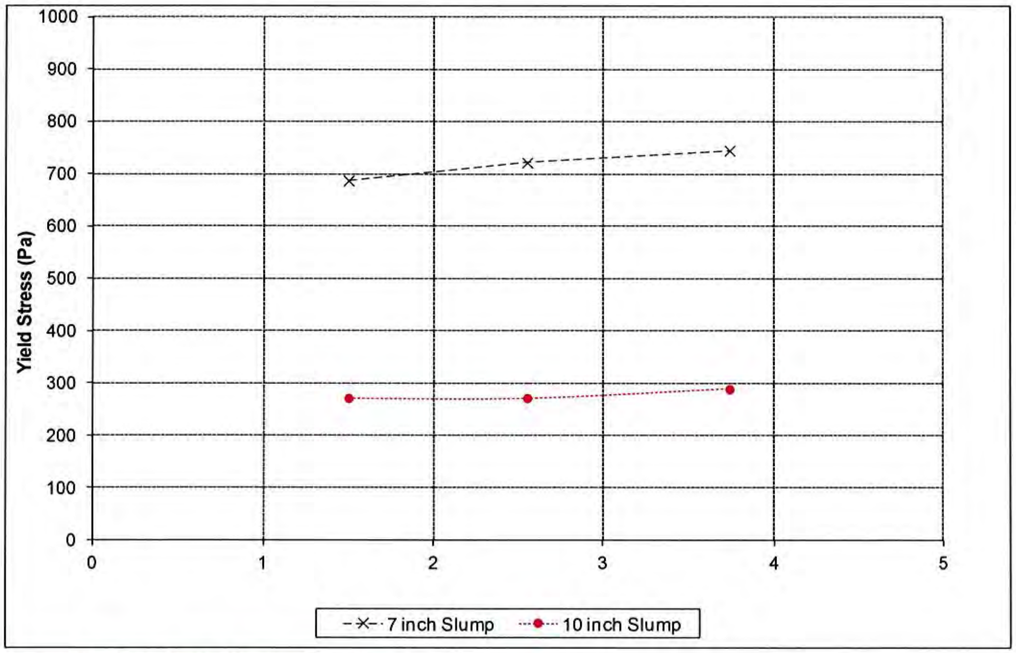

Figure 21: Plug Yield Stress Results

5.5 Viscosity and Dynamic Yield Stress Determination

Viscosity testing provides bench scale flow properties and fluid characterization. Dynamic viscosity and yield stress data is essential for mixer, pump, and pipeline design. In order to compare or duplicate viscosity results of non-Newtonian fluids, it is important to test according to the same conditions. Test conditions and parameters such as cycle time and instrument sensor configuration are critical to producing usable data from bench scale viscometers.

The yield stress determined through this testing is referred to as dynamic yield stress since it is extrapolated from dynamic shear stress data to zero shear. The instrument sensor or bob rotated inside the cup which contained the sample and torque measurements were recorded at several incremental speeds or shear rates.

The rheograms are presented in Appendix B and summarized test results are presented in Table 15 as well as on Figure 22 and Figure 23.

Table 15: Bingham Viscosity and Yield Stress Summary

Wt% Solids	Bingham Yield Stress (Pa)		Bingham Viscosity (PaS)	
	Ramp Up	Ramp Down	Ramp Up	Ramp Down
74	445	515	0.72	0.56
73	328	369	0.49	0.39
72	227	257	0.35	0.27
71	169	181	0.24	0.21
69	101	108	0.14	0.13
65	46	48	0.06	0.06
61	20	21	0.03	0.03

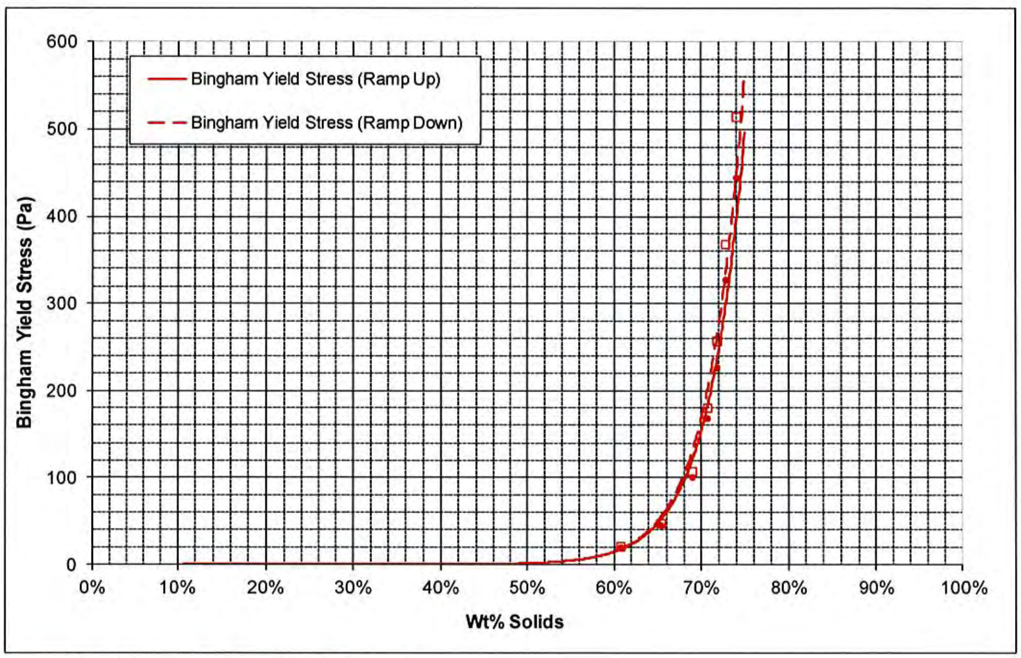

Figure 22: Bingham Yield Stress Results

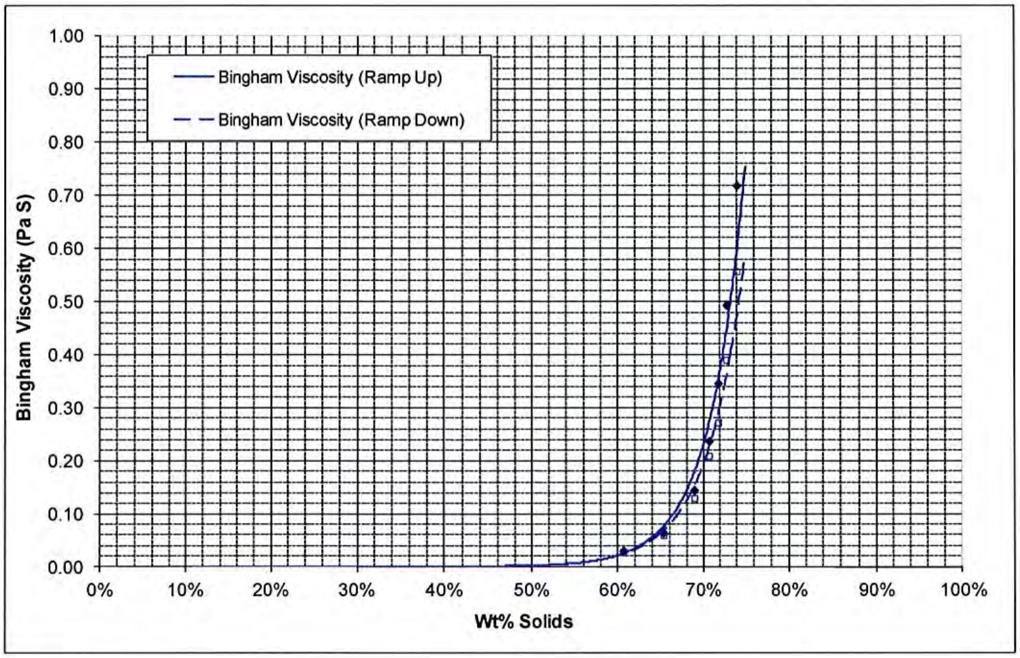

Figure 23: Bingham Viscosity Results

6.0 UNCONFINED COMPRESSIVE STRENGTH TESTING

Unconfined compressive strength (UCS) testing was carried out using a Humboldt HM2800 digital load frame. The loading was measured using s-type load cells. Depending on strength, either a 10 kN or 45 kN (2,000 lb or 10,000 lb) load cell are utilized.

The cured cylinder was placed between two platens and during testing the bottom platen advanced at a rate of 2 mm (0.08 inch) per minute. The load was continuously monitored and the peak load was automatically recorded by the instrument.

6.1 UCS Program and Results

The UCS program was carried out to assess the backfill strength using 2" x 4" cylinders. The cylinders were cured in a high humidity environment maintained at 68 to 77°F. Three cylinders per curing period were cast and the results were averaged. The test program is presented in Table 16 and the results are presented in Table 17 as well as on Figure 24. Figure 25 shows a photo of a cylinder during testing.

Table 16: UCS Testing Program

Mixture	Wt% Binder Addition	Binder Type	Material	Paste Slump Tested	No. of Cylinders Curing 7 days	No. of Cylinders Curing 28 days	Total No. of Cylinders
1	5	NPC	113-88649 Copperwood Total Tails	7"	3	3	6
2	5	NPC		10"	3	3	6
3	3	90:10		7"	3	3	6
4	5	90:10		7"	3	3	6

NPC - Normal Portland Cement
90:10 – 90% ground iron blast furnace slag: 10% NPC

After the 7-day curing period the cylinders had not sufficiently cured to be tested therefore they were returned to the curing room and were tested after 28 and 56-day curing periods.

Table 17: UCS Results

Mixture	Wt% Binder Addition	Binder Type	Material	Paste Slump Tested	Average Strength (PSI)			Average Bulk Density (lb/ft^3)
					Curing 7 days	Curing 28 days	Curing 56 days	
1	5	NPC	113-88649 Copperwood Total Tails	7"	N/A	17.1	34.3	116.8
2	5	NPC		10"	N/A	7.6	15.8	110.9
3	3	90:10		7"	N/A	N/A	N/A	N/A
4	5	90:10		7"	N/A	N/A	N/A	N/A

N/A: cylinders were not sufficiently cured to allow testing

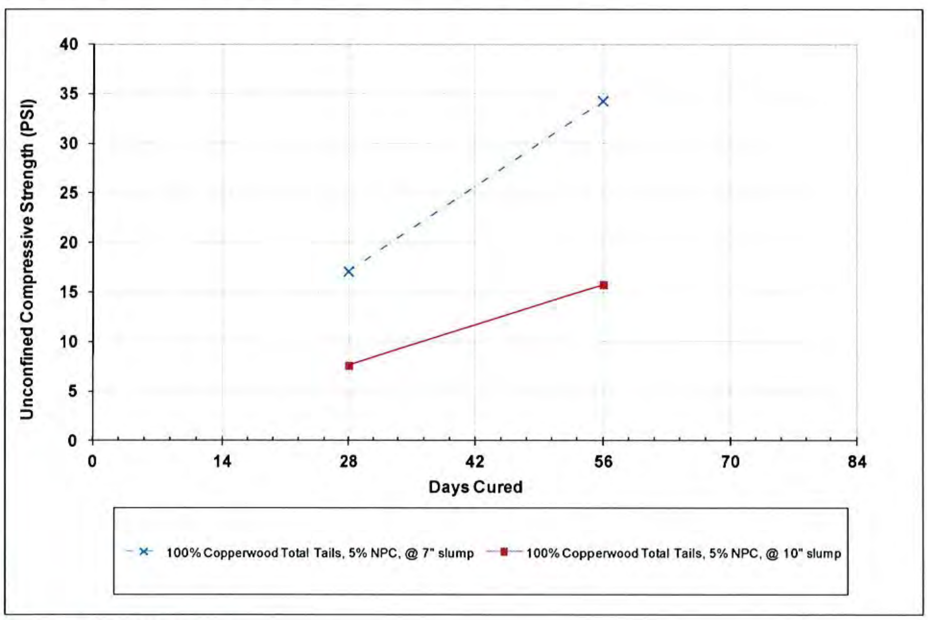

Figure 24: UCS Results

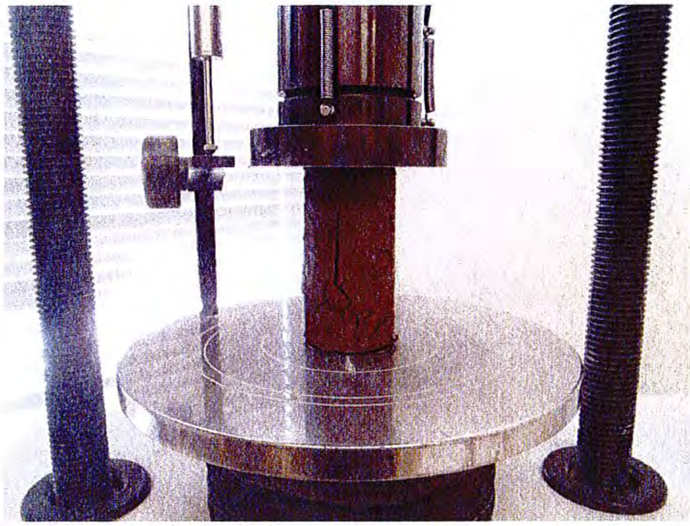

Figure 25: UCS Cylinder, 28-day Curing Period, During Testing

7.0 CLOSURE

If there are any questions regarding this report, please do not hesitate to contact the undersigned.

GOLDER ASSOCIATES LTD.

Mark Labelle
Laboratory Manager

Bruno Mandl, P.Eng.
Senior Mining Engineer

ML/BM/ds

APPENDIX A

ICP-MS Results

TESTMARK Laboratories Ltd.

Committed to Quality and Service

Analytical Report

Client:	Michelle Levesque	**Work Order Number:**	**145568**
Company:	Golder Associates Ltd. - Paste Engineering Lab	**Date Order Received:**	01/09/12
Address:	1010 Lorne St.	**Regulation:**	Information not provided
	Sudbury, ON, P3A 4S4	**PO #:**	
Phone:	(705) 524-5533	**Project #:**	113-88649
Fax:	(705) 524-9636		
Email:	mlevesque@golder.com		

Analyses were performed on the following samples submitted with your order.
The results relate only to the items tested.

Sample Name	Lab #	Matrix	Type	Comments	Date Collected	Time Collected
113-88649 Tailings	392949	Soil	Grab		01/06/12	14:00

The following instrumentation and reference methods were used for your sample(s)

Method Name	Description	Reference
ICPMS Soil	Determination of Metals in Soil by ICP/MS and BCSALM Method **Instrument group:** Perkin Elmer ICPMS	Based on SW846-6020

This report has been approved by:

Mark Charbonneau, Ph.D.
Metals Section Head

TESTMARK Laboratories Ltd.

Committed to Quality and Service

Golder Associates Ltd. - Paste Engineering Lab — Work Order: 145568

Sample Data:

Sample Name: 113-88649 Tailings **Date: 01/06/12** **Matrix: Soil** **Lab #: 392949**

ICPMS Soil				
Parameter	**MDL**	**Result**	**Units**	**QAQCID**
Aluminum	5	26900	µg/g	20120111.R13ba
Aluminum (Dup)	5	28100	µg/g	20120111.R13ba
Antimony	0.5	<0.5	µg/g	20120111.R13ba
Antimony (Dup)	0.5	<0.5	µg/g	20120111.R13ba
Arsenic	0.5	0.64	µg/g	20120111.R13ba
Arsenic (Dup)	0.5	0.66	µg/g	20120111.R13ba
Barium	0.5	121	µg/g	20120111.R13ba
Barium (Dup)	0.5	135	µg/g	20120111.R13ba
Beryllium	0.5	1.1	µg/g	20120111.R13ba
Beryllium (Dup)	0.5	1.2	µg/g	20120111.R13ba
Bismuth	0.5	<0.5	µg/g	20120111.R13ba
Bismuth (Dup)	0.5	<0.5	µg/g	20120111.R13ba
Boron (Not Hot Water Extractable)	1	18.2	µg/g	20120111.R13ba
Boron (Not Hot Water Extractable) (Dup)	1	20.2	µg/g	20120111.R13ba
Cadmium	0.05	1.02	µg/g	20120111.R13ba
Cadmium (Dup)	0.05	1.21	µg/g	20120111.R13ba
Calcium	25	8790	µg/g	20120111.R13ba
Calcium (Dup)	25	8970	µg/g	20120111.R13ba
Cerium	0.5	57.7	µg/g	20120111.R13ba
Cerium (Dup)	0.5	58.6	µg/g	20120111.R13ba
Cesium	0.5	2.4	µg/g	20120111.R13ba
Cesium (Dup)	0.5	2.6	µg/g	20120111.R13ba
Chromium	0.5	43.1	µg/g	20120111.R13ba
Chromium (Dup)	0.5	44.7	µg/g	20120111.R13ba
Cobalt	0.05	23.8	µg/g	20120111.R13ba
Cobalt (Dup)	0.05	24.3	µg/g	20120111.R13ba
Copper	5	6540	µg/g	20120111.R13ba
Copper (Dup)	5	6630	µg/g	20120111.R13ba
Europium	0.5	1.2	µg/g	20120111.R13ba
Europium (Dup)	0.5	1.2	µg/g	20120111.R13ba
Gallium	0.5	13.9	µg/g	20120111.R13ba
Gallium (Dup)	0.5	15	µg/g	20120111.R13ba
Iron	100	46400	µg/g	20120111.R13ba
Iron (Dup)	100	48700	µg/g	20120111.R13ba
Lanthanum	0.5	28.6	µg/g	20120111.R13ba
Lanthanum (Dup)	0.5	29	µg/g	20120111.R13ba
Lead	0.5	5.56	µg/g	20120111.R13ba
Lead (Dup)	0.5	5.66	µg/g	20120111.R13ba
Lithium	2.5	41	µg/g	20120111.R13ba
Lithium (Dup)	2.5	40.4	µg/g	20120111.R13ba
Magnesium	2	21900	µg/g	20120111.R13ba
Magnesium (Dup)	2	22700	µg/g	20120111.R13ba
Manganese	5	1450	µg/g	20120111.R13ba
Manganese (Dup)	5	1510	µg/g	20120111.R13ba

TESTMARK Laboratories Ltd.

Committed to Quality and Service

Golder Associates Ltd. - Paste Engineering Lab | Work Order: 145568

Sample Name: 113-88649 Tailings | **Date: 01/06/12** | **Matrix: Soil** | **Lab #: 392949**

ICPMS Soil				
Parameter	**MDL**	**Result**	**Units**	**QAQCID**
Mercury	0.05	<0.05	µg/g	20120111.R13ba
Mercury (Dup)	0.05	<0.05	µg/g	20120111.R13ba
Molybdenum	0.5	0.52	µg/g	20120111.R13ba
Molybdenum (Dup)	0.5	0.54	µg/g	20120111.R13ba
Nickel	0.5	43.8	µg/g	20120111.R13ba
Nickel (Dup)	0.5	44.9	µg/g	20120111.R13ba
Niobium	0.5	<0.5	µg/g	20120111.R13ba
Niobium (Dup)	0.5	<0.5	µg/g	20120111.R13ba
Phosphorus	25	772	µg/g	20120111.R13ba
Phosphorus (Dup)	25	783	µg/g	20120111.R13ba
Rubidium	0.5	18.5	µg/g	20120111.R13ba
Rubidium (Dup)	0.5	20.5	µg/g	20120111.R13ba
Scandium	0.5	9.67	µg/g	20120111.R13ba
Scandium (Dup)	0.5	10.1	µg/g	20120111.R13ba
Selenium	0.5	<0.5	µg/g	20120111.R13ba
Selenium (Dup)	0.5	<0.5	µg/g	20120111.R13ba
Silicon	300	330	µg/g	20120111.R13ba
Silicon (Dup)	300	<300	µg/g	20120111.R13ba
Silver	0.05	1.61	µg/g	20120111.R13ba
Silver (Dup)	0.05	1.67	µg/g	20120111.R13ba
Strontium	0.5	24.8	µg/g	20120111.R13ba
Strontium (Dup)	0.5	25.9	µg/g	20120111.R13ba
Sulfur	400	680	µg/g	20120111.R13ba
Sulfur (Dup)	400	670	µg/g	20120111.R13ba
Tellurium	0.5	<0.5	µg/g	20120111.R13ba
Tellurium (Dup)	0.5	<0.5	µg/g	20120111.R13ba
Thallium	0.5	<0.5	µg/g	20120111.R13ba
Thallium (Dup)	0.5	<0.5	µg/g	20120111.R13ba
Thorium	0.5	4.5	µg/g	20120111.R13ba
Thorium (Dup)	0.5	4.5	µg/g	20120111.R13ba
Tin	0.5	1	µg/g	20120111.R13ba
Tin (Dup)	0.5	1	µg/g	20120111.R13ba
Titanium	0.5	463	µg/g	20120111.R13ba
Titanium (Dup)	0.5	491	µg/g	20120111.R13ba
Tungsten	0.5	<0.5	µg/g	20120111.R13ba
Tungsten (Dup)	0.5	<0.5	µg/g	20120111.R13ba
Uranium	0.5	1.2	µg/g	20120111.R13ba
Uranium (Dup)	0.5	1.2	µg/g	20120111.R13ba
Vanadium	0.5	70.7	µg/g	20120111.R13ba
Vanadium (Dup)	0.5	74	µg/g	20120111.R13ba
Yttrium	0.5	16.3	µg/g	20120111.R13ba
Yttrium (Dup)	0.5	16.7	µg/g	20120111.R13ba
Zinc	5	134	µg/g	20120111.R13ba
Zinc (Dup)	5	147	µg/g	20120111.R13ba
Zirconium	0.5	21.8	µg/g	20120111.R13ba
Zirconium (Dup)	0.5	22.6	µg/g	20120111.R13ba

TESTMARK Laboratories Ltd.

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Golder Associates Ltd. - Paste Engineering Lab | Work Order: 145568

Sample Name: 113-88649 Tailings **Date: 01/06/12** **Matrix: Soil** **Lab #: 392949**

ICPMS Soil				
Parameter	**MDL**	**Result**	**Units**	**QAQCID**

MDL *Method detection limit or minimum reporting limit.*
% Rec *Surrogate compounds are added to the sample in some cases and the recovery is reported as a percent recovered.*
QAQCID *This is a unique reference to the quality control data set used to generate the reported value.*
Data reported for organic analysis in soil samples are corrected for moisture content
Matrix *If the matrix is a leachate, the sample was extracted according to regulation 558.*
INT *Interferences*
TNTC *Too numerous to count*
ND *Not detected*

TESTMARK Laboratories Ltd.

Committed to Quality and Service

Golder Associates Ltd. - Paste Engineering Lab — Work Order: 145568

Quality Control Data:

ICPMS Soil

Method Blank						
Parameter	**MDL**	**Units**	**UCL**	**Result**	**LCL**	**QAQCID**
Aluminum	2	µg/g	<2	<2	<2	20120111.R13ba
Antimony	0.5	µg/g	<0.5	<0.5	<0.5	20120111.R13ba
Arsenic	0.5	µg/g	<0.5	<0.5	<0.5	20120111.R13ba
Barium	0.5	µg/g	<0.5	<0.5	<0.5	20120111.R13ba
Beryllium	2.5	µg/g	<2.5	<2.5	<2.5	20120111.R13ba
Bismuth	0.5	µg/g	<0.5	<0.5	<0.5	20120111.R13ba
Cadmium	0.5	µg/g	<0.5	<0.5	<0.5	20120111.R13ba
Calcium	2.5	µg/g	<2.5	<2.5	<2.5	20120111.R13ba
Cerium	0.5	µg/g	<0.5	<0.5	<0.5	20120111.R13ba
Cesium	0.5	µg/g	<0.5	<0.5	<0.5	20120111.R13ba
Chromium	0.5	µg/g	<0.5	<0.5	<0.5	20120111.R13ba
Cobalt	0.5	µg/g	<0.5	<0.5	<0.5	20120111.R13ba
Copper	0.5	µg/g	<0.5	<0.5	<0.5	20120111.R13ba
Europium	0.5	µg/g	<0.5	<0.5	<0.5	20120111.R13ba
Gallium	0.5	µg/g	<0.5	<0.5	<0.5	20120111.R13ba
Iron	0.5	µg/g	<0.5	<0.5	<0.5	20120111.R13ba
Lanthanum	0.5	µg/g	<0.5	<0.5	<0.5	20120111.R13ba
Lead	0.05	µg/g	0.1	<0.05	<0.05	20120111.R13ba
Magnesium	1	µg/g	<1	<1	<1	20120111.R13ba
Manganese	0.5	µg/g	<0.5	<0.5	<0.5	20120111.R13ba
Mercury	0.5	µg/g	<0.5	<0.5	<0.5	20120111.R13ba
Molybdenum	0.5	µg/g	<0.5	<0.5	<0.5	20120111.R13ba
Nickel	0.5	µg/g	<0.5	<0.5	<0.5	20120111.R13ba
Niobium	0.5	µg/g	<0.5	<0.5	<0.5	20120111.R13ba
Phosphorus	25	µg/g	<25	<25	<25	20120111.R13ba
Rubidium	0.5	µg/g	<0.5	<0.5	<0.5	20120111.R13ba
Scandium	0.5	µg/g	<0.5	<0.5	<0.5	20120111.R13ba
Selenium	0.5	µg/g	<0.5	<0.5	<0.5	20120111.R13ba
Silver	0.5	µg/g	<0.5	<0.5	<0.5	20120111.R13ba
Strontium	0.5	µg/g	<0.5	<0.5	<0.5	20120111.R13ba
Thallium	0.5	µg/g	<0.5	<0.5	<0.5	20120111.R13ba
Thorium	0.5	µg/g	<0.5	<0.5	<0.5	20120111.R13ba
Tin	2.5	µg/g	<2.5	<2.5	<2.5	20120111.R13ba
Titanium	0.5	µg/g	<0.5	<0.5	<0.5	20120111.R13ba
Tungsten	0.5	µg/g	<0.5	<0.5	<0.5	20120111.R13ba
Uranium	0.5	µg/g	<0.5	<0.5	<0.5	20120111.R13ba
Vanadium	0.5	µg/g	<0.5	<0.5	<0.5	20120111.R13ba
Yttrium	0.5	µg/g	<0.5	<0.5	<0.5	20120111.R13ba
Zinc	0.5	µg/g	<0.5	<0.5	<0.5	20120111.R13ba
Zirconium	0.5	µg/g	<0.5	<0.5	<0.5	20120111.R13ba

SS2 CRM						
Parameter	**MDL**	**Units**	**UCL**	**Result**	**LCL**	**QAQCID**
Aluminum	5	µg/g	19787	13600	6743	20120111.R13ba
Arsenic	0.5	µg/g	125	63.2	25	20120111.R13ba

TESTMARK Laboratories Ltd.

Committed to Quality and Service

Golder Associates Ltd. - Paste Engineering Lab

Work Order: 145568

ICPMS Soil

SS2 CRM						
Parameter	**MDL**	**Units**	**UCL**	**Result**	**LCL**	**QAQCID**
Barium	0.5	µg/g	281	211	149	20120111.R13ba
Calcium	25	µg/g	138279	101000	87443	20120111.R13ba
Chromium	0.5	µg/g	54	33.1	14	20120111.R13ba
Cobalt	0.05	µg/g	15	10.7	9	20120111.R13ba
Copper	5	µg/g	243	194	139	20120111.R13ba
Iron	100	µg/g	29261	19700	12831	20120111.R13ba
Lead	0.5	µg/g	184	115	68	20120111.R13ba
Lithium	2.5	µg/g	23	15	5	20120111.R13ba
Magnesium	2	µg/g	14502	11200	7628	20120111.R13ba
Manganese	0.5	µg/g	590	492	324	20120111.R13ba
Nickel	0.5	µg/g	75	48.8	33	20120111.R13ba
Strontium	0.5	µg/g	272	219	156	20120111.R13ba
Titanium	5	µg/g	1402	1040	298	20120111.R13ba
Vanadium	0.5	µg/g	51	39.8	17	20120111.R13ba
Zinc	5	µg/g	597	469	337	20120111.R13ba

UCL Upper Control Limit
LCL Lower Control Limit

APPENDIX B

Rheograms

Golder Associates Ltd.
Viscosity / Flow Curve Testing R/S Plus Rheometer

Client:	GAI-Orvana - Copperwood Review - Michigan
Project Number:	113-88649
Date:	20/01/2012
Technologist	CJC

		Status	Reviewer	Date Complete
Data Entry		Complete	CJC	25/01/2012
Data Review	1st Review	Complete	CJC	25/01/2012
	2nd Review	Complete	MYL	25/01/2012

Sample ID: 113-88649 Copperwood Total Tails
Sample Description: smooth thin brown tails
Bob: CC25 Profiled Bob
Additional Info:
Specific Gravity 2.8

VISCOSITY DATA

Ramp Up

REF	Trial 1	Trial 2	Trial 3	AVG
1	0.7256	0.7106	0.7206	0.72
2	0.4822	0.4963	0.5025	0.49
3	0.3445	0.3425	0.3514	0.35
4	0.2425	0.2379	0.2316	0.24
5	0.1424	0.1429	0.1490	0.14
6	0.0662	0.0653	0.0620	0.06
7	0.0315	0.0298	0.0304	0.03

Ramp Down

Trial 1	Trial 2	Trial 3	AVG
0.5552	0.5318	0.5835	0.56
0.3947	0.3718	0.4032	0.39
0.2808	0.2564	0.2791	0.27
0.2113	0.2088	0.2071	0.21
0.1286	0.1260	0.1337	0.13
0.0581	0.0603	0.0609	0.06
0.0282	0.0282	0.0286	0.03

YIELD STRESS DATA

Ramp Up

REF	Trial 1	Trial 2	Trial 3	AVG
1	437.9846	439.1099	457.9729	445
2	333.8414	317.3430	332.3870	328
3	227.1409	225.5227	227.2422	227
4	167.9957	169.9119	168.0084	169
5	100.9106	102.3156	99.9344	101
6	44.0873	46.4649	46.0069	46
7	20.2149	20.0698	19.8334	20

Ramp Down

Trial 1	Trial 2	Trial 3	AVG
513.6858	513.3108	517.6567	515
367.9966	365.9128	372.0917	369
252.9426	260.8012	258.3319	257
180.9732	182.1837	178.3528	181
107.2473	109.2087	106.4991	108
47.5018	48.7199	46.8182	48
21.6621	21.0970	20.7549	21

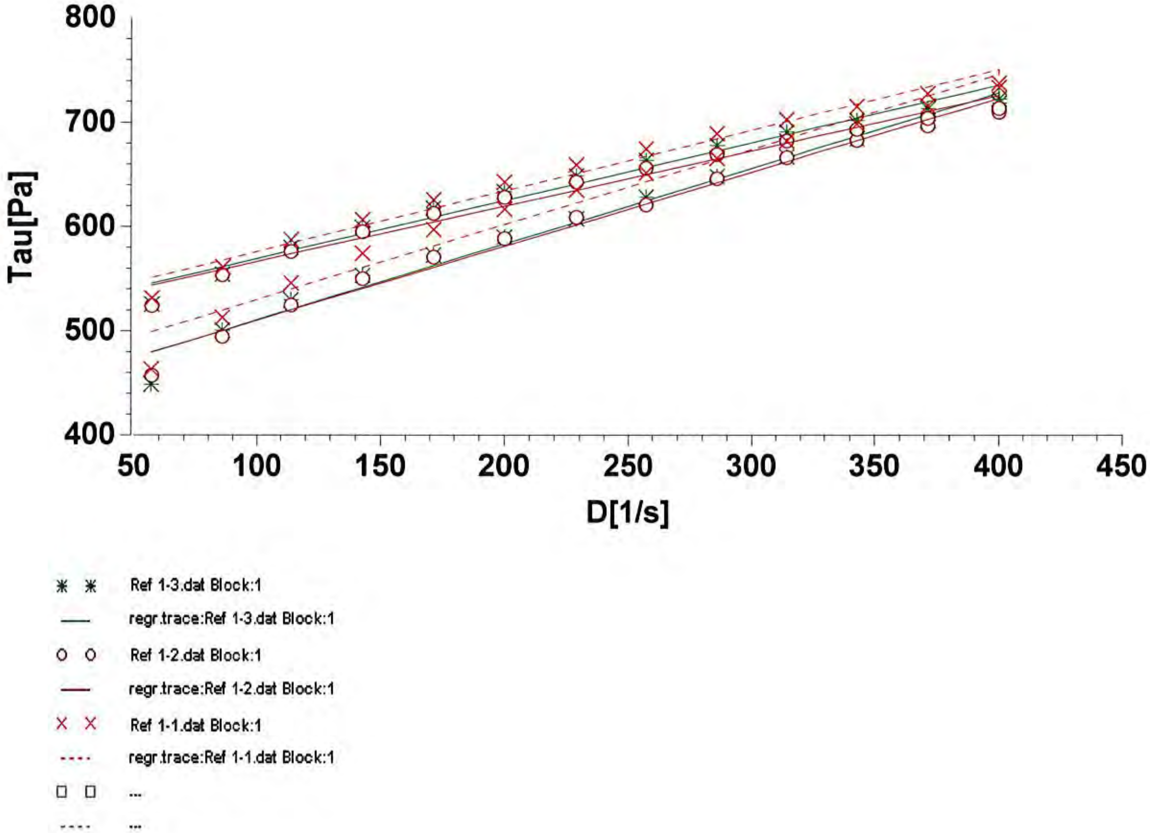
WEIGHT PERCENT SOLIDS

REF	Pan #	Pan Wt. (g)	Wet (g)	Dry (g)	Wt% Solids	SVF (Φ)
1	N12	2.6600	27.4300	20.9800	74%	0.51
2	MY04	2.6500	31.6700	23.7600	73%	0.49
3	mm92	2.6800	27.4000	20.4200	72%	0.48
4	mm04	2.6600	24.8100	18.3100	71%	0.46
5	x20	6.0600	23.8700	18.3300	69%	0.44
6	mm94	2.7500	23.1700	16.0900	65%	0.40
7	mm01	2.7700	18.0800	12.0600	61%	0.36

Additional Notes:

Appendix B

10:56 23/01/12
Manual Report Analysis/Regression



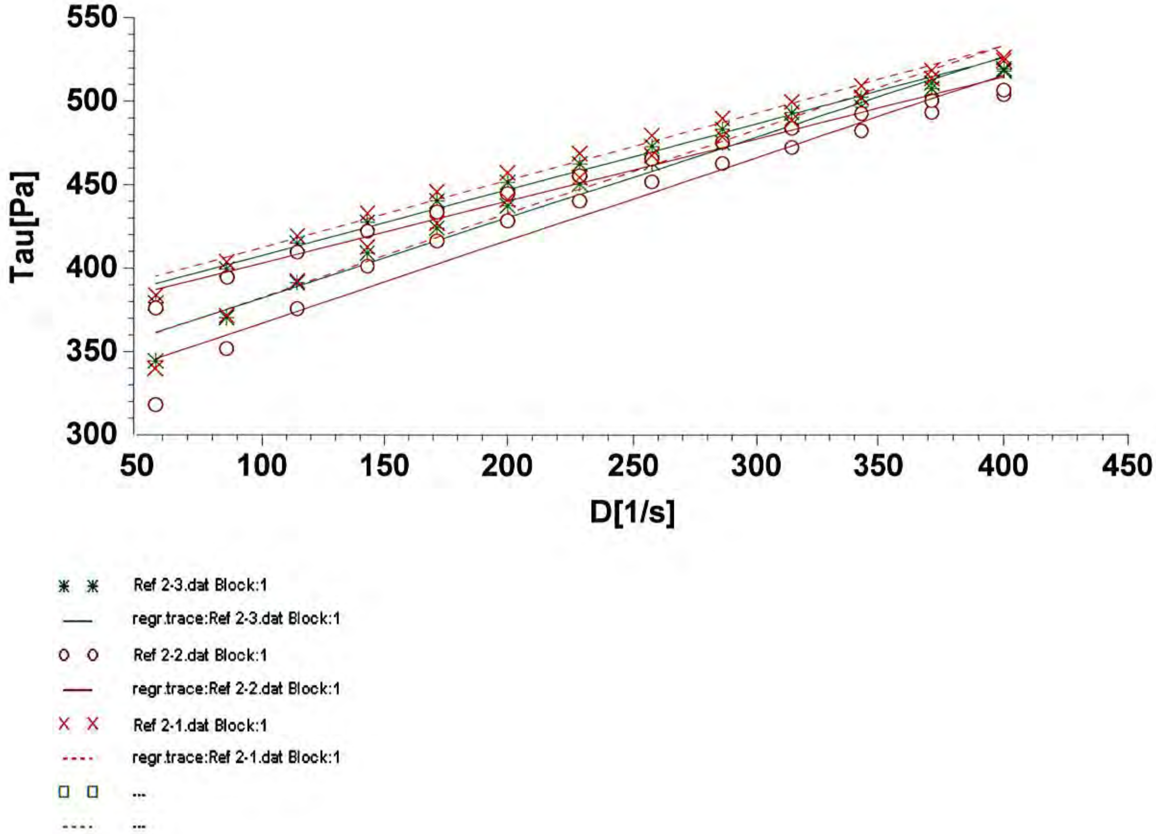
Analysis-results
Analysis data source: Ref 1-3.dat Block:1
filter activated: D[1/s]>40
step1: Bingham: Y=437.98+0.72563*X ;B=0.98167; S=11.5
step1: Bingham yieldstress[Pa]=437.9846
step1: Bingham viscosity[Pas]=0.7256
step2: Bingham: Y=513.69+0.55516*X ;B=0.97825; S=9.62
step2: Bingham yieldstress[Pa]=513.6858
step2: Bingham viscosity[Pas]=0.5552

filter activated: D[1/s]>40
step1: Bingham: Y=439.11+0.71063*X ;B=0.98592; S=9.87
step1: Bingham yieldstress[Pa]=439.1099
step1: Bingham viscosity[Pas]=0.7106
step2: Bingham: Y=513.31+0.53177*X ;B=0.97814; S=9.24
step2: Bingham yieldstress[Pa]=513.3108
step2: Bingham viscosity[Pas]=0.5318

filter activated: D[1/s]>40
step1: Bingham: Y=457.97+0.72062*X ;B=0.96844; S=15.1
step1: Bingham yieldstress[Pa]=457.9729
step1: Bingham viscosity[Pas]=0.7206
step2: Bingham: Y=517.66+0.58346*X ;B=0.98099; S=9.44
step2: Bingham yieldstress[Pa]=517.6567
step2: Bingham viscosity[Pas]=0.5835
End of report

Appendix B

11:22 23/01/12
Manual Report Analysis/Regression



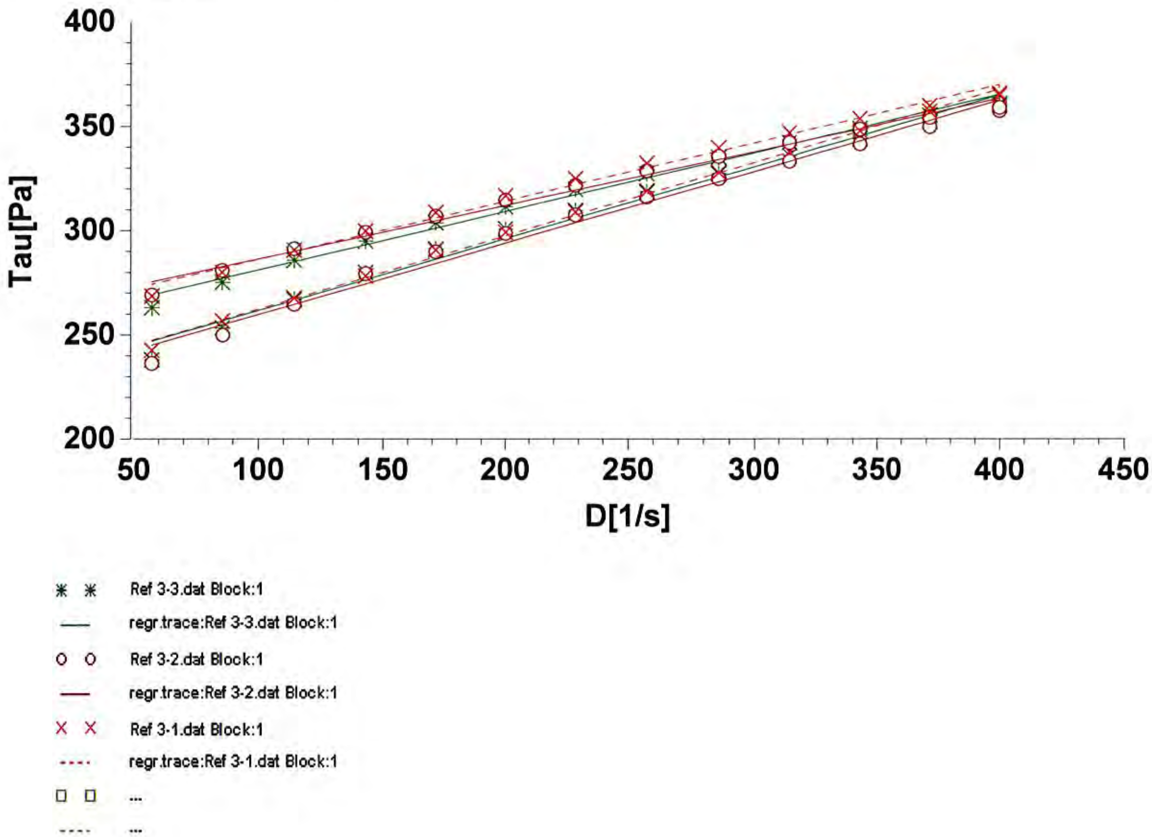
Analysis-results
Analysis data source: Ref 2-3.dat Block:1
filter activated: D[1/s]>40
step1: Bingham: Y=333.84+0.4822*X ;B=0.98135; S=7.73
step1: Bingham yieldstress[Pa]=333.8414
step1: Bingham viscosity[Pas]=0.4822
step2: Bingham: Y=368+0.39467*X ;B=0.98783; S=5.09
step2: Bingham yieldstress[Pa]=367.9966
step2: Bingham viscosity[Pas]=0.3947

filter activated: D[1/s]>40
step1: Bingham: Y=317.34+0.49629*X ;B=0.95559; S=12.4
step1: Bingham yieldstress[Pa]=317.343
step1: Bingham viscosity[Pas]=0.4963
step2: Bingham: Y=365.91+0.37176*X ;B=0.98512; S=5.31
step2: Bingham yieldstress[Pa]=365.9128
step2: Bingham viscosity[Pas]=0.3718

filter activated: D[1/s]>40
step1: Bingham: Y=332.39+0.5025*X ;B=0.97709; S=8.94
step1: Bingham yieldstress[Pa]=332.387
step1: Bingham viscosity[Pas]=0.5025
step2: Bingham: Y=372.09+0.40321*X ;B=0.98807; S=5.15
step2: Bingham yieldstress[Pa]=372.0917
step2: Bingham viscosity[Pas]=0.4032
End of report

Appendix B

11:47 23/01/12
Manual Report Analysis/Regression



Analysis-results
Analysis data source: Ref 3-3.dat Block:1
filter activated: D[1/s]>40
step1: Bingham: Y=227.14+0.34454*X ;B=0.98826; S=4.36
step1: Bingham yieldstress[Pa]=227.1409
step1: Bingham viscosity[Pas]=0.3445
step2: Bingham: Y=252.94+0.28079*X ;B=0.99161; S=3
step2: Bingham yieldstress[Pa]=252.9426
step2: Bingham viscosity[Pas]=0.2808

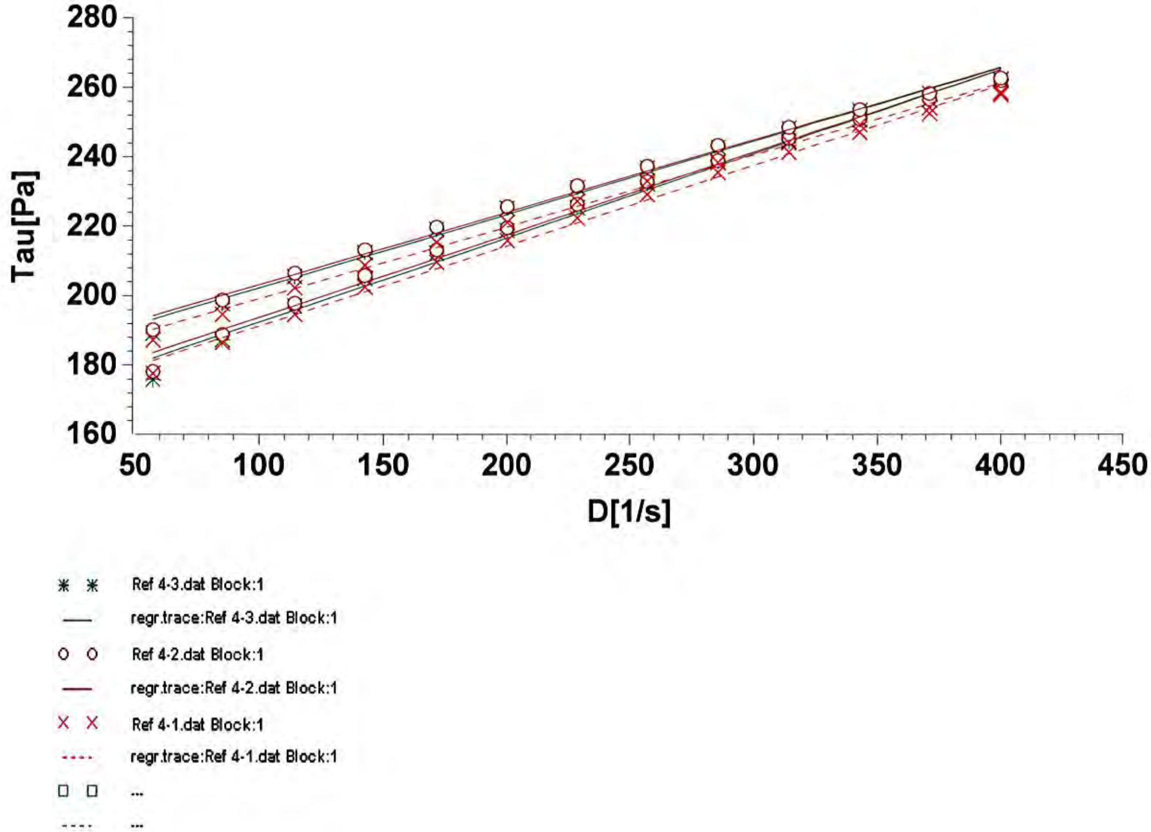
filter activated: D[1/s]>40
step1: Bingham: Y=225.52+0.34253*X ;B=0.98655; S=4.65
step1: Bingham yieldstress[Pa]=225.5227
step1: Bingham viscosity[Pas]=0.3425
step2: Bingham: Y=260.8+0.2564*X ;B=0.99038; S=2.94
step2: Bingham yieldstress[Pa]=260.8012
step2: Bingham viscosity[Pas]=0.2564

filter activated: D[1/s]>40
step1: Bingham: Y=227.24+0.35138*X ;B=0.9972; S=2.17
step1: Bingham yieldstress[Pa]=227.2422
step1: Bingham viscosity[Pas]=0.3514
step2: Bingham: Y=258.33+0.27907*X ;B=0.99228; S=2.86
step2: Bingham yieldstress[Pa]=258.3319
step2: Bingham viscosity[Pas]=0.2791
End of report

Appendix B

multiple data sources

13:19 23/01/12
Manual Report Analysis/Regression



Analysis-results
Analysis data source: Ref 4-3.dat Block:1
filter activated: D[1/s]>40
step1: Bingham: Y=168+0.24251*X ;B=0.99103; S=2.68
step1: Bingham yieldstress[Pa]=167.9957
step1: Bingham viscosity[Pas]=0.2425
step2: Bingham: Y=180.97+0.21131*X ;B=0.99332; S=2.01
step2: Bingham yieldstress[Pa]=180.9732
step2: Bingham viscosity[Pas]=0.2113

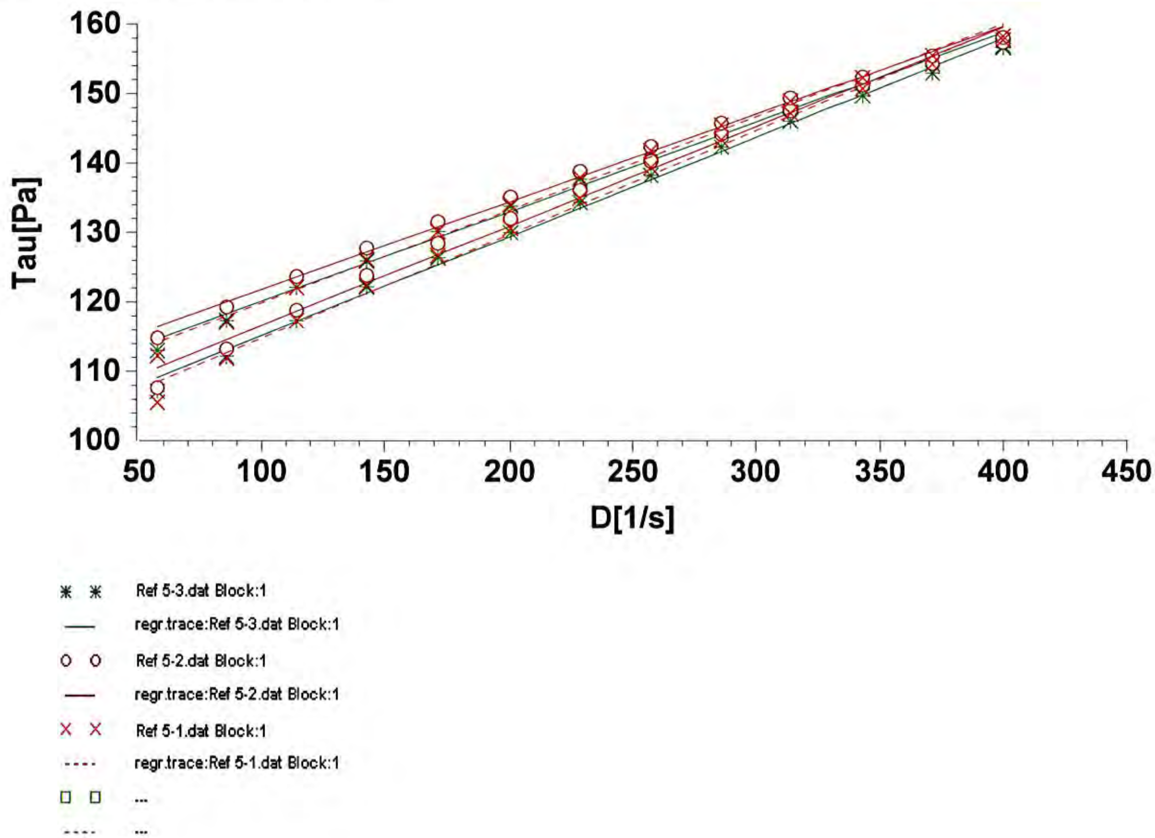
filter activated: D[1/s]>40
step1: Bingham: Y=169.91+0.23789*X ;B=0.99204; S=2.48
step1: Bingham yieldstress[Pa]=169.9119
step1: Bingham viscosity[Pas]=0.2379
step2: Bingham: Y=182.18+0.20876*X ;B=0.99335; S=1.99
step2: Bingham yieldstress[Pa]=182.1837
step2: Bingham viscosity[Pas]=0.2088

filter activated: D[1/s]>40
step1: Bingham: Y=168.01+0.23158*X ;B=0.99485; S=1.94
step1: Bingham yieldstress[Pa]=168.0084
step1: Bingham viscosity[Pas]=0.2316
step2: Bingham: Y=178.35+0.20706*X ;B=0.99534; S=1.65
step2: Bingham yieldstress[Pa]=178.3528
step2: Bingham viscosity[Pas]=0.2071
End of report

Appendix B

multiple data sources

13:45 23/01/12
Manual Report Analysis/Regression



Analysis-results
Analysis data source: Ref 5-3.dat Block:1
filter activated: D[1/s]>40
step1: Bingham: Y=100.91+0.14236*X ;B=0.99634; S=1
step1: Bingham yieldstress[Pa]=100.9106
step1: Bingham viscosity[Pas]=0.1424
step2: Bingham: Y=107.25+0.12864*X ;B=0.99513; S=1.05
step2: Bingham yieldstress[Pa]=107.2473
step2: Bingham viscosity[Pas]=0.1286

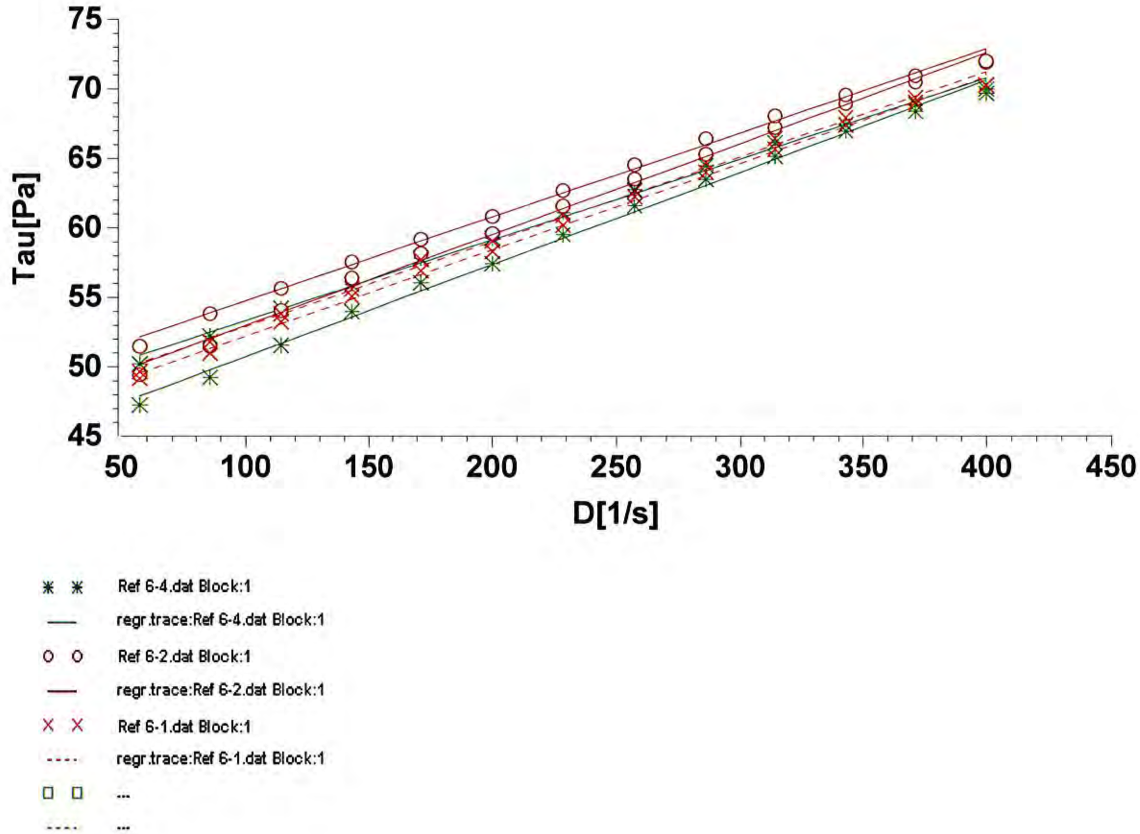
filter activated: D[1/s]>40
step1: Bingham: Y=102.32+0.14294*X ;B=0.99208; S=1.48
step1: Bingham yieldstress[Pa]=102.3156
step1: Bingham viscosity[Pas]=0.1429
step2: Bingham: Y=109.21+0.12599*X ;B=0.99616; S=0.909
step2: Bingham yieldstress[Pa]=109.2087
step2: Bingham viscosity[Pas]=0.126

filter activated: D[1/s]>40
step1: Bingham: Y=99.934+0.14896*X ;B=0.99389; S=1.36
step1: Bingham yieldstress[Pa]=99.9344
step1: Bingham viscosity[Pas]=0.149
step2: Bingham: Y=106.5+0.13371*X ;B=0.99562; S=1.03
step2: Bingham yieldstress[Pa]=106.4991
step2: Bingham viscosity[Pas]=0.1337
End of report

Appendix B

multiple data sources

14:18 23/01/12
Manual Report Analysis/Regression



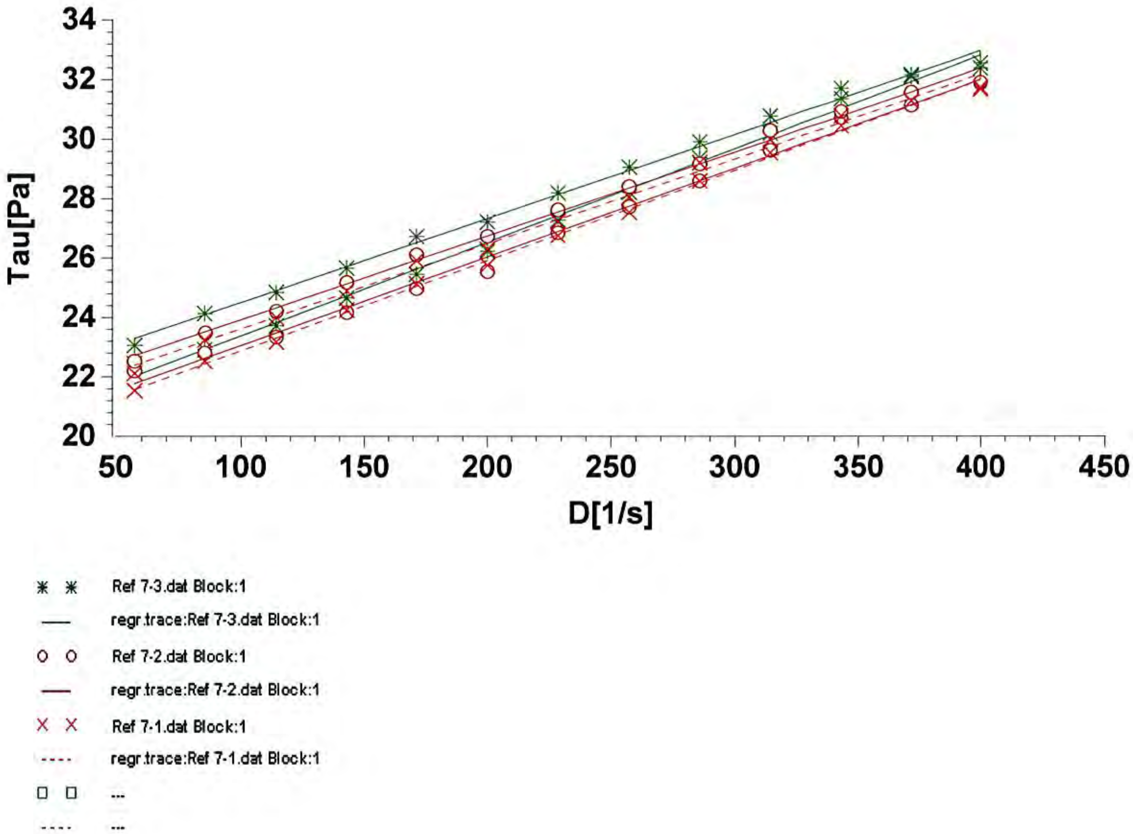
Analysis-results
Analysis data source: Ref 6-4.dat Block:1
filter activated: D[1/s]>40
step1: Bingham: Y=44.087+0.066249*X ;B=0.99562; S=0.511
step1: Bingham yieldstress[Pa]=44.0873
step1: Bingham viscosity[Pas]=0.0662
step2: Bingham: Y=47.502+0.058094*X ;B=0.99658; S=0.396
step2: Bingham yieldstress[Pa]=47.5018
step2: Bingham viscosity[Pas]=0.0581

filter activated: D[1/s]>40
step1: Bingham: Y=46.465+0.065264*X ;B=0.99645; S=0.453
step1: Bingham yieldstress[Pa]=46.4649
step1: Bingham viscosity[Pas]=0.0653
step2: Bingham: Y=48.72+0.060322*X ;B=0.9966; S=0.41
step2: Bingham yieldstress[Pa]=48.7199
step2: Bingham viscosity[Pas]=0.0603

filter activated: D[1/s]>40
step1: Bingham: Y=46.007+0.062047*X ;B=0.99838; S=0.291
step1: Bingham yieldstress[Pa]=46.0069
step1: Bingham viscosity[Pas]=0.062
step2: Bingham: Y=46.818+0.06095*X ;B=0.99611; S=0.442
step2: Bingham yieldstress[Pa]=46.8182
step2: Bingham viscosity[Pas]=0.0609
End of report

multiple data sources

14:41 23/01/12
Manual Report Analysis/Regression



Analysis-results
Analysis data source: Ref 7-3.dat Block:1
filter activated: D[1/s]>40
step1: Bingham: Y=20.215+0.031475*X ;B=0.99578; S=0.238
step1: Bingham yieldstress[Pa]=20.2149
step1: Bingham viscosity[Pas]=0.0315
step2: Bingham: Y=21.662+0.028227*X ;B=0.99442; S=0.246
step2: Bingham yieldstress[Pa]=21.6621
step2: Bingham viscosity[Pas]=0.0282

filter activated: D[1/s]>40
step1: Bingham: Y=20.07+0.0298*X ;B=0.99429; S=0.263
step1: Bingham yieldstress[Pa]=20.0698
step1: Bingham viscosity[Pas]=0.0298
step2: Bingham: Y=21.097+0.028183*X ;B=0.99628; S=0.2
step2: Bingham yieldstress[Pa]=21.097
step2: Bingham viscosity[Pas]=0.0282

filter activated: D[1/s]>40
step1: Bingham: Y=19.833+0.030371*X ;B=0.998; S=0.158
step1: Bingham yieldstress[Pa]=19.8334
step1: Bingham viscosity[Pas]=0.0304
step2: Bingham: Y=20.755+0.028553*X ;B=0.99513; S=0.232
step2: Bingham yieldstress[Pa]=20.7549
step2: Bingham viscosity[Pas]=0.0286
End of report

At Golder Associates we strive to be the most respected global company providing consulting, design, and construction services in earth, environment, and related areas of energy. Employee owned since our formation in 1960, our focus, unique culture and operating environment offer opportunities and the freedom to excel, which attracts the leading specialists in our fields. Golder professionals take the time to build an understanding of client needs and of the specific environments in which they operate. We continue to expand our technical capabilities and have experienced steady growth with employees who operate from offices located throughout Africa, Asia, Australasia, Europe, North America, and South America.

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Golder Associates

DRAFT **TECHNICAL MEMORANDUM**

Date:	August 17, 2012	**Project No.:**	123-88885
To:	Don Poulter	**Company:**	Golder Associates, Inc.
From:	Gordan Gjerapic		
cc:	David List, Matt Barrett	**Email:**	ggjerapic@golder.com

RE: ORVANA/COPPERWOOD PROJECT– CONSOLIDATION PROPERTIES FOR COMPOSITE TAILINGS SAMPLE

1.0 INTRODUCTION

Golder Associates Inc., (Golder) has prepared this technical memorandum to summarize tailings consolidation properties for the composite tailings sample received from KD Engineering earlier this year.

2.0 LABORATORY TEST RESULTS

Tailings consolidation properties were determined from the settling column and slurry-consolidation tests conducted in Golder's Soil Laboratory in Lakewood, Colorado. Initial conditions (after partially decanting water on top of the sample for the slurry consolidation test) and geotechnical classification results for the composite tailings samples are summarized in Table 1.

Table 1: Initial Conditions and Summary of Geotechnical Classification for Composite Tailings

Sample	Initial Solid Content Settling Column	Initial Solid Content Slurry-Consol	Specific Gravity	%Fines	Plasticity Index, PI[1]
Composite Sample	40.2%	48.2%	2.86	87.9%	2

1) Based on the liquid limit and plastic limit values of LL=21 and PL=19. Void Ratio Correction

During the slurry-consolidation test, the vertical load on the sample is applied via the top platen placed on the surface of the sample. As the magnitude of friction between the top platen and the sides of the containment cylinder is difficult to evaluate, the void ratio and compressibility at lower effective stresses are often better determined from the settling column test results. For the known amount of solids in the settling column, the height of solids in the settling column cylinder can be determined as

$$H_s = \frac{m_s}{A\,\rho_w\,G_s},$$

where

m_s = mass of solids in settling column cylinder, i.e., dry mass of the sample (g);

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A = cross-sectional area of the settling column cylinder (cm^2);

ρ_w = density of water (g/cm^3);

G_s = specific gravity.

The effective stress at the base of the settling column cylinder for a single-drain test can now be calculated as

$$\sigma_B = H_s \rho_w \, g \, (G_s - 1) \, ,$$

where g denotes the gravity acceleration (9.81 m/s^2). An average void ratio at the end of the settling column test can be calculated as

$$e_{avg} = \frac{H}{H_s} - 1 \, ,$$

where H denotes the sample height at the end of the settling column test. Summary of the settling column results are shown in Table 2.

Table 2: Average Void Ratio and Maximum Effective Stress – Settling Column Tests

Test Type	Effective Stress at the Base[1] (kPa)	Average Void Ratio (-)
Single Drained	1.06	1.89
Double Drained	TBD	TBD
Slurry Consolidation[2]	0.39	2.08

1) Assume hydrostatic pressure, i.e. neglect seepage forces.
2) Average void ratio calculated prior to loading.

The effective stress corresponding to the average void ratio was estimated as one-half of the base effective stress.

2.1 Constitutive Relationships

Laboratory measurement values in Table 1 and Table 2, as well as derived compressibility and permeability parameters were used to developed material parameters *A, B, C, D* and *Z* defining the following consolidation relationships (see e.g. Abu-Hejleh and Znidarcic, 1994, 1996):

$$e = A(\sigma' + Z)^B$$

and

$$k = C \, e^D$$

In the above relationships, *e* denotes the void ratio, σ' stands for the effective stress and *k* is hydraulic conductivity functionally dependent on void ratio. Consolidation parameters selected for modeling input for different systems of units are shown in tables 3 and 4.

Table 3: Selected Compressibility Parameters

Units	A	B	Z
(kPa)	1.594	-0.1497	0.085
(psf)	2.512	-0.1497	1.779
(psi)	1.194	-0.1497	0.012

Table 4: Selected Permeability Parameters

Units	C	D
(cm/s)	1.702×10^{-6}	3.653
(ft/day)	4.825×10^{-3}	3.653

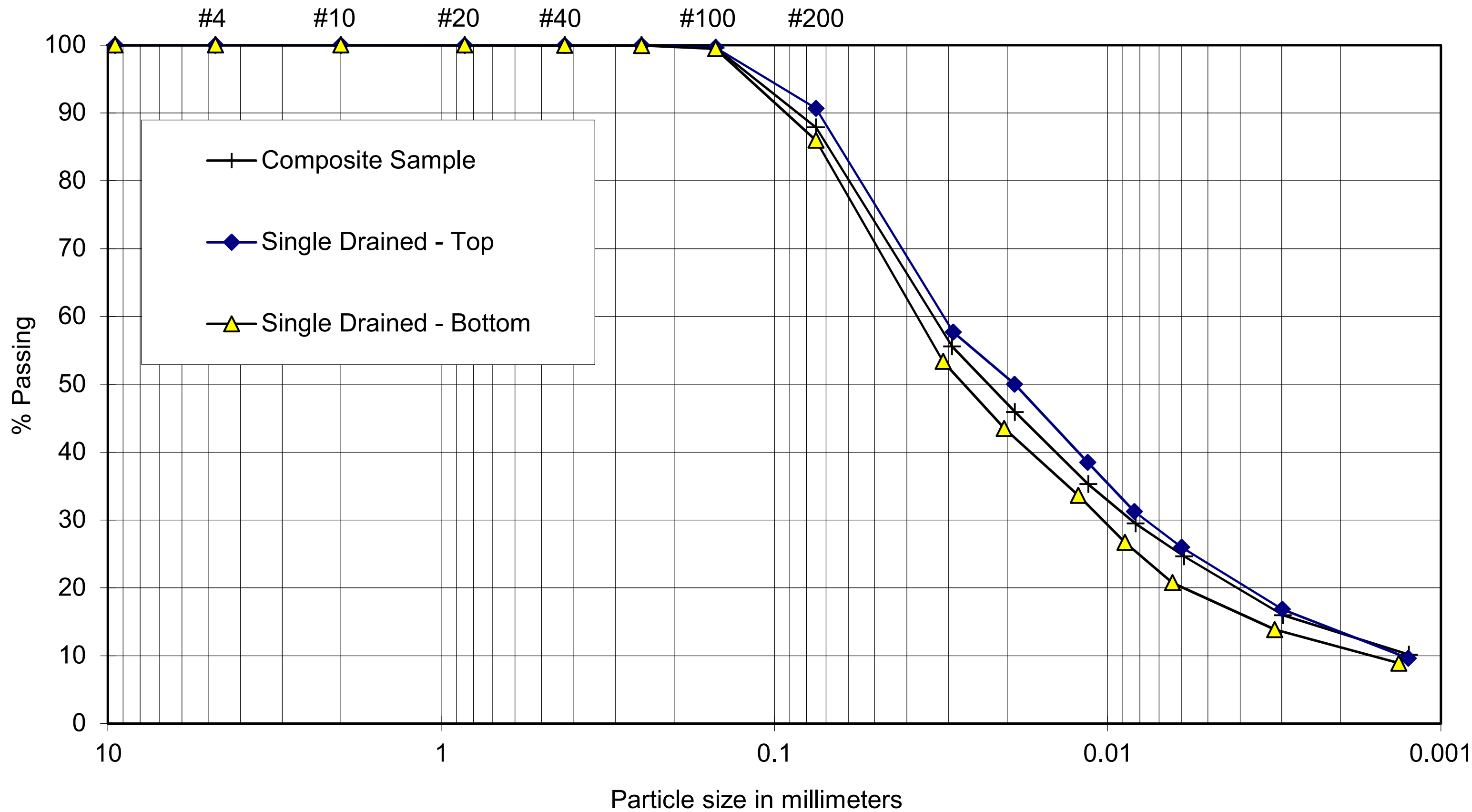
Grain size distribution curves and consolidation relationships are shown in figures 1 and 2. A complete set of laboratory results are shown in Attachment 1.

3.0 REFERENCES

Abu-Hejleh, A.N. and Znidarcic, D., 1994, “Estimation of the Consolidation Constitutive Relations”, Computer Methods and Advances in Geomechanics, Siriwardane & Zaman (eds) Balkema, Rotterdam, pp. 499-504.

Abu-Hejleh, A. N. and Znidarcic, D., 1996, “Consolidation Characteristics of Phosphatic Clays”, Journal of Geotechnical Engineering, ASCE, New-York, Vol. 122, No. 4. pp. 295-301.

FIGURES



Golder Associates

Denver, Colorado

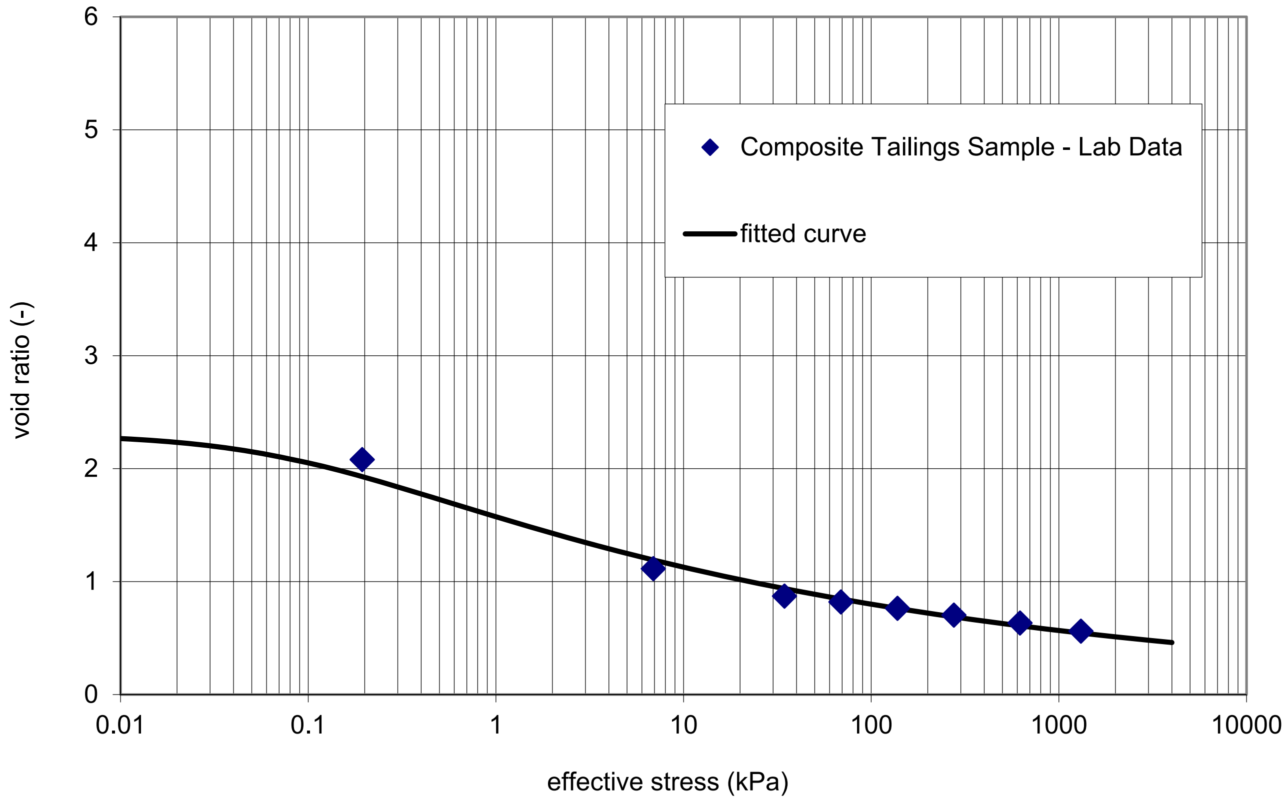
TITLE

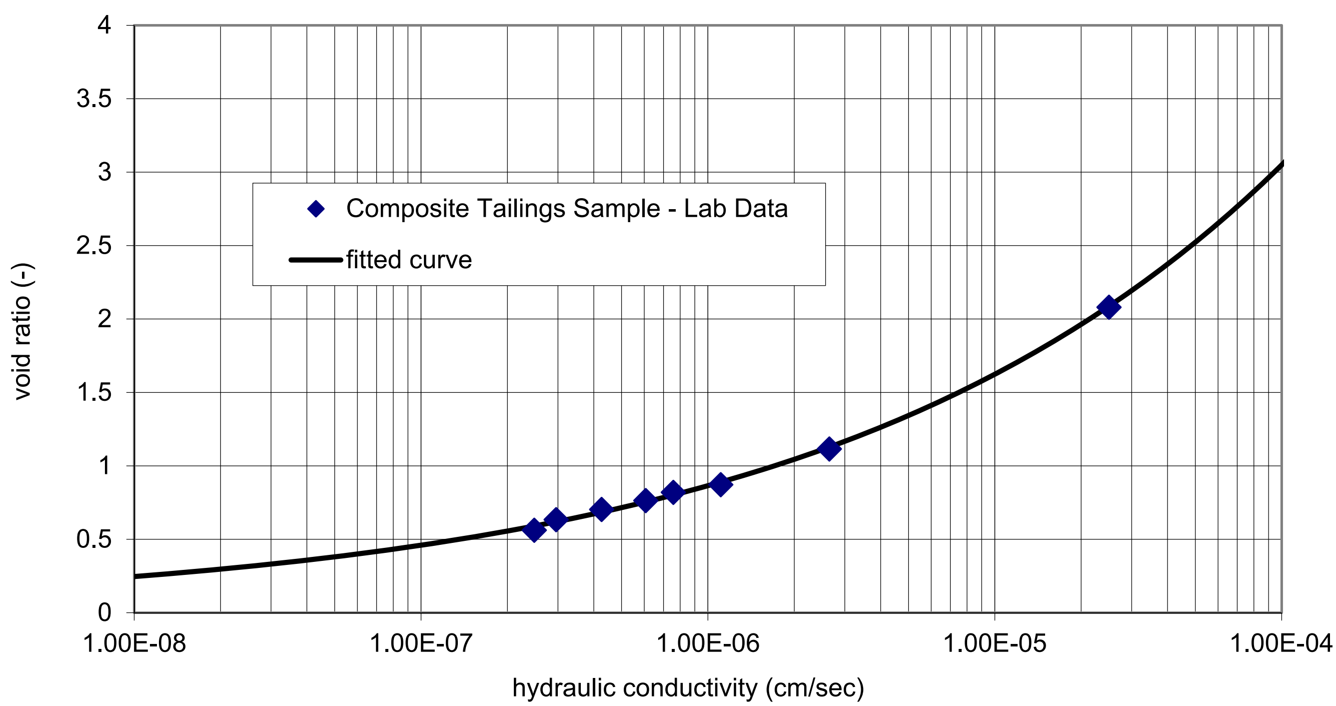
Grain Size Distribution Curves - Composite Tailings Sample and Top and Bottom Settling Column Samples

CLIENT/PROJECT

ORVANA / COPPERWOOD PROJECT
TAILINGS CHARACTERIZATION

DRAWN	GG	DATE	Aug-12	JOB NO.	123-88885
CHECKED	GG	SCALE	AS SHOWN	DWG. NO.	
REVIEWED	DP	FILE NO.		FIGURE NO.	FIGURE 1





Golder Associates

TITLE

CONSOLIDATION PROPERTIES
COMPOSITE TAILINGS SAMPLE

CLIENT/PROJECT

ORVANA / COPPERWOOD PROJECT
TAILINGS CHARACTERIZATION

DRAWN	**GG**	DATE	**8/17/2012**	JOB NO.	**123-88885**
CHECKED	**GG**	SCALE	**AS SHOWN**	DWG. NO. / REV. NO.	
REVIEWED	**DP**	FILE NO	MatProperties.xls		**FIGURE 2**

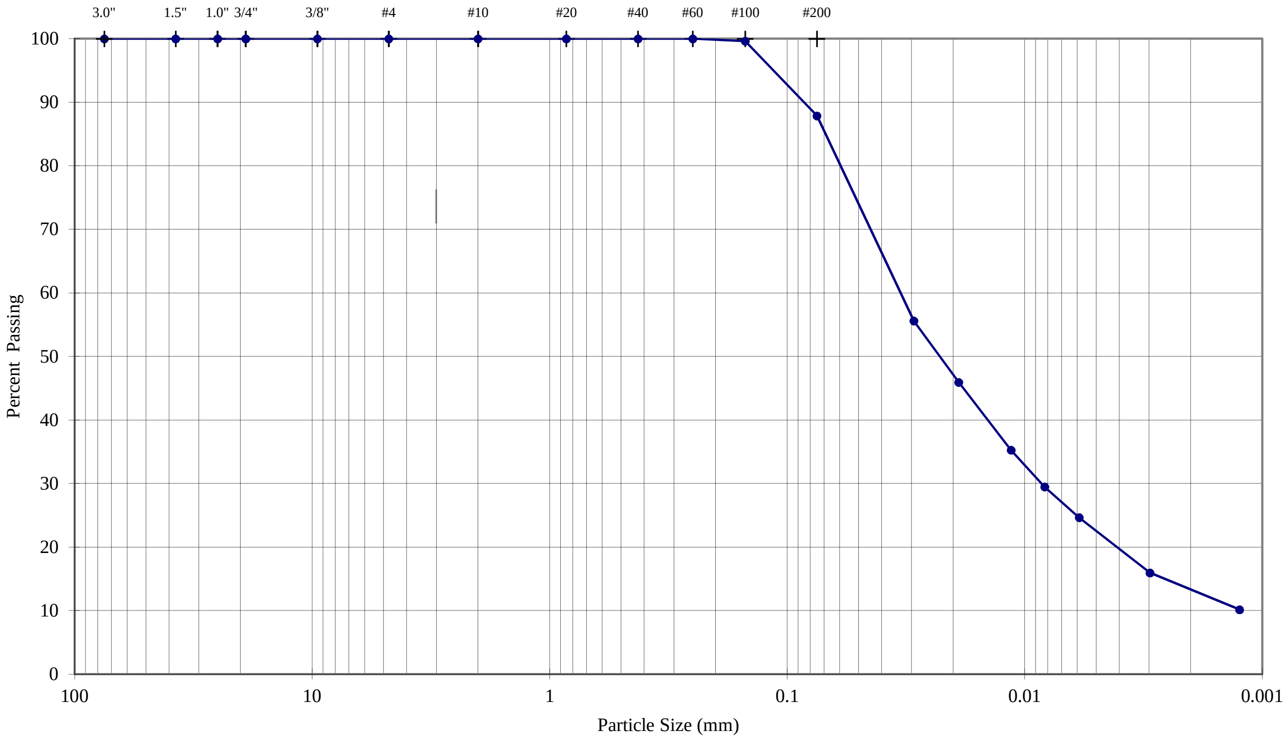
ATTACHMENT 1

LABORATORY RESULTS

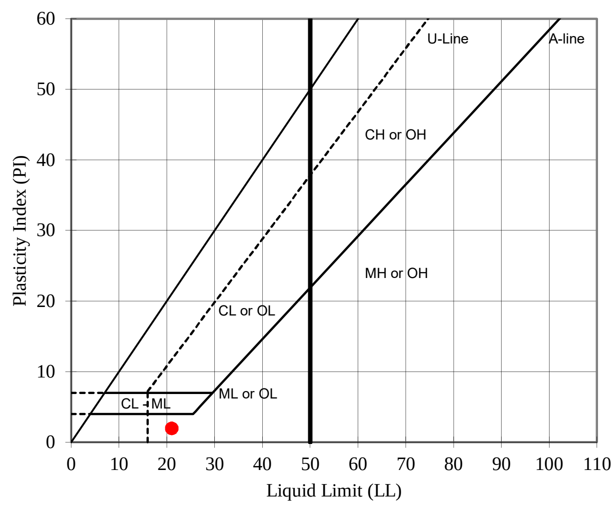
PARTICLE SIZE DISTRIBUTION & ATTERBERG LIMITS

ASTM D421, D422, D4318

PROJECT NAME: **Orvana/Copperwood TDF Final Des/MI**
SAMPLE ID: **Bucket Samples** Depth (ft): --
TYPE: **Buckets**



	Sieve	Particle Size (mm)	% Passing	Description	Percentage
Sieve Analysis (Initial Separation on No. 4 Sieve)	3.0"	75.0	**100.0**		
	1.5"	37.5	**100.0**	Coarse Gravel	**0.00**
	1.0"	25.0	**100.0**		
	3/4"	19.0	**100.0**		
	3/8"	9.5	**100.0**	Fine Gravel	**0.00**
	#4	4.8	**100.0**		
	#10	2.00	**100.0**	Coarse Sand	**0.00**
	#20	0.85	**100.0**	Medium Sand	**0.00**
	#40	0.43	**100.0**		
	#60	0.25	**100.0**	Fine Sand	**12.13**
	#100	0.15	**99.6**		
	#200	0.075	**87.9**		
Hydrometer Analysis		0.029	**55.6**	Silt or Clay Fines	**87.87**
		0.019	**45.9**		
		0.011	**35.3**		
		0.008	**29.5**		
		0.006	**24.6**		
		0.003	**15.9**		
		0.001	**10.1**		



LL	PL	PI
21	**19**	**2**

Visual Description (Golder Procedure)

Dry, dark brown CLAYEY SILT

As-Received Moisture Content (%): **148.5**

Gs: **2.86**

USCS Group Symbol: ML

Notes: 0g of particles up to 4.75mm maximum size were removed from particle size analysis sample prior to testing
Particle size analysis sample mechanically dispersed using Stirring Apparatus A for about 1 minute
Sample prepared for Atterberg Limits testing by the dry method
Material retained on No. 40 sieve removed from Atterberg Limits sample by sieving
Plastic Limit test performed by hand rolling. Method A Liquid Limit test performed using mechanical device

TECH	CP
DATE	8/6/2012
REVIEW	MB

Single Drain

Sample: Bucket Sample	
Initial Moisture Content (%) =	148.90%
Mass of Slurry + Cylinder (g) =	2700.00
Mass of Cylinder (g) =	1410.00
Total mass of Slurry (g) =	1290.00
Mass of Water Initial (g) =	771.71
Total Mass of Solids (g) =	518.29
Slurry % Solids =	40.18%

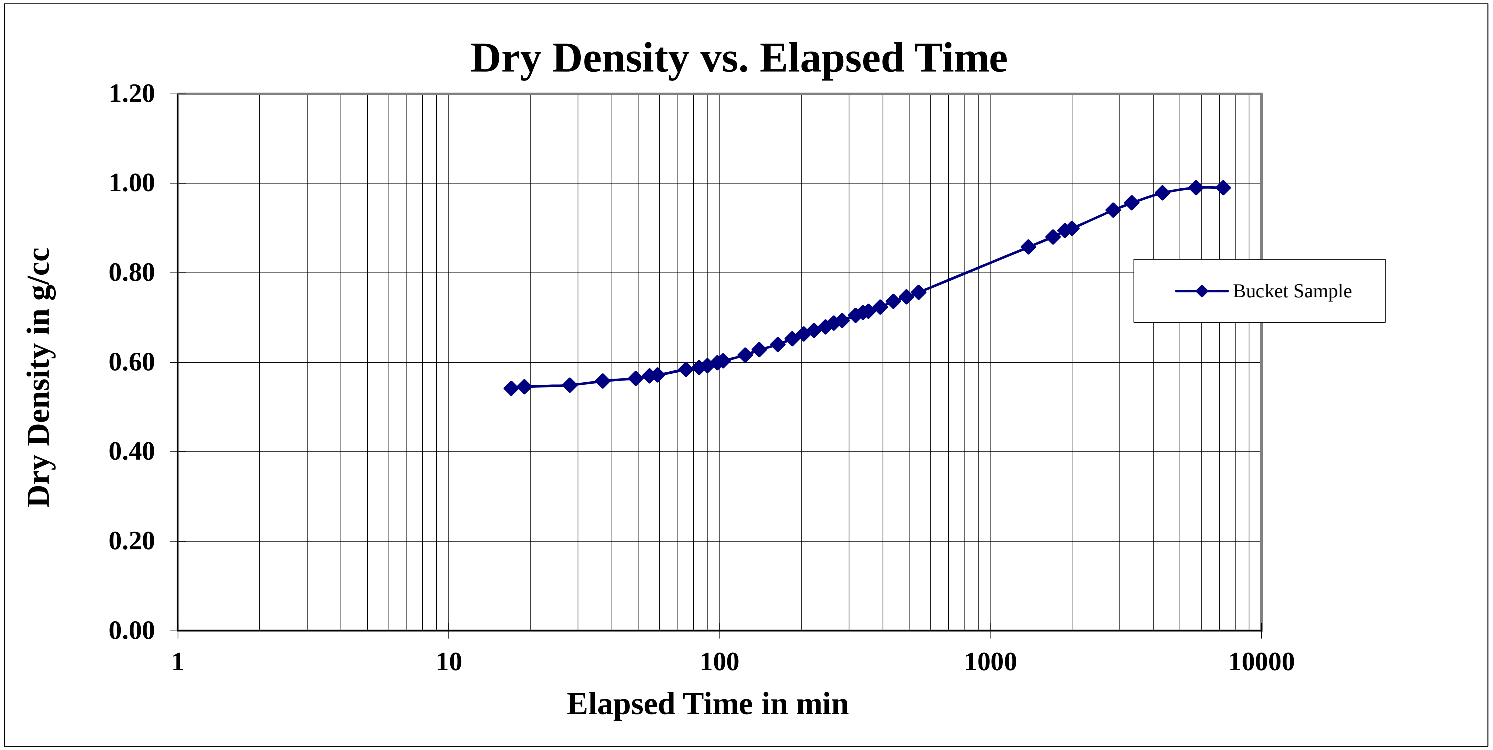
Cylinder diameter (cm) =	6.299

INITIAL MOISTURE:	
Wet Weight (g) =	204.66
Dry Weight (g) =	98.75
Tare (g) =	27.62

Reading Number	Reading Date and Time (mm/dd/yyyy hh:mm)	Elapsed Time (min)	Height of Soil (cm)	Height of Water (cm)	Vol of Water Drained (ml)	Undrained Water (ml)	Pore Water (ml)	Volume of Soil (cm^3)	Moisture Content %	Dry Density of Slurry g/cc	Dry Density of Slurry lbs/ft^3	Wet Density of Slurry lbs/ft^3
1	7/16/2012 7:35	0.0	30.8	30.8	0.0	771.7	771.7	959.8	148.9%	0.54	33.7	83.9
2	7/16/2012 7:52	17.0	30.7	30.8	0.0	771.7	768.6	956.7	148.3%	0.54	33.8	83.9
3	7/16/2012 7:54	19.0	30.5	30.8	0.0	771.7	762.4	950.5	147.1%	0.55	34.0	84.1
4	7/16/2012 8:03	28.0	30.3	30.8	0.0	771.7	756.1	944.2	145.9%	0.55	34.3	84.2
5	7/16/2012 8:12	37.0	29.8	30.8	0.0	771.7	740.5	928.6	142.9%	0.56	34.8	84.6
6	7/16/2012 8:24	49.0	29.5	30.8	0.0	771.7	731.2	919.3	141.1%	0.56	35.2	84.8
7	7/16/2012 8:30	55.0	29.2	30.8	0.0	771.7	721.9	909.9	139.3%	0.57	35.5	85.0
8	7/16/2012 8:34	59.0	29.1	30.8	0.0	771.7	718.7	906.8	138.7%	0.57	35.7	85.1
9	7/16/2012 8:50	75.0	28.5	30.8	0.0	771.7	700.0	888.1	135.1%	0.58	36.4	85.6
10	7/16/2012 8:59	84.0	28.3	30.8	0.0	771.7	693.8	881.9	133.9%	0.59	36.7	85.8
11	7/16/2012 9:05	90.0	28.1	30.8	0.0	771.7	687.6	875.7	132.7%	0.59	36.9	85.9
12	7/16/2012 9:13	98.0	27.8	30.8	0.0	771.7	678.2	866.3	130.9%	0.60	37.3	86.2
13	7/16/2012 9:18	103.0	27.6	30.8	0.0	771.7	672.0	860.1	129.7%	0.60	37.6	86.4
14	7/16/2012 9:39	124.0	27.0	30.8	0.0	771.7	653.3	841.4	126.0%	0.62	38.4	86.9
15	7/16/2012 9:55	140.0	26.5	30.8	0.0	771.7	637.7	825.8	123.0%	0.63	39.2	87.4
16	7/16/2012 10:19	164.0	26.0	30.8	0.0	771.7	622.1	810.2	120.0%	0.64	39.9	87.8
17	7/16/2012 10:40	185.0	25.5	30.8	0.0	771.7	606.6	794.6	117.0%	0.65	40.7	88.3
18	7/16/2012 10:59	204.0	25.1	30.8	0.0	771.7	594.1	782.2	114.6%	0.66	41.3	88.7
19	7/16/2012 11:18	223.0	24.8	30.8	0.0	771.7	584.7	772.8	112.8%	0.67	41.8	89.1
20	7/16/2012 11:41	246.0	24.5	30.8	0.0	771.7	575.4	763.5	111.0%	0.68	42.4	89.4
21	7/16/2012 11:59	264.0	24.2	30.8	0.0	771.7	566.0	754.1	109.2%	0.69	42.9	89.7
22	7/16/2012 12:18	283.0	24.0	30.8	0.0	771.7	559.8	747.9	108.0%	0.69	43.2	89.9
23	7/16/2012 12:52	317.0	23.6	30.8	0.0	771.7	547.3	735.4	105.6%	0.70	44.0	90.4
24	7/16/2012 13:13	338.0	23.4	30.8	0.0	771.7	541.1	729.2	104.4%	0.71	44.4	90.7
25	7/16/2012 13:29	354.0	23.3	30.8	0.0	771.7	538.0	726.1	103.8%	0.71	44.5	90.8
26	7/16/2012 14:06	391.0	23.0	30.8	0.0	771.7	528.6	716.7	102.0%	0.72	45.1	91.1
27	7/16/2012 14:52	437.0	22.6	30.8	0.0	771.7	516.2	704.3	99.6%	0.74	45.9	91.7
28	7/16/2012 15:44	489.0	22.3	30.8	0.0	771.7	506.8	694.9	97.8%	0.75	46.5	92.0
29	7/16/2012 16:37	542.0	22.0	30.8	0.0	771.7	497.5	685.6	96.0%	0.76	47.2	92.5
30	7/17/2012 6:33	1378.0	19.4	30.8	0.0	771.7	416.5	604.6	80.4%	0.86	53.5	96.5
31	7/17/2012 11:53	1698.0	18.9	30.8	0.0	771.7	400.9	589.0	77.3%	0.88	54.9	97.4
32	7/17/2012 14:53	1878.0	18.6	30.8	0.0	771.7	391.5	579.6	75.5%	0.89	55.8	97.9
33	7/17/2012 16:49	1994.0	18.5	30.8	0.0	771.7	388.4	576.5	74.9%	0.90	56.1	98.1
34	7/18/2012 6:46	2831.0	17.7	30.8	0.0	771.7	363.5	551.6	70.1%	0.94	58.6	99.8
35	7/18/2012 14:52	3317.0	17.4	30.8	0.0	771.7	354.1	542.2	68.3%	0.96	59.6	100.4
36	7/19/2012 7:19	4304.0	17.0	30.6	6.2	765.5	341.7	529.8	65.9%	0.98	61.0	101.3
37	7/20/2012 7:15	5740.0	16.8	30.6	6.2	765.5	335.4	523.5	64.7%	0.99	61.8	101.8
38	7/21/2012 7:50	7215.0	16.8	30.6	6.2	765.5	335.4	523.5	64.7%	0.99	61.8	101.8

Note: The expected error is +/- 1% based on the calcutations of the dry mass.

Golder Associates, Inc. Denver, Colorado			Title: SEDIMENTATION TESTING SAMPLE DATA AND CALCULATIONS		
Job Short Title: **Orvana/Copperwood TDF Final Des/MI**					
Sample No. Bucket Sample	**System** Single Drain	**Reviewed:** MB	**Date:** 23-Jul-12	**Job Number:** 123-88885.0002	**Figure:** 1

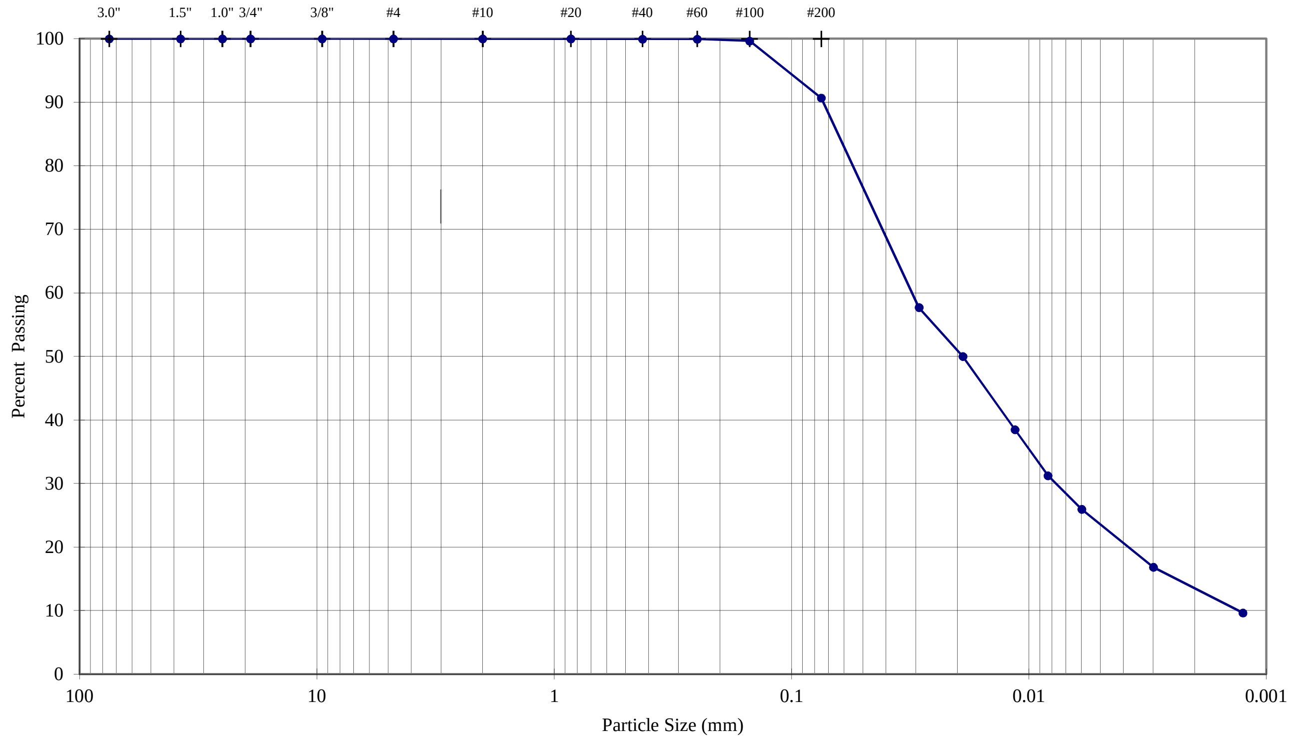


Golder Associates, Inc. **Denver, Colorado**			**Title:** **SEDIMENTATION TESTING** **GRAPHICAL DATA**		
Job Short Title: Orvana/Copperwood TDF Final Des/MI					
Sample No. Bucket Sample	**System** Single Drain	**Reviewed** MB	**Date:** 23-Jul-12	**Job Number:** 123-88885.0002	**Figure:** **2**

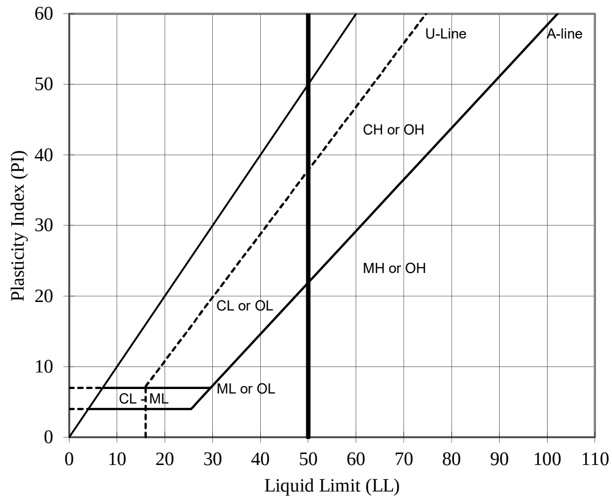
PARTICLE SIZE DISTRIBUTION & ATTERBERG LIMITS

ASTM D421, D422, D4318

PROJECT NAME: **Orvana/Copperwood TDF Final Des/MI**
SAMPLE ID: **Bucket Sample** Depth (ft): **Top 1/3**
TYPE: **Post Single Drain**



	Sieve	Particle Size (mm)	% Passing	Description	Percentage
Sieve Analysis (Initial Separation on No. 4 Sieve)	3.0"	75.0	**100.0**		
	1.5"	37.5	**100.0**	Coarse Gravel	**0.00**
	1.0"	25.0	**100.0**		
	3/4"	19.0	**100.0**		
	3/8"	9.5	**100.0**	Fine Gravel	**0.00**
	#4	4.8	**100.0**		
	#10	2.00	**100.0**	Coarse Sand	**0.00**
	#20	0.85	**100.0**	Medium Sand	**0.06**
	#40	0.43	**99.9**		
	#60	0.25	**99.9**	Fine Sand	**9.28**
	#100	0.15	**99.6**		
	#200	0.075	**90.7**		
Hydrometer Analysis		0.029	**57.7**	Silt or Clay Fines	**90.66**
		0.019	**50.0**		
		0.011	**38.5**		
		0.008	**31.2**		
		0.006	**26.0**		
		0.003	**16.8**		
		0.001	**9.6**		



LL	PL	PI
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Sample Description

Dry, dark brown

As-Received Moisture Content (%): --

Gs (Assumed): **2.86**

USCS Group Symbol: --

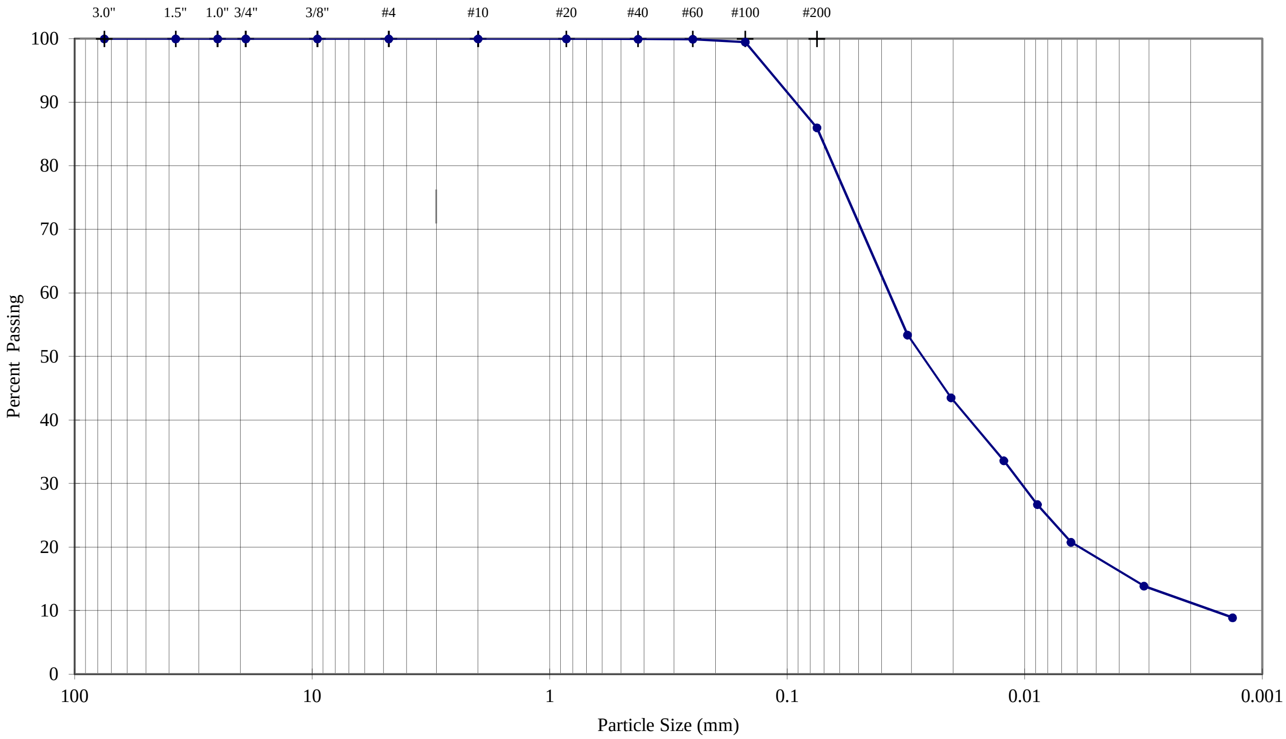
Notes: 0g of particles up to 4.8mm maximum size were removed from particle size analysis sample prior to testing
Particle size analysis sample mechanically dispersed using Stirring Apparatus A for about 1 minute

TECH	RJM/SS
DATE	8/1/2012
REVIEW	MB

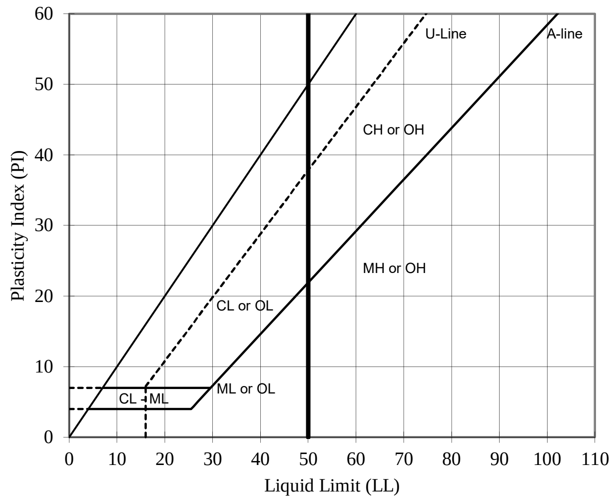
PARTICLE SIZE DISTRIBUTION & ATTERBERG LIMITS

ASTM D421, D422, D4318

PROJECT NAME: **Orvana/Copperwood TDF Final Des/MI**
SAMPLE ID: **Bucket Sample** Depth (ft): **Bottom 1/3**
TYPE: **Post Single Drain**



	Sieve	Particle Size (mm)	% Passing	Description	Percentage
Sieve Analysis (Initial Separation on No. 4 Sieve)	3.0"	75.0	**100.0**		
	1.5"	37.5	**100.0**	Coarse Gravel	**0.00**
	1.0"	25.0	**100.0**		
	3/4"	19.0	**100.0**		
	3/8"	9.5	**100.0**	Fine Gravel	**0.00**
	#4	4.8	**100.0**		
	#10	2.00	**100.0**	Coarse Sand	**0.00**
	#20	0.85	**100.0**	Medium Sand	**0.06**
	#40	0.43	**99.9**		
	#60	0.25	**99.9**	Fine Sand	**13.97**
	#100	0.15	**99.5**		
	#200	0.075	**86.0**		
Hydrometer Analysis		0.031	**53.4**	Silt or Clay Fines	**85.97**
		0.020	**43.5**		
		0.012	**33.6**		
		0.009	**26.7**		
		0.006	**20.8**		
		0.003	**13.8**		
		0.001	**8.9**		



LL	PL	PI
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Sample Description

Dry, dark brown

As-Received Moisture Content (%): --

Gs (Assumed): **2.70**

USCS Group Symbol: --

Notes: 0g of particles up to 4.8mm maximum size were removed from particle size analysis sample prior to testing
Particle size analysis sample mechanically dispersed using Stirring Apparatus A for about 1 minute

TECH	RJM/SS
DATE	8/1/2012
REVIEW	MB

Double Drain

Sample: Bucket Sample	
Initial Moisture Content (%) =	148.00%
Mass of Slurry + Cylinder (g) =	2900.00
Mass of Cylinder (g) =	1610.00
Total mass of Slurry (g) =	1290.00
Mass of Water Initial (g) =	769.84
Total Mass of Solids (g) =	520.16
Slurry % Solids =	40.32%

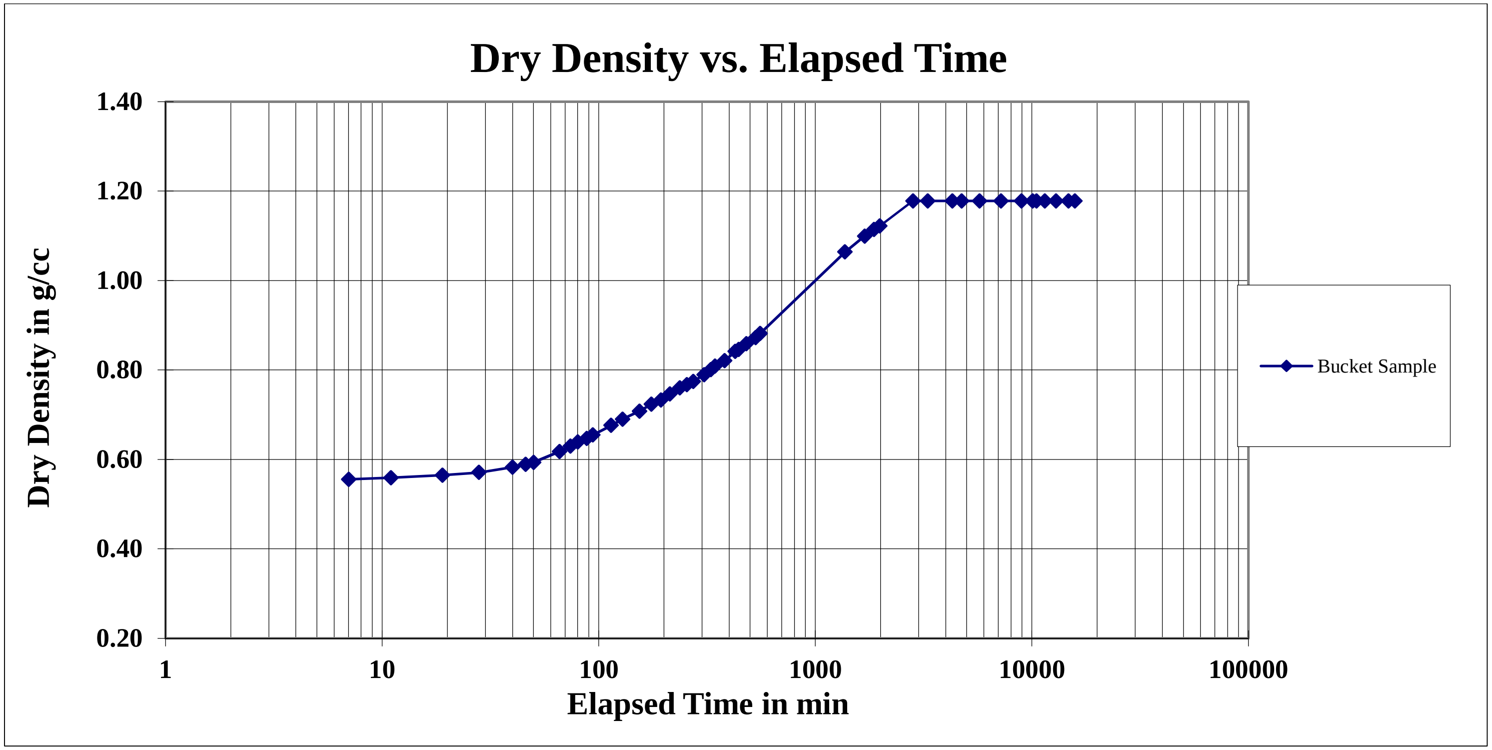
Cylinder diameter (cm) =	6.338

INITIAL MOISTURE:	
Wet Weight + tare (g) =	194.23
Dry Weight + tare (g) =	94.61
Tare (g) =	27.30

Reading Number	Reading Date and Time (mm/dd/yyyy hh:mm)	Elapsed Time (min)	Height of Soil (cm)	Height of Water (cm)	Vol of Water Drained (ml)	Undrained Water (ml)	Pore Water (ml)	Volume of Soil (cm^3)	Moisture Content %	Dry Density of Slurry g/cc	Dry Density of Slurry lbs/ft^3	Wet Density of Slurry lbs/ft^3
1	7/16/2012 7:44	0.0	30.0	30.0	0.0	769.8	769.8	946.5	148.0%	0.55	34.3	85.0
2	7/16/2012 7:51	7.0	29.7	30.0	0.0	769.8	760.4	937.0	146.2%	0.56	34.6	85.3
3	7/16/2012 7:55	11.0	29.50	29.8	5.4	764.5	755.0	930.7	145.2%	0.56	34.9	85.5
4	7/16/2012 8:03	19.0	29.2	29.8	5.4	764.5	745.6	921.2	143.3%	0.56	35.2	85.7
5	7/16/2012 8:12	28.0	28.9	29.3	5.4	764.5	751.9	911.8	144.5%	0.57	35.6	87.1
6	7/16/2012 8:24	40.0	28.3	29.1	18.8	751.1	725.9	892.9	139.5%	0.58	36.4	87.1
7	7/16/2012 8:30	46.0	28.0	29.1	18.8	751.1	716.4	883.4	137.7%	0.59	36.7	87.3
8	7/16/2012 8:34	50.0	27.8	29.1	18.8	751.1	710.1	877.1	136.5%	0.59	37.0	87.5
9	7/16/2012 8:50	66.0	26.7	28.8	18.8	751.1	684.8	842.4	131.7%	0.62	38.5	89.3
10	7/16/2012 8:58	74.0	26.2	28.8	18.8	751.1	669.1	826.6	128.6%	0.63	39.3	89.8
11	7/16/2012 9:04	80.0	25.8	28.8	18.8	751.1	656.4	814.0	126.2%	0.64	39.9	90.2
12	7/16/2012 9:12	88.0	25.5	28.7	31.9	738.0	637.0	804.5	122.5%	0.65	40.3	89.8
13	7/16/2012 9:18	94.0	25.2	28.7	31.9	738.0	627.6	795.1	120.6%	0.65	40.8	90.1
14	7/16/2012 9:38	114.0	24.4	28.7	31.9	738.0	602.3	769.8	115.8%	0.68	42.2	91.0
15	7/16/2012 9:53	129.0	23.9	28.5	40.2	729.7	584.5	754.0	112.4%	0.69	43.0	91.4
16	7/16/2012 10:18	154.0	23.3	28.5	40.2	729.7	565.6	735.1	108.7%	0.71	44.2	92.2
17	7/16/2012 10:39	175.0	22.8	28.5	40.2	729.7	549.8	719.3	105.7%	0.72	45.1	92.8
18	7/16/2012 10:58	194.0	22.5	28.1	50.5	719.3	542.6	709.9	104.3%	0.73	45.7	93.4
19	7/16/2012 11:17	213.0	22.1	28.1	50.5	719.3	530.0	697.2	101.9%	0.75	46.6	94.0
20	7/16/2012 11:41	237.0	21.7	28.0	50.5	719.3	520.5	684.6	100.1%	0.76	47.4	94.9
21	7/16/2012 11:59	255.0	21.5	28.0	50.5	719.3	514.2	678.3	98.9%	0.77	47.9	95.2
22	7/16/2012 12:17	273.0	21.3	28.0	50.5	719.3	507.9	672.0	97.6%	0.77	48.3	95.5
23	7/16/2012 12:51	307.0	20.9	27.7	72.8	697.0	482.5	659.4	92.8%	0.79	49.2	94.9
24	7/16/2012 13:13	329.0	20.6	27.7	72.8	697.0	473.0	649.9	90.9%	0.80	49.9	95.4
25	7/16/2012 13:28	344.0	20.4	27.7	72.8	697.0	466.7	643.6	89.7%	0.81	50.4	95.7
26	7/16/2012 14:05	381.0	20.1	27.4	80.7	689.2	458.9	634.1	88.2%	0.82	51.2	96.3
27	7/16/2012 14:51	427.0	19.6	27.3	80.7	689.2	446.3	618.4	85.8%	0.84	52.5	97.5
28	7/16/2012 15:07	443.0	19.5	27.3	80.7	689.2	443.1	615.2	85.2%	0.85	52.8	97.7
29	7/16/2012 15:44	480.0	19.2	27.3	80.7	689.2	433.6	605.8	83.4%	0.86	53.6	98.3
30	7/16/2012 16:34	530.0	18.9	27.3	80.7	689.2	424.2	596.3	81.5%	0.87	54.4	98.8
31	7/16/2012 17:00	556.0	18.7	27.3	96.0	673.8	402.5	590.0	77.4%	0.88	55.0	97.6
32	7/17/2012 6:33	1369.0	15.5	25.6	136.8	633.1	314.4	489.0	60.5%	1.06	66.4	106.5
33	7/17/2012 11:53	1689.0	15.0	25.3	136.8	633.1	308.1	473.2	59.2%	1.10	68.6	109.2
34	7/17/2012 14:51	1867.0	14.8	25.1	152.2	617.6	292.7	466.9	56.3%	1.11	69.5	108.6
35	7/17/2012 16:47	1983.0	14.7	25.1	152.2	617.6	289.5	463.8	55.7%	1.12	70.0	108.9
36	7/18/2012 6:44	2820.0	14.0	24.3	177.8	592.1	267.1	441.7	51.4%	1.18	73.5	111.2
37	7/18/2012 14:51	3307.0	14.0	23.8	190.2	579.7	270.5	441.7	52.0%	1.18	73.5	111.7
38	7/19/2012 7:17	4293.0	14.0	23.0	215.0	554.8	270.9	441.7	52.1%	1.18	73.5	111.7
39	7/19/2012 14:45	4741.0	14.0	22.7	226.0	543.8	269.3	441.7	51.8%	1.18	73.5	111.5
40	7/20/2012 7:14	5730.0	14.0	21.9	250.1	519.8	270.5	441.7	52.0%	1.18	73.5	111.7
41	7/21/2012 7:44	7200.0	14.0	20.7	284.7	485.2	273.8	441.7	52.6%	1.18	73.5	112.2
42	7/22/2012 12:42	8938.0	14.0	19.5	323.9	445.9	272.4	441.7	52.4%	1.18	73.5	112.0
43	7/23/2012 7:25	10061.0	14.0	18.6	347.8	422.1	276.9	441.7	53.2%	1.18	73.5	112.6
44	7/23/2012 14:48	10504.0	14.0	18.2	356.3	413.5	281.0	441.7	54.0%	1.18	73.5	113.2
45	7/24/2012 6:46	11462.0	14.0	17.6	375.7	394.2	280.6	441.7	53.9%	1.18	73.5	113.1
46	7/25/2012 6:38	12894.0	14.0	16.7	403.7	366.1	280.9	441.7	54.0%	1.18	73.5	113.2
47	7/26/2012 13:54	14770.0	14.0	15.5	438.9	330.9	283.6	441.7	54.5%	1.18	73.5	113.5
48	7/27/2012 6:42	15778.0	14.0	14.8	458.1	311.7	286.5	441.7	55.1%	1.18	73.5	114.0

Note: The expected error is +/- 2% based on the calcutations of the dry mass.

Golder Associates, Inc. **Denver, Colorado**			**Title:** **SEDIMENTATION TESTING** **SAMPLE DATA AND CALCULATIONS**		
Job Short Title: **Orvana/Copperwood TDF Final Des/MI**					
Sample No. Bucket Sample	**System** Double Drain	**Reviewed:** MB	**Date:** 30-Jul-12	**Job Number:** 123-88885.0002	**Figure:** 1



Golder Associates, Inc. Denver, Colorado			Title: SEDIMENTATION TESTING GRAPHICAL DATA		
Job Short Title: Orvana/Copperwood TDF Final Des/MI					
Sample No. **Bucket Sample**	Depth **Double Drain**	Reviewed: MB	Date: 30-Jul-12	Job Number: 123-88885.0002	Figure: 2

Copperwood Project Alternatives Analysis Update
Copperwood Resources Inc.
June 7, 2018

TABLE OF CONTENTS

LIST OF FIGURES

LIST OF TABLES

Revised Copperwood Project Alternatives Analysis Update

Copperwood Resources, Inc.

June 7, 2018

1.0 INTRODUCTION

On February 22, 2013, Orvana Resources US Corp ("Orvana") was issued Permit Number 12-27-0050-P by the Michigan Department of Environmental Quality (MDEQ) under Part 301 (Inland Lakes and Streams) and Part 303 (Wetland Protection) of the Natural Resources and Environmental Protection Act (NREPA), Public Act 451 of 1994, as amended, to develop a copper mine ("Copperwood Project") in Gogebic County. Highland Copper Company Inc. ("Highland"), a Canadian copper development company, acquired the Copperwood Project from Orvana Minerals Corp. in June 2014. Subsequent to this acquisition, Highland changed the name of Orvana to Copperwood Resources, Inc. ("Copperwood") by amending the articles of incorporation. The Michigan Department of Licensing and Regulatory Affairs issued a certificate of endorsement to Highland for the Copperwood amended articles of incorporation on March 21, 2017.

Highland also signed an interim agreement in May 2014 with First Quantum Minerals Ltd. to acquire the Copper Range Co. ("CRC") assets that CRC owned, and environmental remediation liabilities CRC is responsible for, at the former White Pine Mine Site ("White Pine"), which is approximately 44 miles (by public roadways) and approximately miles (overland, straight-line distance) from Copperwood. Due to the complexity of the environmental liability acceptance and economic constraints associated with the White Pine site, Highland has not yet executed a final closing of this acquisition but intends to do so in the near future. The Copper Range acquisition will include mineral rights, certain surface property, the former South Tailings Disposal Area and the North #1 and North #2 Tailings Basins of the White Pine Mine Site.

Subsequent to acquisition of the Copperwood Project and the interim agreement for acquisition of the White Pine Mine assets, Highland initiated a prefeasibility study ("PFS") with the objectives of updating the Copperwood and White Pine mineral resource estimates, examining options for mining, ore processing, tailings disposal and water management while evaluating the economics of these options. At the time, the Copperwood Project held all the major permits necessary to construct and operate a stand-alone facility at Copperwood, producing ore concentrate as a final product with an on-site tailings disposal facility. Conversely, at White Pine, Highland held no environmental permits for that project. All baseline studies that have been conducted at White Pine are dated and/or related to ongoing monitoring required by CRC's Consent Decree with the State of Michigan and Remedial Action Plan for restoration, cleanup and closure activities at the White Pine site.

In addition, prior to the Highland Acquisition of White Pine, CRC had sold much of the White Pine on-site equipment and demolished the entire ore processing facility. CRC was also well into the process of restoring the former tailings disposal areas mentioned above. Restoration of the South Basin included capping the area of deposited tailings with clay material, establishment of vegetation on the cap and construction of a new drainage outlet to the North #1 Basin. The restoration plan for the North #1 and #2 Tailings Basins is to establish vegetation directly on the tailings deposit surfaces to achieve performance standards specified in the RAP and construction of a new drainage outlet from the North #2 Basin leading to the existing outfall covered by a NPDES permit. In addition, a third-party company controls access to the remaining underground mine workings that are not flooded with water, as part of CRC's site closure activity; and other third-party companies have purchased most of the remaining facilities at the process plant area of the site such as the power plant, former copper refinery, shops and warehouses. For Highland to now be able to use the White Pine site to concentrate ore into a salable product, or to access the mineral reserves being acquired from CRC, it would be necessary to construct a new processing plant, and a new mine entry would be required to reach the ore on that site.

The initial focus of the PFS mentioned above was to develop Copperwood as a mine-only facility, with ore transported to White Pine for processing in a new mill and tailings disposal at White Pine in either one or more of the former tailings basins, in the underground mine workings or in a new disposal facility. Concurrently with development of a Copperwood mine-only facility with ore processing and a tailings disposal plan in place at White Pine, a new underground mine access would be developed to re-start mining at White Pine. Highland completed two draft studies for ore transport options from Copperwood to White Pine and a draft tailings disposal tradeoff study for the White Pine site. Engineering studies for mine access and process plant options at White Pine were ongoing when PFS activity was suspended by Highland in early 2016 due to a number of economic constraints.

When Highland secured additional project financing in early 2017, Highland made their decision to develop the Copperwood Project in a manner similar to that described in Orvana's 2012 feasibility study and permitted by the MDEQ in 2012 and 2013. The reasons for the change in focus from a combined Copperwood-White Pine Project to a stand-alone Copperwood Project included Copperwood having previously completed environmental baseline and feasibility studies and having obtained all major permits. In addition, Highland determined that the shorter estimated time to project completion and production startup at Copperwood, along with a much smaller scale of financing required, made the stand-alone Copperwood project the most feasible and prudent alternative. At the same time, a new plan for the White Pine project is being developed, including a plan to prepare new baseline environmental studies, to prepare a feasibility study and to begin an effort to obtain the required permits such that construction can be completed prior to depletion of ore reserves at the Copperwood Project. From an operational perspective, Highland would apply economic returns from a successfully operating Copperwood Project to the White Pine Project to in order to offset a portion of the White Pine financing requirement.

As it relates to the Copperwood Project, Highland engaged G Mining Services Inc. ("GMining") of Brossard, Quebec in May 2017 to review preliminary design plans, incorporate necessary revisions and improvements, and ultimately produce an updated feasibility study report based upon those revisions. That feasibility study, which is a critical component of Canadian governmental review for financial market purposes, is not yet complete but is expected to become final in late May of this year. However, project planning (including site planning and resource processing) has advanced enough to submit new permit applications, including this specific application under Part 301 and Part 303. At the same time, Copperwood has also applied to the MDEQ Oil, Gas and Minerals Division on March 21, 2018 for an amendment to its MDEQ Mining Permit Number MP 01 2012 issued in accordance with the Part 632 Nonferrous Metallic Mining regulations of NREPA to allow for changes to the Copperwood Project design plans. In addition, an NREPA Part 55 Air Permit to Install application was recently submitted to the MDEQ Air Quality

Division, on March 22, 2018. Surface facilities and operations described in the original Part 632 permit and subsequent amendment of February 7, 2013, remain valid. However, the locations of some of the proposed facilities have been modified, and two new facilities have been added to the Copperwood Project (Figure 1-1) as compared to that considered and permitted in 2013. The primary changes being proposed are:

- Process plant (mill site) proposed further west than the previous site;
- Addition of an outdoor ore stockpile;
- Modification of the mine ventilation plan;
- Addition of on-site power generation, using natural gas fuel.

Revised Figure 1-1, June 7, 2018

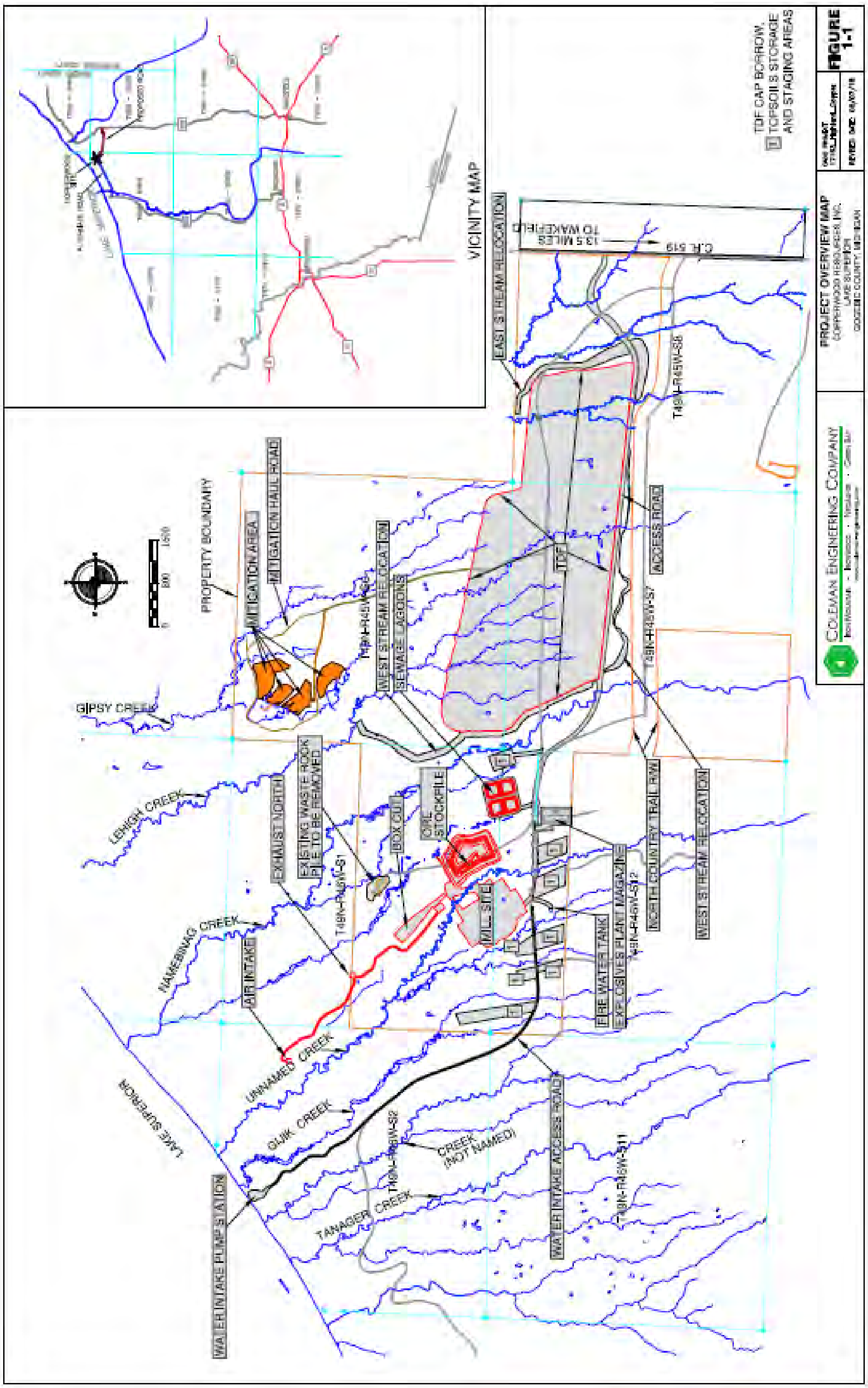


FIGURE 1-1
PROJECT OVERVIEW MAP
COLEMAN ENGINEERING COMPANY
VICINITY MAP
TDF CAP BORROW, TOPSOILS STORAGE AND STAGING AREAS
PROPERTY BOUNDARY
MITIGATION AREA
MITIGATION HAUL ROAD
WEST STREAM RELOCATION
SEWAGE LAGOONS
EAST STREAM RELOCATION
13.5 MILES TO WAKEFIELD
C.R. 519
TDF
ACCESS ROAD
GIPSY CREEK
LEHIGH CREEK
EXHAUST NORTH
EXISTING WASTE ROCK PILE TO BE REMOVED
BOX CUT
ORE STOCKPILE
MILL SITE
NORTH COUNTRY TRAIL R/W
WEST STREAM RELOCATION
FIRE WATER TANK
EXPLOSIVES PLANT MAGAZINE
NAMEBINAG CREEK
AIR INTAKE
UNNAMED CREEK
LAKE SUPERIOR
GUIK CREEK
CREEK (NOT NAMED)
TANAGER CREEK
WATER INTAKE ACCESS ROAD
WATER INTAKE PUMP STATION

2.0 PROJECT PURPOSE

The proposed Project Purpose for the Copperwood Project is: **To construct an underground copper mine and related above-ground processing facilities.**

3.0 EXPLANATION OF COSTS AND BUDGET

The understanding of the role of costs in the proposed Copperwood Project is an important consideration due to the ramifications of costs on the viability of the project. Therefore, the explanation of costs and budget is provided in this Alternatives Analysis to provide the proper context and information necessary to assist in the review of this aspect of the application for permit. This information is not provided in an attempt to demonstrate that all such costs override the proposed natural resources impacts, but instead to provide the importance of the relative costs to this project as part of the complete analysis of the alternatives to any given aspect of the Copperwood Project. In order to meet State and Federal statutory requirements, a complete analysis of alternatives is necessary in order to demonstrate that there is no feasible or prudent alternative to the project as proposed, and it represents the least environmentally damaging practicable alternative, respectively.

As explained in Section 1.0 above, an updated feasibility study report for the project is currently being developed. Since Highland, the parent company of Copperwood, is a publicly-owned mining company listed on the financial stock exchanges in Canada, Highland must comply with specific Canadian regulations such as Canadian National Instrument 43-101 ("NI 43-101"). NI 43-101 is a Canadian National Instrument for the Standards of Disclosure for Mineral Projects. This Instrument is a codified set of rules and guidelines for reporting and displaying information related to mineral properties owned by, or explored by, companies that report these results on stock exchanges in Canada. This Canadian government disclosure process is rigid and exhaustive. Individuals providing input in the development of the NI 43-101 reports must meet minimum requirements to be considered a Qualified Person. There are a series of NI 43-101 compliant reports that must be completed by a Canadian company such as Highland as a proposed project is reviewed and studies are conducted in order to meet project viability threshold requirements.

As a requirement of NI 43-101, Highland is in the process of updating Orvana's original Bankable Feasibility Study ("BFS") for the Copperwood Project that was dated March 21, 2012. Highland has enlisted the aid of outside consulting groups with Qualified Persons who have expertise in various areas of knowledge to provide input into the updated BFS. Under NI 43-101, Highland is required to provide public disclosure of scientific and technical information about the Copperwood Project, which in itself requires alternatives analysis for the various aspects of the project. While the updated BFS report is not expected to be completed for public release until late May 2018, stream and wetland impacts for the Copperwood Project have been identified and defined, such that a Part 301 and Part 303 permit application can be submitted.

The staged process for developing a BFS typically involves the completion of a Preliminary Economic Assessment ("PEA") that provides initial estimates within 30%-40% of capital cost requirements and timing, production levels, operating costs, and revenue projections. The PEA is usually the first hurdle for a project to show positive results before moving into a more detailed

Pre-Feasibility Study ("PFS") that further refines the costs and continues above and beyond the PEA to provide confidence that a project is financially viable. Normal guidelines for an acceptable PFS require that cost estimates be within 25%-30%.

A BFS provides an even more detailed study based upon the work to-date on a project and is typically the last hurdle requiring positive results that a project needs to clear in order to be considered financially viable by the Canadian government (and investors). Higher confidence levels are required on costs, usually within 10%-15%, to the extent that budgets for the primary capital equipment investment are required, along with detailed explanation of the capital investment and startup costs. The final result of the BFS is a determination of project viability from an investor's standpoint.

With the results of BFS cost estimates, there are several measures that may be calculated to gauge the viability of a project by investors; the most common being Internal Rate of Return ("IRR"), Years Payback, and Net Present Value ("NPV"). In each case, the timing of the capital costs, total operating costs and projected revenues based on the BFS are used to calculate the results. In the case of NPV there is discount rate that is used which could be considered a risk factor. This risk factor takes into account different variables that could alter the outcome of project such as company risk, country risk, project type risk (e.g. surface, underground, dredging), and commodity type. The higher the risk, the lower the results of a project will be calculated, which shows what the value of a project is in NPV, bringing all costs into today's dollars.

All three of these measures (IRR, Years Payback and NPV) are commonly used to measure the viability of a project by investors with different weights being given to each. The IRR is commonly used to gauge the viability of most mining projects (which can have returns in the wide range of 15% to 50%, or sometimes even higher). These rates of return are typically necessary within the industry to allow for increases in costs over the longer period of mine development, as well as to support exploration costs for other sites that do not develop into viable projects, as well as widely fluctuating metal prices which are common in the world economy.

Therefore, as with all mining projects, the financial viability of the Copperwood Project depends on a number of variables. In the case of Copperwood, those variables include the price of copper on world markets, capital and development costs of the project, operating costs, production rates and grade of the ore among others. Based on a sensitivity analysis of Orvana's 2012 BFS, the price of copper and the capital costs have the most influence on the outcome the Copperwood Project's IRR. A minimum price (price deck) for copper is utilized in the BFS in order to provide the most conservative financial scenario for the project. Depending on the sentiment or outlook of investors, the forecasted metal price can vary considerably depending on whether an investor is bearish or bullish on the outlook. In most cases, the BFS uses a published, or otherwise generally accepted, price deck for copper. As an example, in the Orvana BFS for Copperwood the price deck is $2.75/lb. With copper priced at $2.75/lb., the IRR for the Copperwood Project calculates to 17.2%. If copper prices exceed the price deck, financial viability of the project improves.

A more conservative investor, using a lower copper price deck of $2.50/lb., would estimate the IRR for the Copperwood Project at 11.1%, which demonstrates how the price of copper can affect this project, providing for additional difficulty in attracting investors. With copper at $2.50/lb., the project would be considered a “marginal” rating for attracting investors. This 11.1% IRR reflects the best estimate of the 2012 Orvana design for the Copperwood Project, with any additional costs further reducing this IRR. If any changes to the Orvana project resulted in substantially higher capital costs or operating costs, there would be a significant negative impact to the financial viability of the project. For example, if pre-production capital, estimated in 2012 at approximately $218 million, is increased by 10% (approximately $22 million), the IRR would drop to 9.1%. If operating costs for life-of-mine increased by 10% (approximately $95.4 million), the IRR would decrease to 7.2%. Any one of these increases in cost would impact the project to the point that the ability to attract investors would be greatly diminished and put the project in jeopardy of not being able to obtain financing. This simple analysis is offered, by way of example, to demonstrate that any changes to the Project that substantially increase capital costs or operating costs would have a significantly detrimental effect on the economic viability of the Copperwood Project.

The low-grade copper-bearing ore at the Copperwood Project consists of fine-grained chalcocite that occurs in the Nonesuch Shale. This type (grad) of ore requires extensive processing to remove the copper-bearing minerals (chalcocite) from the Shale. The process involves handling, milling, and processing large volumes of rock to concentrate the copper into a salable product suitable for further refinement via smelting. While historically not practicable, the mining process for such a low grade of copper ore such as what is found at Copperwood has recently advanced such that extraction and processing is financially feasible if the price of copper stays above the price deck and capital investment and life-of-mine mining costs are reasonable.

Orvana’s completed BFS from March 2012 is included for reference as Appendix A. For the updated BFS being completed by GMining for the Copperwood Project, capital and operating cost estimates are not currently advanced to a point where they can be used to calculate IRR values for alternatives being considered. Completion of the study, as noted above, is expected in late May 2018.

4.0 DESCRIPTION OF MINE DEVELOPMENT AND TIMELINE

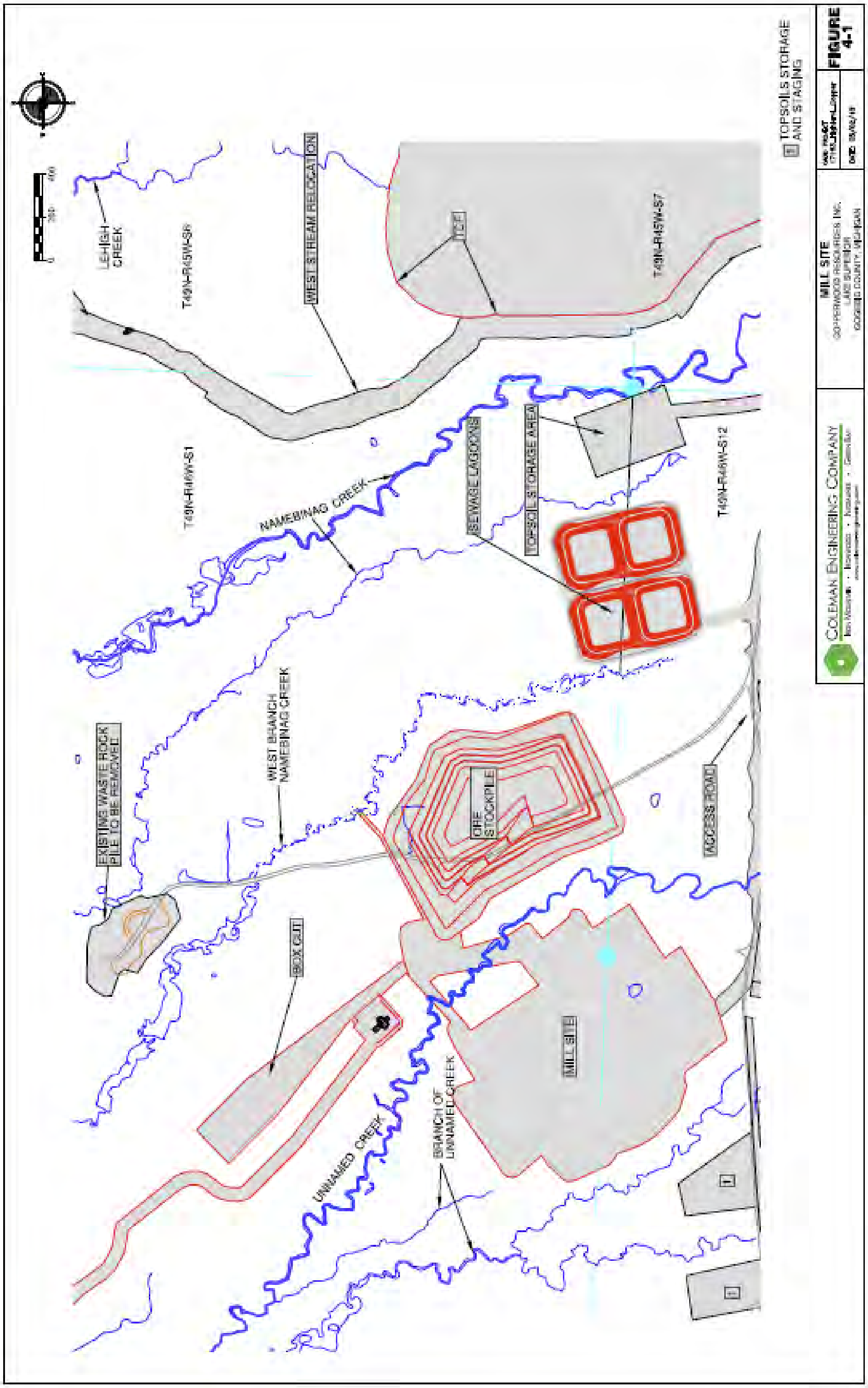
There are seven categories of activities proposed at the Copperwood Project that require a permit under Part 301 and Part 303 of NREPA. The seven categories are:

1. Main access road
2. Underground mine entrance
3. Mill site and outdoor ore stockpile
4. Tailings disposal facility (TDF)
5. Surface water and stormwater discharge management
6. Mine ventilation raises and access road
7. Water intake and access road

The general description of project development begins with construction of an access road to enable equipment and materials to be brought to the mine site for construction of the underground mine entrance and TDF. Following completion of the access road and development of the construction staging areas, work will commence on the underground mine entrance, mill site grading, the ore stockpile and TDF. The initial activity will involve clearing the site; i.e. removal of vegetation and topsoil. Woody vegetation will be harvested by logging merchantable timber and chipping the tops and smaller trees for sale to available users. Herbaceous vegetation will be removed, with the topsoil being stockpiled for mine reclamation purposes at locations shown on the project plans. Topsoil from the TDF site will be stockpiled in the locations shown on Figure 1-1 (also shown on Figure 4-1), with low permeability subsoils being stockpiled separately for use during mine closure/reclamation in the area between the mine access road and the south face of the TDF.

Once the mill site is prepared, construction will begin on the various buildings shown on Figure 4-1, with the exception of the water treatment plant which will be built at a future date as explained later in this document. Construction of the outdoor ore stockpile and Phase 1 of the TDF will begin after site preparation is completed. Construction of the underground mine entrance will not commence until the ore stockpile base is prepared and Phase 1 of the TDF is completed and able to store water pumped from the underground mine development.

A description of the seven categories of activities proposed at the Copperwood Project is provided in the following sections, including the alternatives that have been considered for each of the activities.


LEHIGH CREEK
WEST STREAM RELOCATION
TCF
T49N-R45W-S6
T49N-R45W-S7
T49N-R46W-S1
T49N-R46W-S12
NAMEBINAG CREEK
SEWAGE LAGOONS
TOPSOIL STORAGE AREA
EXISTING WASTE ROCK PILE TO BE REMOVED
WEST BRANCH NAMEBINAG CREEK
ORE STOCKPILE
ACCESS ROAD
BOX CUT
MILL SITE
UNNAMED CREEK
BRANCH OF UNNAMED CREEK
TOPSOILS STORAGE AND STAGING
MILL SITE
COPPERWOOD RESOURCES INC.
COLEMAN ENGINEERING COMPANY
FIGURE 4.1

5.0 ACCESS ROADS

Alternatives for public road access to the Copperwood Project site and the mine access road from existing Gogebic County roads are assessed in this section. All-season public road access is required for the Copperwood Project due to the materials that will be transported to the site during construction and operation. That access will be used by contractors to bring equipment, construction materials and other supplies to the site, as well to provide for employee access and emergency vehicles. In addition, once operational, ore concentrate will be trucked from the site to off-site smelting facilities.

5.1 County Road Access

Two public roads provide access north from U.S. Highway 2 ("US-2") and Michigan Highway 28 ("M-28") from Bessemer and Wakefield to the shores of Lake Superior and are adjacent to the mine site (Figure 5-1). Gogebic County Road 513 ("CR 513") runs north from Bessemer to the Black River Harbor, which is located on the west side of the Black River. The Copperwood Project site is about two miles east-northeast of the Black River Harbor. Extending CR 513 to the Copperwood site would require construction of a new road approximately 3 miles in length and would need to include a bridge over the Black River approximately 180 feet in length. There would likely be wetland impacts resulting from construction of this new road. Portions of the Black River on U.S. Forest Service land is a designated Scenic River, therefore construction of a bridge over the river may be problematic in regard to the river designation (Figure 5-1a). Proposing an extension of CR 513 on Federal land (i.e. Ottawa National Forest) would likely require a Special Use Permit and an Environmental Assessment.

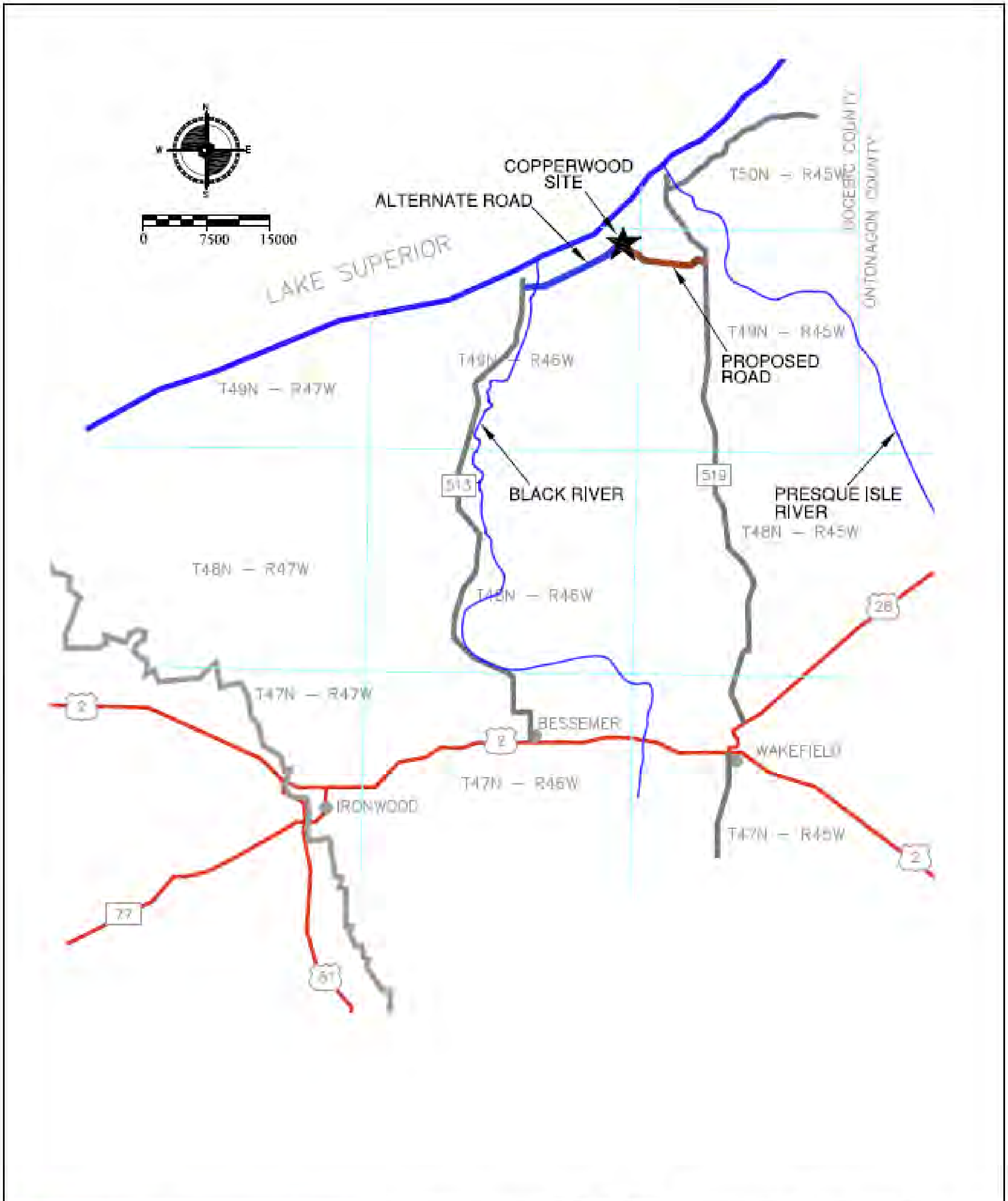
Due to the presence of a more prudent and practicable alternative for public road access as described below and proposed in this permit application, the CR 513 route was determined to be not feasible or prudent and was not selected.

Gogebic County Road 519 ("CR 519") runs north from M-28 east of Wakefield and continues to the Porcupine Mountains Wilderness State Park campground at the mouth of the Presque Isle River and runs just east of the project site (Figure 5-1). Use of CR 519 for mine site access will require upgrades so as to ensure year-round access to the Copperwood site, including replacement of road base to provide drainage, widening the road section, and asphalt paving. The length of the reconstruction of CR 519 from M-28 to the Copperwood site is 13 miles terminating at the proposed access road entrance to the Copperwood Project.

In order to determine the most appropriate county road access to the Copperwood site, in 2011 the Gogebic County Road Commission ("GCRC") conducted a project review process in conjunction with the Michigan Department of Transportation ("MDOT"). As a result, GCRC and MDOT determined that CR 519 was the best alternative route for access to the Copperwood Project, having the least impact on natural resources in comparison to any CR 513 route extension. As a result, the necessary CR 519 improvement projects were permitted by MDEQ in 2011, somewhat independent of the subject Copperwood wetland/stream permit application.

At that time, funding was acquired for the CR 519 reconstruction through an MDOT grant in the amount of $2.3 million, with Orvana to provide $841,000 and the GCRC to provide $350,000 matching funds for the project. During the 2011 and 2012 construction seasons, the GCRC completed culvert replacements along the CR 519 route. However, the remainder of the reconstruction project was placed on hold due to economic constraints on copper production and markets. This inactivity resulted in the MDOT grant being awarded to another applicant.

In the 4th quarter of 2017, Copperwood and the GCRC applied for new Transportation Economic Development Fund ("TEDF") Category A grant funding for the renewed CR 519 reconstruction project in 2019. That grant was subsequently awarded by MDOT in February 2018, with a revised overall cost estimate for the project of $7,958,869, including $4,775,321 in TEDF Category A funding and $3,183,548 in funding from Copperwood Resources. However, the TEDF 2019 grant is contingent upon the mine obtaining all applicable State and Federal permits and the continued funding of the TEDF Category A program by the Michigan Legislature.

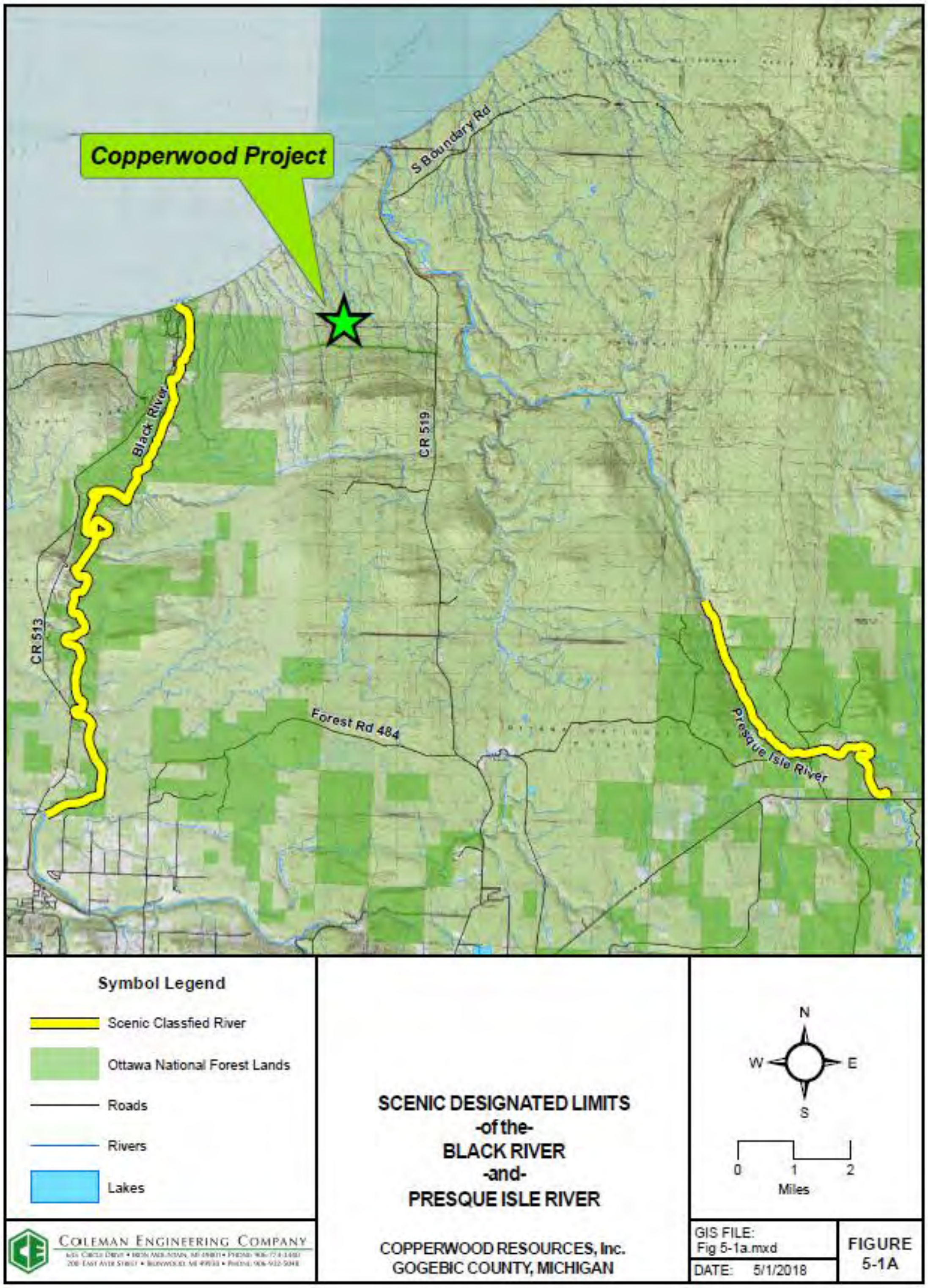


COLEMAN ENGINEERING COMPANY
IRON MOUNTAIN • IRONWOOD • NEGAUNEE • GREEN BAY
www.colemanengineering.com

VICINITY MAP
COPPERWOOD RESOURCES, INC.
LAKE SUPERIOR
GOGEBIC COUNTY, MICHIGAN

CADD PROJECT 17163_Highland_Copper
DATE: 05/02/2018

FIGURE 5-1



Copperwood Project
S Boundary Rd
Black River
CR 519
CR 513
Forest Rd 484
Presque Isle River
Symbol Legend
Scenic Classfied River
Ottawa National Forest Lands
Roads
Rivers
Lakes
SCENIC DESIGNATED LIMITS
-of the-
BLACK RIVER
-and-
PRESQUE ISLE RIVER
N
W
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S
0
1
2
Miles
COLEMAN ENGINEERING COMPANY
COPPERWOOD RESOURCES, Inc.
GOGEBIC COUNTY, MICHIGAN
GIS FILE:
Fig 5-1a.mxd
DATE: 5/1/2018
FIGURE
5-1A

5.2 Mine Access Road

An existing logging road (Camp 7 Grade) presently provides access to the Copperwood site from CR 519. Camp 7 Grade is located in the proposed TDF footprint, so using that existing logging road along its entire length is not a possibility for the mine access road (Figure 5-2). Besides the Camp 7 Grade, three other alternative mine access road routes were evaluated; for purposes of this application, referred to as the south route, the north route, and the proposed route (Figure 5-2).

5.2.1 South Route Alternative for the Mine Access Road

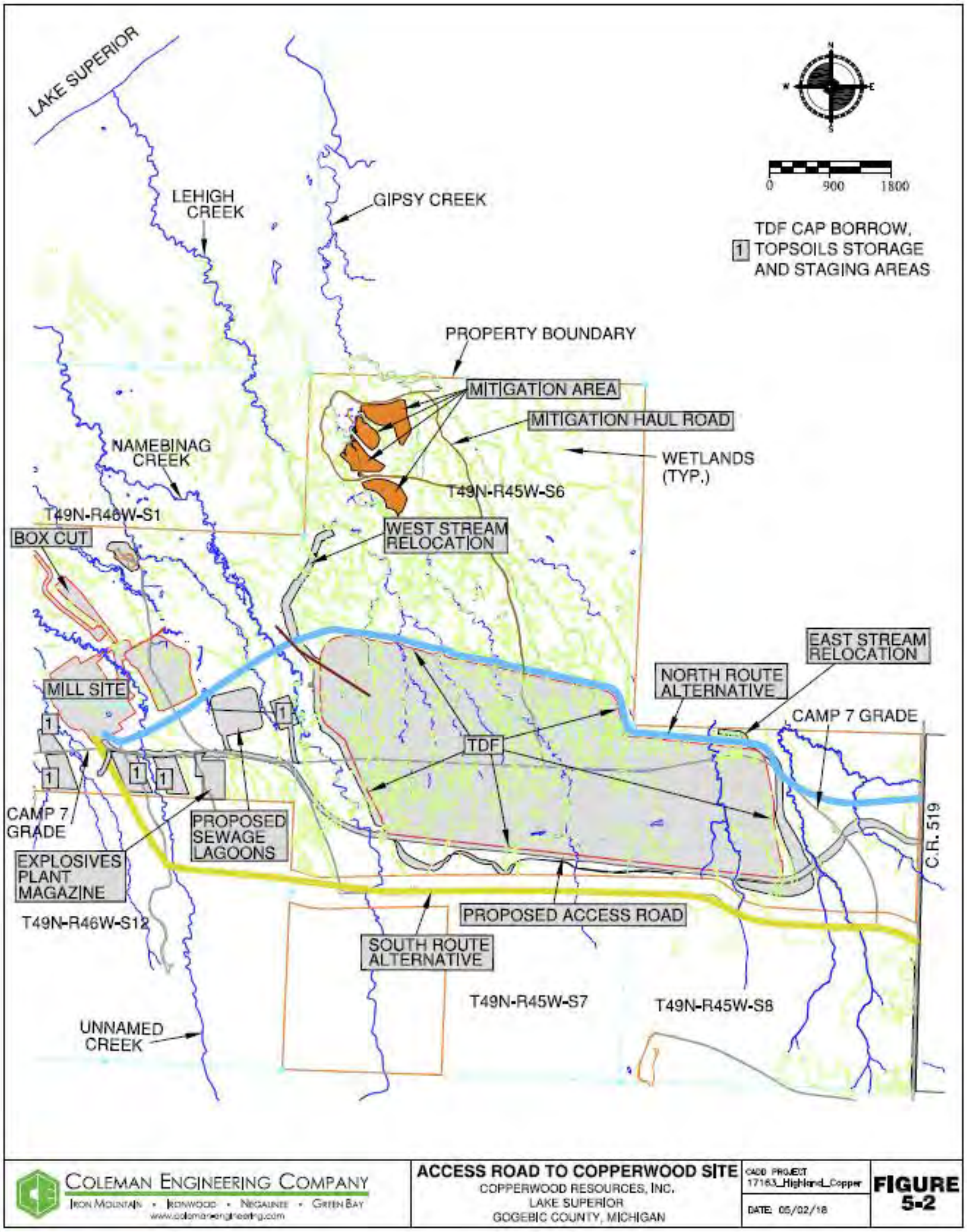
The south route, which is the existing North Country Trail, is mostly within a corridor of land owned in fee title by the U.S. Forest Service ("Forest Service") and is part of the Ottawa National Forest (Figure 5-2). This route would be approximately 13,470 in length, of which approximately 8,615 feet is on public land. There would be six stream crossings and wetland impacts of approximately 2.96 acres. In addition, any proposed abandonment or relocation of the North Country Trail by the Forest Service would likely require an application for a Special Use Permit and an Environmental Assessment. Orvana participated in direct discussions with staff of the Ottawa National Forest in 2011 regarding a possible land trade that would have allowed Orvana to take ownership of the federally-owned lands in Sections 7 and 8 that are part of the North Country Trail. The Ottawa National Forest Supervisor at the time, Anthony Scardina, indicated to Orvana during those discussions that the associated process required of the Federal government to make such a land deal would take four to five years due to the NEPA/EIS compliance process, and might ultimately not be approved. In addition, Mr. Scardina indicated at that time he would not be in favor of such a plan because the Forest Service intends/needs their current property to someday access other Forest Service property to the west. Orvana took no further action on this specific issue after receipt of that information from Mr. Scardina.

Utilizing the Forest Service property would likely require a relocation of the North Country Trail to connect back to the unaffected segments of the Trail. In addition, the current intersection of the North Country Trail and CR 519 is located on a steep grade on CR 519. As such, truck traffic ingress and egress on the south route alternative would have difficulties, and likely be a significant traffic safety concern.

This alternative was determined by Copperwood to not be feasible or prudent due in large part to public ownership of most of the route and the truck traffic concerns at the intersection with CR 519.

5.2.2 North Route Alternative for the Mine Access Road

The north route alternative for the mine access road was initially investigated by Orvana in an effort to increase the buffer distance from the North Country Trail. The north route would be approximately 13,540 feet in length (Figure 5-2), with 14 stream crossings and approximately 4.27 acres of wetland impacts. The north route alternative was also determined by Copperwood to not be feasible or prudent, in this case due to the large amount of wetland impacts and number of stream crossings.



LAKE SUPERIOR
LEHIGH CREEK
GIPSY CREEK
0 900 1800
TDF CAP BORROW,
1 TOPSOILS STORAGE
AND STAGING AREAS
PROPERTY BOUNDARY
MITIGATION AREA
MITIGATION HAUL ROAD
NAMEBINAG CREEK
WETLANDS (TYP.)
T49N-R45W-S6
T49N-R46W-S1
BOX CUT
WEST STREAM RELOCATION
EAST STREAM RELOCATION
NORTH ROUTE ALTERNATIVE
MILL SITE
CAMP 7 GRADE
TDF
CAMP 7 GRADE
PROPOSED SEWAGE LAGOONS
EXPLOSIVES PLANT MAGAZINE
C.R. 519
PROPOSED ACCESS ROAD
T49N-R46W-S12
SOUTH ROUTE ALTERNATIVE
T49N-R45W-S7
T49N-R45W-S8
UNNAMED CREEK
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ACCESS ROAD TO COPPERWOOD SITE
COPPERWOOD RESOURCES, INC.
LAKE SUPERIOR
GOGEBIC COUNTY, MICHIGAN
CADD PROJECT
17163_Highland_Copper
DATE: 05/02/18
FIGURE 5-2

5.2.3 Proposed Mine Access Road

The proposed access road is approximately 13,100 feet in length, with about 1,500 feet of the proposed route being located on an existing logging road (Camp 7 Grade). The proposed road would be connected to CR 519 at a location that provides a safe horizontal and vertical alignment at the intersection with CR 519. The proposed route is near the south side of the TDF, with a strip of land between the TDF and the access road to be used for a low permeability soil stockpile that will be used to cap the TDF during mine closure.

The proposed mine access road would provide a buffer of approximately 300 feet from the North Country Trail and would impact approximately 2.07 acres of wetlands. There are eight stream crossings on the proposed route, of which five are existing crossings that will be upgraded and improved by the installation of new pipe arch culverts. Presently these five road crossings are corrugated metal pipe culverts and are either undersized or installed improperly, which causes streambed erosion during periods of high flow, usually during spring snowmelt. As determined by stream surveys conducted for the Part 632 Environmental Impact Assessment of the Copperwood Project, streams that cross the proposed mine access road are ephemeral streams that have flow only part of the year; fish do not inhabit these stream segments.

In regard to minimizing the length of new/replacement culverts, the access road profile has been designed to minimize wetland impacts as much as practical while attempting to keep the profile as level as possible. The proposed alignment of the access road crosses several deep ravines, especially near its east end. Based on the necessary depth of the culverts and the design of the road embankment 1 on 2 side slopes, the lengths of the proposed culverts are the minimum necessary to accommodate safe road design.

The proposed replacement and new pipe arch culverts for the proposed Mine Access Road have been designed using aspects of Stream Simulation methodology. This methodology provides assurances that the proposed stream crossing structure will not cause long-term degradation of the stream, and that wildlife movements through the structures, either in the water or on the stream banks, are enhanced or facilitated. The goal is to provide a structure that minimizes stream habitat fragmentation.

Table 5-1. Comparison of Access Road Alternatives

Alternative	Length (Feet)	Wetland Impacts	Stream Crossings
North	13,540	4.27 ac.	14
Proposed	13,100	2.07 ac.	8
South	13,470	2.96 ac.	6

Copperwood's conclusion is that the proposed mine access road location is the most feasible and prudent road location and also the least environmentally damaging practicable alternative of the three mine access road alternatives. Wetland impacts for the proposed route are approximately 2.20 acres less than the north route alternative and 0.89 acres less than on the south route alternative. In addition, the proposed access road will also serve as the perimeter road on the south side of the proposed TDF.

6.0 UNDERGROUND MINE ENTRANCE ALTERNATIVES

The proposed underground mine entrance is located directly adjacent to the outdoor ore stockpile location (Figure 4-1). Four underground mine access alternatives were evaluated for Orvana by two mining consulting engineering firms, Marston and Thyssen Mining (TMCC). The mine access alternatives are:

- Shaft
- Underground Ramp
- Cut and Cover
- Box Cut (Proposed Alternative)

A shaft is a vertical entrance through the overburden and rock to reach the ore. A shaft requires hoisting equipment similar to elevators to move equipment, personnel, and mined ore from the underground workings. A box cut entrance is the term used for an inclined ramp structure to access the underground workings. The box cut requires excavation to remove overburden and construct a portal into the bedrock. The underground ramp and cut and cover entrance are forms of a box cut with enclosed ramps. These alternatives are explained in more detail in the following sections.

6.1 Shaft Mine Access Alternative

The shaft mine access alternative evaluated by TMCC included two 20-foot diameter shafts into the center of the ore body 650 feet deep. The location of the shaft would have provided a central location in the ore body for access to mining areas planned by Orvana when they commissioned the TMCC study. Each shaft would be equipped with a hoisting system, one shaft would be used for production and the other would be a service shaft. TMCC estimated the construction costs of a shaft mine access complex to be $72 million, with a completion time of approximately 1,079 days, both of which are relevant to the determination of whether this alternative is feasible or prudent.

The wetland impacts for the shaft mine access alternative were not determined because the exact location where the shaft would be located is not known. However, due to the fact that only about 14 acres would be needed for the shaft mine access alternative, wetland impacts would likely be minimal or avoided. The shaft alternative would however, require an access road for employees and mining supplies entering the mine. Since the mill for ore processing cannot be safely located above active mining areas as described in Section 7.0, ore produced in the mine and brought to the surface would then have to be trucked to the mill site using the access road or transported by other means such as an overland conveyor system, resulting in additional costs.

6.2 Underground Ramp Mine Access Alternative

To reduce the timeline and costs associated with the construction of the shaft mine access, Orvana asked TMCC to review an underground ramp mine access alternative, which would begin in the same location as the proposed box cut mine access (no figure was prepared for this alternative). TMCC proposed using a drill and blast operation to develop a 14-foot high by 16-foot wide slope at a 10 percent grade into the mine at a depth of 200 feet.

Total ramp length in this alternative was estimated to be 2,500 feet. A ventilation raise with a vertical conveyor was included in this estimate. TMCC estimated the construction costs of the ramp mine access alternative to be $31 million, with a completion time of approximately 742 days.

The wetland impacts for the underground ramp mine access alternative were not specifically determined because drawings for the ramp were not completed due to comparatively high construction costs; however, the wetland impacts would likely be similar to the proposed box cut alternative, although some of the impacts may only be temporary.

6.3 Cut-and-Cover Mine Access Alternative

To alleviate potential concerns about long-term stability of the box cut alternative (see below), Marston proposed a "cut-and-cover" mine access alternative in the same location as the proposed box cut alternative (no figure was prepared for this alternative). In the cut-and-cover alternative, the overburden material is removed, two tunnels and a shaft are constructed, and the excavated overburden material is replaced. The cut-and-cover alternative reduces the long-term requirement for a surface stockpile for the excavated overburden. The wetland impacts associated with this alternative are also likely to be similar to the box cut, although as with the Underground Ramp alternative, some of the impacts could likely be temporary.

To ventilate the mine via cut-and-cover alternative, Marston proposed building a vertical concrete shaft. Given its relatively short length, a 10-foot inside diameter shaft would result in no significant loss of ventilation pressure and would be built concurrently with the tunnels while the box cut is open. Marston estimated the construction costs of the cut-and-cover alternative to be $18.9 million, with a completion time of approximately 900 days.

6.4 Box Cut Mine Access Alternative (Proposed Alternative)

Due to the relatively shallow depth of overburden along the south and southeast edge of the mineralized zone, Marston evaluated the box cut mine access alternative. Marston's design intercepts the copper ore zone at a depth of approximately 116 feet. GMining's 2018 updated feasibility study design is to stay with the box cut mine access.

The configuration of the geologic formations within the Copperwood Project limits the location of a box cut mine access. The box cut needs to be located where the ore body is closest to the land surface to minimize the depth of the box cut. The ore body at Copperwood slopes downward from south to north at approximately 10 degrees; the ore body is closest to the ground surface in the area just east of the proposed mill site. For efficient project operations, it is extremely important that the box cut be located in proximity to the mill site due to the cost and related issues associated with the transport of ore from the mine by conveyor to the mill site.

The location chosen for the box cut is between Unnamed Creek and the West Branch Namebinag Creek, providing access to the center of the ore body while avoiding impacts to these water courses while avoiding the historic mine workings on the site (Figure 1-1).

An open box cut mine entrance was determined to be the best alternative design based on input from geotechnical experts from the region. The experts' observations indicated that the over-consolidated clay-rich glacial till comprising the overburden would remain stable at high angles of repose after excavation The box cut mine entrance is designed to provide a 150-foot by 200-foot staging area at the mine entrance with sufficient room for a water sump, pump, fan, and other mining equipment. One side of the box cut is an inclined access ramp that will be constructed through the glacial overburden at a 10% decline from the ground surface for about 1,000 feet before engaging the ore body in the floor of the box cut. This method allows efficient access to the ore body with a minimal amount of waste rock.

The volume of material that would need to be removed for the box cut is approximately 230,800 cubic yards of unconsolidated material and approximately 10,900 cubic yards of consolidated material (rock) for a total excavation of approximately 241,700 cubic yards. The box cut mine access excavation will impact approximately 0.13 acres of wetlands. GMining provided a preliminary cost estimate update for the box cut mine access alternative of $9.98 million (includes ancillary building and equipment costs from the 2012 estimate) with a completion time of 180 days.

6.5 Selected Alternative for Underground Mine Access

After the four mine access alternatives were evaluated (i.e. shaft, underground ramp, cut-and-cover, and box cut) Orvana determined the optimal method of mine entry is to access the ore horizon via the box cut mine entrance, using a portal system briefly described above. There will be a minimum of three portal entries; one portal entry for miners and equipment; one portal entry for a conveyor haulage system to bring ore to the surface; and a ventilation portal. A minimum of two portal entrances are required to comply with Mine Safety and Health Administration (MSHA) regulations to provide secondary means of personnel ingress and egress for safety reasons.

The ore body will be accessed at the bottom of the inclined ramp and box cut entrance, thereby substantially reducing the need to mine non-ore bearing rock. The box cut mine access alternative would impact approximately 0.13 acres of wetlands. Wetland impacts were not determined for the cut-and-cover and underground ramp mine access alternatives, both of which are in the same location as the box cut alternative and would be expected to have similar wetland impacts. The shaft mine access alternative would be expected to have minimal wetland impact due to the 14-acre land area needed for the shaft mine access and an undetermined impact for the necessary access road. Although the wetland impact is important, there are other significant factors that weigh against the shaft, underground ramp, and cut-and-cover mine access alternatives. The two main factors are 1.) cost; and 2.) time of development:

- Cost. The costs associated with each of the other three alternatives exceed the costs of the proposed box cut as follows: the shaft is 8 times more expensive; the underground ramp is 3.5 times more expensive; and the cut-and-cover alternative is 2 times more expensive. A comparison of those projected development costs is provided in Table 6-1.

Table 6-1. Comparison of Mine Access Alternative Development Costs

Alternative	Shaft	Ramp	Cut-and Cover	Box Cut
Evaluated By	Thyssen Mining	Thyssen Mining	Marston	GMining [1]
Estimates From	Thyssen Mining	Thyssen Mining	Local Contractors	Local Contractors
Mobilization/Demobilization	$13,600,636	$1,459,574	$60,000	$60,000
Electrical Equipment	$6,770,000	$1,935,000	$1,935,000	$1,935,000
Permanent Fan	$306,000	$306,000	$306,000	$306,000
Dry Building	$760,000	$760,000	$760,000	$760,000
Office Building	$590,000	$590,000	$590,000	$590,000
Lateral Development	$2,256,616	$4,378,731		
Conveyor Belt		$1,661,000	$304,000	$304,000
Shafts/Ramps/Tunnels	$32,699,098	12,144,693	$3,987,930	$2,361,000
Earthmoving			$7,161,000	$3,058,000
Daily Indirect Costs	8,457,297$	$5,077,687		
Contingency (10%-TMCC; 25% Marston, 10% GMining)	$6,543,965	$2,831,268	$3,775,983	$907,000
Total Development Cost	**$71,983,612**	**$31,143,953**	**$18,879,913**	**$9,977,000**

[1] Preliminary GMining cost estimate for 2018 feasibility study update.

- Time of development. The time of development of the mine access alternatives is another consideration in determining whether the alternative is feasible or prudent. Table 6-2 compares the time of development for the four mine access alternatives.

Table 6-2. Comparison of Mine Access Alternative Impacts and Time of Development

Alternative	Wetland Impact	Time of Development (days)
Shaft	0 ac.	1,079 (36 months)
Underground ramp	0.08 ac.*	742 (25 months)
Cut-and-Cover Mine access	0.08 ac.*	900 (30 months)
Box cut	0.08 ac.	180 (6 months)

*Assumed to be the same as the box cut.

Although the box cut has 0.13 acres more wetland impact than the shaft alternative and approximately the same impact as the and cut-and-cover and underground ramp alternatives, the box cut is the most feasible and prudent alternative for mine access due to the substantial costs and time of development of the other alternatives, as both of these factors are critical to the financial sustainability of the Copperwood Project.

7.0 MILL SITE AND OUTDOOR ORE STOCKPILE DEVELOPMENT

The proposed “mill site and outdoor ore stockpile” referred to in this application for permit are the ore storage and processing site for the mine. The overall operation is comprised of the following facilities (Figure 4-1):

- Gatehouse/Plant Security
- Mine Office/Change Room
- Maintenance Shops
- Warehouse/Offices
- Outdoor Ore Stockpile/Reclaim System/Ore Bins
- Mill Building (includes grinding, flotation, MCC room, reagent handling, offices, lab)
- Copper Concentrate Handling and Truck Wash Building
- Water Treatment Plant
- Natural Gas Power Generators

The mill and ore stockpile are proposed in a location as near as possible to the mine entrance to minimize the distance that ore must be conveyed as well as to concentrate the development of the project surface facilities within as limited area as possible to minimize impacts. The mill will be constructed over areas underlain by a rock formation known as Copper Harbor Conglomerate, which is not a part of the copper ore body. The mill site cannot be located over the Nonesuch Shale, which contains the copper ore, because of mine safety concerns about having underground mine workings beneath the mill site due to the weight of the facilities, vibrations, and other safety-related factors. The ore body is closest to the ground surface in the vicinity of the mill site, exacerbating the safety concerns if the mill were to be located over the Nonesuch Shale in that area. Ore will be transported out of the mine by conveyor and either sent directly to the mill through a pair of day-bins or transferred to an outdoor stockpile if the mine production rate is higher than mill capacity. Locating the mill as close as possible to the mine entrance is important for mining efficiency and also for consolidation of mine facilities on the landscape.

GMining’s review of Orvana’s permitted plan for ore storage and processing identified both operational and environmental concerns that could be addressed by establishing an outdoor ore stockpile and ore reclaim system at the previous mill site and relocating the mill to the west. During project construction and development, the previous plan was to transport ore a distance of more than one mile for temporary storage in a prepared area of the Tailings Disposal Facility followed by reclaiming and transport back to the mill when construction was completed. Ore storage capacity after mine/mill startup in the previous plan was limited to one day (24 hours) of mine production in day-bins. If mine production exceeded mill capacity to process the ore, it would have to be left at the production sites in the mine, as there was no surge capacity planned for ore storage. This situation would eventually have halted ore production in the mine until newly blasted ore would have been removed and transported to the mill.

An additional concern with the previous mill site facility layout was that, except for traffic only stopping at the administration building or lab, all vehicles into the site had to pass underneath an overhead conveyor transporting ore to the mill and near the top of the box cut entrance ramp. This situation had potential safety issues with delivery vehicles and employees passing under the ore conveyor and near the mine access ramp. An environmental management issue would also result due to this being an active mining area with a contact area designation for runoff water management and vehicle washing to prevent reactive materials from entering the environment.

The updated location and layout for the mill site will allow for all employee traffic, most deliveries and traffic to the on-site power plant and water treatment plant to be in areas designated as non-contact for runoff and vehicle management, which is a much better design.

The mill site includes a process water treatment plant ("WWTP"). Conceptual design of the facility was completed by Golder and Associates ("Golder") following the sequence listed below:

1. Identification and analysis of the sources and flow quantities to determine an influent design basis for required treatment capacity and associated water quality;

2. Identification and analysis of effluent and reclaimed water quality targets to identify Contaminants of Potential Concern ("COPC");

3. Evaluation and development of technologies appropriate for treating the COPC and resulting in a candidate treatment alternative with associated order-of-magnitude cost estimate.

Based on the design basis and the end-of-pipe treatment goals, a conceptual treatment process was developed. The major unit processes include the following:

- Microfiltration
- Adsorption on granular activated carbon ("GAC")
- Membrane filtration by reverse osmosis
- Ion exchange (mercury polish)
- pH adjustment
- Evaporator
- Crystallizer
- Belt press

The WWTP is projected to meet end-of-pipe treatment goals established for the facility NPDES permit and will be a state-of-the-art facility.

Due to the mine site development, the WWTP cost (estimated to be approximately $24 million) and the resultant capital cost timing, the WWTP will not be constructed until the third year of mine development, with planned start-up in year four. Until the WWTP is operational, contact water will be stored in Phase 1 of the TDF. The present water balance for the Copperwood Project does not indicate a need to discharge excess water until about year five of the mine life, thus the WWTP is not necessary sooner.

Water from mine dewatering, tailings decant water, the ore stockpile and mill site runoff will be stored in the TDF and used in addition to fresh water from the proposed Lake Superior intake as mill process water prior to the site water balance gaining enough volume to necessitate treatment in the WWTP and discharge. The TDF decant barge system will collect water that will either be treated in the WWTP (0 – 275 gpm) recycled through the process mill (0 – 1,089 gpm) treated in the WWTP (0 – 275 gpm). Excess water that is not reclaimed for the process mill will be discharged from the WWTP through the outfall to the West Branch Namebinag Creek, at an estimated maximum discharge rate of 275 gpm. The WWTP will include sedimentation of particulates, pH adjustment, microfiltration, adsorption on granular activated carbon, and membrane filtration by reverse osmosis and ion exchange. The MDEQ NPDES permit specifies effluent limits to be met by the discharge from the plant to the West Branch Namebinag Creek.

The previous project plan for electric power supply was delivery by a regulated utility company to the mine site via overhead transmission line from a substation in Ironwood to a substation at the mine site; a distance of approximately 25 miles. The transmission line was being planned to follow existing right-of-ways as much as possible to reach the mine site. The previous owner, Orvana, reviewed options for on-site power generation, but dismissed this option due to the estimated operational mine load requirement of as much as 20 megawatts.

Current tradeoff studies for power supply as part of the project update considerations have concluded that on-site power generation with natural gas fueled engine-generator sets has become a feasible alternative as the economics of natural gas fuel usage for the mine operations are more favorable now than in 2012. In addition, the estimated 54-month scoping level time frame for engineering, permitting and installation of an overhead transmission line to the mine site is not compatible with the planned project development schedule. The natural gas fuel supply plan is for a 4-inch diameter pipeline from a transmission line connection within the CR 519 roadway limits, a distance of approximately 12 miles. With the pipeline proposed to be installed within the CR 519 corridor, wetland impacts will be minimal, if any, and temporary.

While taking up another land area on-site that was not previously proposed, this power generation approach for the proposed action presents a better alternative from both an environmental and timely project completion perspective.

The milling, processing, and concentrating process that will occur in the mill will result in a concentrate containing approximately 25% copper and 9% moisture content, which is the final on-site product. This final concentrate will be stored in an enclosed building that will have capacity for seven days of production. Trucks will be loaded and washed in the concentrate storage building, and then will transport the concentrate to an off-site facility for further processing (i.e. smelting). No smelting will be done at the site.

The construction of Orvana's planned mill site permitted in 2013 would have resulted in impacts to 2.59 acres of wetlands. The updated project plan for an outdoor ore stockpile and mill site will have a combined wetland impact of 2.48 acres which is a slight reduction. The proposed mill site location will require a clear span bridge and overhead conveyor crossing of Unnamed Creek, an ephemeral stream that flows between the proposed mill site and ore stockpile, as can be seen in Figure 4-1.

7.1 Alternative Mill Site Locations

No other copper ore processing facilities currently operate in the local area. The White Pine Mine, approximately 20 miles east of the Copperwood site, was closed in the mid-1990s. No ore processing capabilities remain at this facility. The next closest copper ore processing facility is at the Eagle Project Humboldt Mill near Champion, Michigan, which is about 125 miles away. The Humboldt Mill is not available for processing ore from the Copperwood Project.

For reasons described in Section 1.0 Introduction, Highland evaluated transporting ore to White Pine and constructing a new ore processing plant there but ultimately made the decision not to pursue this alternative. Before suspension of the combined Copperwood-White Pine PFS, a draft ore transport feasibility report was completed by MHF Services ("MHF") in March 2015 (Appendix B) for truck and rail options from Copperwood to White Pine. In addition, a scoping level design study for an ore slurry pipeline from Copperwood to White Pine was completed by a Michigan Tech University ("MTU") design class in May 2015 (Appendix C). Further, another MTU design class completed a scoping level study report for Highland's proposed mine access method and available areas to re-start mining at White Pine (Appendix D). The location of a new ore processing facility at White Pine was not yet determined while the PFS was being prepared, but it would have been within the area boundary noted on the Conceptual Portal and Surface Facilities Footprint Encumbered Areas Map and Notes (Appendix E). Appendix E also includes an overall White Pine map showing only delineated wetlands.

Ore transport options considered for the Copperwood-White Pine North PFS are listed below with a brief discussion of their advantages and disadvantages. For purposes of evaluating these options, the MHF report estimated Copperwood ore production rate of 2,350,000 tonnes per year. All options considered by MHF included initial shipment from the Copperwood mine site in trucks by a 3rd party contract hauler. Although not using the exact same ore production rate, the MTU design study was based on a similar amount (2,192,000 tonnes per year).

<u>7.1.1</u> Truck only from Copperwood to White Pine.

Estimated cost = $12.50 per tonne with 48-ton loads. Annual cost = $29,400,000 per year for an average of 135 to 140 truckload round trips per day. No pre-production capital expense, but highest operating cost. This alternative would create a major increase in heavy truck traffic along the route to White Pine, especially on CR 519.

<u>7.1.2</u> Truck from Copperwood to a new rail transload facility in Thomaston (approximately 10 miles) then rail to White Pine.

Capital cost estimate for the transload facility = $3,700,000 for owner built and financed facility versus $4,700,000 for MHF built and financed facility. Overall operating cost estimate = $7.64 per tonne for an annual cost of $18,000,000, which is in the middle range of the truck/rail operating cost estimates. Rail would be in 100-tonne loads by a short-line rail operator on existing leased track from Thomaston to White Pine. This option would still have 135 to 140 truckload round trips per day on CR 519.

7.1.3 Truck from Copperwood to a new rail transload facility at an existing gravel pit along CR 519 (approximately 2.5 miles)

A new transload facility would be required at the gravel pit and a new rail line constructed to Thomaston for connection with the existing rail line to White Pine. This is the proposed option of the MHF study report. Capital cost estimate for the transload facility and new rail line to Thomaston = $18,090,000 for owner built and financed versus $23,100,000 for MHF built and financed. Overall operating cost estimate = $3.25 per tonne for an annual cost of $7,600,000. This option only has trucks on CR 519 for a distance of approximately 0.75 miles with the possibility of eliminating this traffic if a private road were constructed utilizing large off-road trucks and a single crossing of the county road. The main disadvantage of this option is that a significant amount of wetland indicators were observed during a preliminary ground survey of the former rail grade from Thomaston to the gravel pit. The exact area of wetland impact from new rail line construction for this ore transport option was not determined as part of the MHF report.

7.1.4 Ore slurry pipeline to transport ore from Copperwood to White Pine

The MTU design study for this option had a very high estimated capital cost for construction of $46,100,000, with the lowest annual operating cost for ore transportation options of $1,540,000. The total pipeline length would be approximately 31 miles, with crossings of two relatively large rivers (the Presque Isle and Big Iron), and construction of five pump stations with necessary access roads and power supply for those pump stations. Wetland impacts were not part of the MTU study work scope but would likely be significant. There would be multiple landowner agreements required, including approval of the Ottawa National Forest.

It should be noted that both the stand-alone Copperwood Project and the combined Copperwood-White Pine Project would require construction of a completely new ore processing facility; however, in the early years, the combined project would have the extra economic burden of the capital and operating costs for transporting the Copperwood ore to White Pine. The MTU scoping level mine access report for Highland's preferred mine access option at White Pine, where the majority of the remaining ore is at a depth of approximately 2,450 feet, was a combined box cut and 2.1-mile underground ramp with an estimated construction cost of $46,100,000 and time frame of 2 to 4 years. Wetland impacts and selection of a process plant location were outside the scope of the MTU design study. Review of the wetland delineations shown on the maps included in Appendix E indicate that wetland impacts of the preferred box cut location would likely be similar to the Copperwood box cut mine access alternative. There are also areas available at White Pine within the footprint boundary shown on the encumbered area map that would have minimal wetland impact for construction of a new ore processing facility.

While the costs to construct and operate a new ore processing facility would be similar regardless of its location, the cost estimates for tailings disposal are much greater at White Pine compared to the proposed stand-alone Copperwood TDF. Details of tailings disposal alternatives at White Pine are discussed in Section 8.0 with the most feasible TDF alternative at White Pine (Section 8.7 – North #2 Basin redevelopment and closure) estimated to cost over $100,000,000 more than the preferred TDF alternative at the stand-alone Copperwood project (Section 8.4 – Alternative 4B).

Direct shipping of the copper ore to another state or country for processing into a concentrate would not be prudent due to the relatively low copper content in the ore and costs of transporting an average of 6,600 tons of raw ore per day during full production, which totals 2.2 million tons of ore per year, based upon 8,000 hours of production. The economics of such an off-site ore-processing alternative would render the Copperwood Project economically unviable and is therefore not prudent. As a result, an on-site ore processing facility is the feasible and prudent alternative for the Copperwood Project.

7.2 Summary of the Proposed Mill Site Alternative

For the reasons noted above Highland has concluded the feasible and prudent alternative for processing the Copperwood ore is to locate the mill at the Copperwood mine site. The selected on-site alternative provides the most favorable project economics with the mill site in close proximity to the mine entrance and TDF, avoids sites overlying the ore body and eventual mine workings, accommodates the need for ore storage capacity during development and operation of the project and enhances the safety and environmental management issues with the Orvana plans noted above, Copperwood has determined that the proposed ore stockpile and mill site configuration is the most feasible and prudent alternative.

8.0 TAILINGS DISPOSAL FACILITY

The waste product of the milling process is commonly called tailings. The host mineral for the copper at the Copperwood Project is chalcocite. The chalcocite within the ore body is in very fine grains (5 to 50 microns diameter) within the lower portions of the Nonesuch Shale. To liberate the chalcocite from the enclosing rock, the ore processing method is as follows:

1. The processing facility will receive run-of-mine ore that will be minus 8-inch material. The current flow diagram of the processing facility utilizes a semi-autogenous grinding (SAG) mill to process the ore to a P80 of 150-micron size (i.e. 80% by weight of the ore is ground to 150 microns or smaller).

2. The ore from the SAG mill passes through one of two ball mills operating in closed circuit with a cyclone bank to grind the material to a P80 of 45-micron size.

3. After an initial rougher flotation step, the ore passes through a regrind mill to achieve a size of P80 of 20 microns, which is considered the optimal size to separate the chalcocite from the host rock.

4. The ore slurry is processed through a series of flotation cleaner and recleaner cells to separate the chalcocite from the ground rock. The final sellable product of the processing facility is a concentrate of chalcocite, containing approximately 25% copper. The waste rock is slurry comprised of 79% process water and 21% solids with a P80 of 20 microns. The waste rock slurry is called tailings, which will be pumped to the TDF.

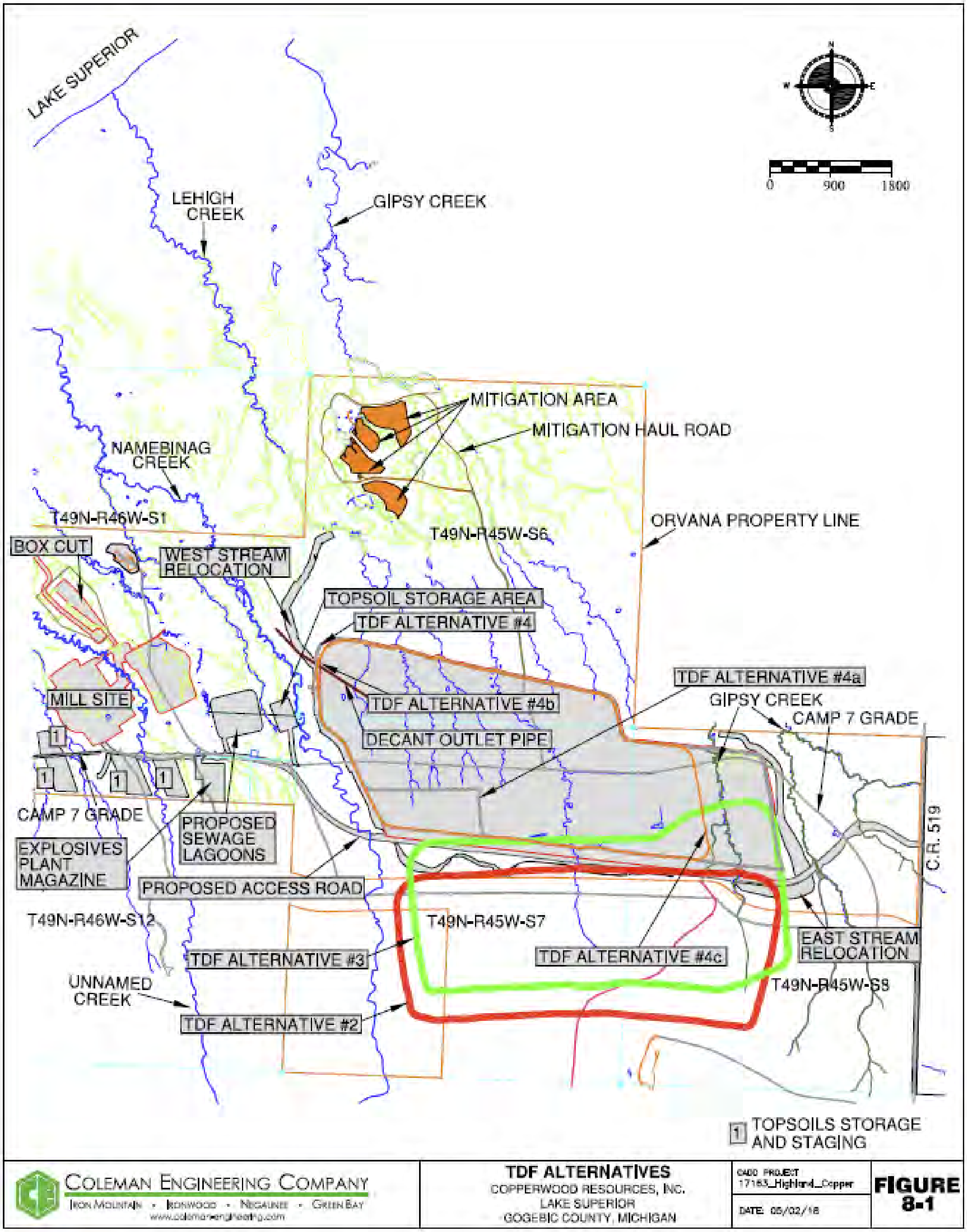
5. The ore concentrate is dried to approximately nine percent moisture for storage or shipping.

Based on current design criteria, the waste tailings will require a large area to enable the water to separate from the solids, a process termed “consolidation”. Due to the fine nature of these tailings, the present consolidation model indicates that the solids will stay in suspension for a long period of time and, as a result, will need a large area of deposition to allow for this separation. The separation of water and solids will be accelerated as the tailings pond increases in depth. The proposed project involves construction of a TDF to dispose of 100% of the approximately 31.9 million tons (dry weight) of tailings that will be generated during the life of the mine.

The 31.9 million tons of tailings to be produced from the milling operation will require a volume of 29.6 million cubic yards of disposal volume. The following alternatives for tailings disposal are evaluated in this Alternatives Analysis (Figure 8-1):

- Alternative 1: Underground tailings disposal and on-site TDF
- Alternative 2: South on-site TDF
- Alternative 3: Center on-site TDF
- Alternative 4: North on-site TDF
- Alternative 4A: Reduced-footprint version of Alternative 4
- Alternative 4B: West berm moved east

- Alternative 4C: East berm moved west
- Alternative 5: Off-site TDF
- Alternative 6: On-site TDF over mine workings
- Alternative 7: White Pine


LAKE SUPERIOR
LEHIGH CREEK
GIPSY CREEK
0 900 1800
MITIGATION AREA
MITIGATION HAUL ROAD
NAMEBINAG CREEK
T49N-R46W-S1
T49N-R45W-S6
ORVANA PROPERTY LINE
BOX CUT
WEST STREAM RELOCATION
TOPSOIL STORAGE AREA
TDF ALTERNATIVE #4
TDF ALTERNATIVE #4a
MILL SITE
TDF ALTERNATIVE #4b
GIPSY CREEK
CAMP 7 GRADE
DECANT OUTLET PIPE
CAMP 7 GRADE
PROPOSED SEWAGE LAGOONS
EXPLOSIVES PLANT MAGAZINE
C.R. 519
PROPOSED ACCESS ROAD
T49N-R46W-S12
T49N-R45W-S7
EAST STREAM RELOCATION
TDF ALTERNATIVE #3
TDF ALTERNATIVE #4c
T49N-R45W-S8
UNNAMED CREEK
TDF ALTERNATIVE #2
TOPSOILS STORAGE AND STAGING
COLEMAN ENGINEERING COMPANY
IRON MOUNTAIN • IRONWOOD • NEGAUNEE • GREEN BAY
TDF ALTERNATIVES
COPPERWOOD RESOURCES, INC.
LAKE SUPERIOR
GOGEBIC COUNTY, MICHIGAN
DATE: 05/02/18
FIGURE 8-1

The on-site TDF alternative evaluation conducted by Orvana and previously permitted by MDEQ generally included the following considerations with an explanation for each below:

- Mine safety;
- Topography, low permeability soils, and depth to bedrock;
- Optimal geotechnical and hydrogeological conditions;
- Avoidance of sites over underground mining area;
- Wetland and stream avoidance;
- Tailings volume for disposal, dike size required, and construction costs;
- Avoiding key ecological receptors, visual impacts, and the North Country Trail;
- Proximity to mine/mill complex;
- Proximity to Lake Superior.

<u>Mine safety:</u> This consideration is primarily for the underground tailings disposal, which is fraught with safety concerns for miners. However, other safety concerns with the TDF were taken into consideration, including embankment stability, height of berms in proximity to other mining activity, etc.

<u>Topography, low permeability soils, and depth to bedrock:</u> These three factors are especially critical for siting the TDF. Severe topographic slope would result in the construction of larger dikes on the downslope side of the TDF if it were to be located further south than proposed. Low permeability soils are necessary to create enhanced isolation from groundwater and minimize seepage from the TDF. Depth to bedrock is important because the base of the TDF must have a minimum layer of low permeability soil and if bedrock is too close to the ground surface, siting of the TDF is not feasible.

<u>Optimal geotechnical and hydrogeological conditions:</u> These two conditions relate to soil types and the presence of groundwater; lenses of permeable soils in the strata are not desirable for siting the TDF because of seepage and groundwater issues.

<u>Avoidance of sites over underground mining area:</u> Having the sheer weight of millions of tons of water and tailings over active mining areas is a safety concern for underground miners and must be avoided.

<u>Wetland and stream impact avoidance and minimization:</u> To comply with MDEQ and EPA regulations, impacts to wetlands and streams must be avoided and minimized to the greatest practicable extent. Avoidance of the more important streams on the site (i.e. Namebinag Creek and Gipsy Creek) was therefore an important factor in the siting of the TDF. Given the necessary large size of the TDF and the other constraints for siting the TDF on this site, complete wetland avoidance is impossible but was taken into consideration to the extent practicable.

<u>Tailings volume for disposal, dike size required, and construction costs:</u> The first two factors are derived from the amount of ore to be mined (i.e. TDF volume), when considered with the topography, depth to bedrock, and soil conditions determine the size of the dikes required for the TDF. All of these factors contribute to the cost of construction, which is a critical factor in the mine feasibility.

Avoiding key ecological receptors, visual impacts, and the North Country Trail: Some of the key ecological receptors referenced here are: redside dace, which is a State endangered fish species and the lower stream reaches associated with them; Porcupine Mountains Wilderness State Park; and the Presque Isle River and Black River watersheds. Visual impacts from vantage points in the State Park and other locations have been evaluated, as described later in this document. The North Country Trail is less of a concern because relocation of the trail could be accomplished, if necessary, without impairing the use or scenic values of that segment of the trail.

Proximity to mine/mill complex: The proximity to the mill site is important to reduce pipeline and pumping costs and to minimize the exposure of the landscape to a pipeline accident that could cause release of tailings to the environment.

Proximity to Lake Superior: Locating the TDF away from Lake Superior is desirable to avoid visual impacts from the lake and to minimize the release of raw tailings into the lake if a pipeline incident or TDF berm breach were to occur.

An economic comparison of each of the on-site TDF alternatives using the completed 2012 BFS report for the Copperwood Project is provided in summary fashion in Table 8.4. Completion of the 2018 feasibility study update is expected in late May at which time updated IRR's can be determined.

8.1 Alternative 1: Underground Tailings Disposal

Alternative 1 is underground disposal of tailings produced by the Copperwood Project. This section of the Alternatives Analysis summarizes a Golder Associates September 14, 2012 Memorandum to Orvana that evaluated the feasibility and practicality of backfilling the Copperwood mine with tailings while the mine is active. This Golder technical document/memorandum is included in Appendix G for reference. The purpose of the Golder evaluation was to assess opportunities to utilize methodologies to backfill the mine with tailings with the goal of thereby reducing the currently proposed 348 -acre total TDF footprint area, and consequently reducing impacts to wetlands and streams.

Over its mine life, the Copperwood Project is expected to produce approximately 31.9 million tons of dry waste tailings. Production of this tonnage will result in a total volume of 29.6 million cubic yards of waste material that will need disposal. At a maximum, the space available within the mine where such wastes could be disposed of safely and without impairing mining operations is approximately eight million cubic yards. Some of the potentially available space in the mine will not be possible to fill due to irregularities in the mine surface. In addition, bulkhead dams would need to be constructed to contain mine backfill. Therefore, under any alternative, some form of a large TDF would be required. The estimated potential reductions in the size of the TDF footprint that are reported in the Golder Memorandum are approximate, given that the tailings basin will not have a uniform depth, and that an average tailings disposal efficiency is assumed.

In order to address the potential to utilize the mine for backfill of waste tailings, three backfill alternatives were evaluated by Golder:

- Raw Tailings

- Hydraulic Sand
- Paste

Backfilling the mine cavity could occur after several years of mining, once sufficient cavity space has been created. For disposal of raw tailings and hydraulic sand backfill, bulkhead dams would be required to isolate the areas of backfill from the active mining areas. However, mine backfill requiring bulkheads brings with it safety risks in the form of increased equipment and traffic in the mine environment as well as the risk of backfill liquefaction resulting from blasting or seismic activities. The unacceptable and dangerous occurrence of liquefaction has been well documented and unfortunately experienced worldwide in the mining industry. When backfill materials liquefy and flow uncontrollably into the mine workings, active work areas can be rapidly inundated, putting mine workers in severe peril.

8.1.1 Raw Tailings Mine Backfill

In the alternative where raw tailings are put into the mine, dewatering of the tailings would be required. After dewatering, raw tailings could potentially be pumped directly into the mine cavity, resulting in an approximately 26 percent (84-acre) reduction in the TDF footprint. However, in this alternative, safety of the mine workers would be severely compromised and therefore this alternative is not feasible. In addition, costs associated with this alternative include construction of bulkhead dams, added water treatment plant costs, and the cost of pumping the tailings to the mine, minus the cost savings of constructing a smaller TDF. In total, the net increase in cost for this alternative is estimated by Golder to be approximately $26 million. While this additional cost alone would likely render this alternative to be not prudent, the overriding consideration of the raw tailings alternative is human safety.

8.1.2 Hydraulic Sand Mine Backfill

Using a different methodology, raw tailings could be mechanically separated with cyclones into both sand-sized particles and very fine particles (slimes). The separated sand-sized particles could eventually be piped into the mine as backfill without further treatment and stored behind bulkhead dams. The very fine particles/slimes however, could not be used as mine backfill and would have to be placed in the TDF. The storage of these slimes would require a larger diameter thickener as well as increased retention times before the slimes could be pumped to the TDF. Due to the lower density of these slimes, only an estimated two percent overall reduction in necessary TDF volume would be achieved in this alternative. Additional costs associated with hydraulic sand mine backfill include construction of bulkhead dams, use of required separation and thickening equipment, added water treatment plant costs, pumping to the mine, and costs associated with additional materials and, in the long-term, techniques needed to evacuate water from the slimes during TDF closure. In spite of the fact there would be little to no reduction in the required TDF footprint for the hydraulic sand mine backfill alternative, additional costs are estimated by Golder to be approximately $93 million.

8.1.3 Paste Mine Backfill

In this methodology, raw tailings would be augmented with a cement binder (as compared to simply a thickener) to create a non-liquefying structural paste backfill which does not require bulkhead containment dams. In the mining industry where applications utilize paste backfill, the target product for paste backfill is to reach a non-liquefying state at seven days of curing in order to allow production blasting to safely continue. Tests performed on representative tailings from the Copperwood mine with 100 pounds of binder per ton of tailings (five percent dry weight) indicated there was insufficient strength in the cement-bound tailings to avoid the potential for liquefaction of those tailings. Additional testing would be required to determine the increased rate of binder application beyond the 100 pounds of binder per ton of tailings needed to provide paste backfill that would cure in a time period that would be sufficient to avoid liquefaction. If additional cement binder could be added so as to make paste backfill a feasible alternative, it could result in a 40 percent (129 acres) reduction in the TDF footprint. However, the costs associated with this alternative are substantial. Those costs include construction and operation of a paste plant for mixing the tailings with the binder, purchase of the binder, added water treatment plant costs, and placing the paste in the mine.

Using the above-referenced five percent binder application rate (which is known by testing to be insufficient), and then including the cost savings of constructing a smaller TDF, the net increase in cost for this alternative is estimated by Golder to be, at a minimum, approximately $63 million. Such excessive costs cannot be supported by Orvana while still maintaining a financially viable project.

Golder also evaluated the use of aggregate as part of the backfill, as backfill is sometimes required for structural purposes in order to allow mining operations for such things as pillar recovery and maximization of resource extraction. When backfilling, aggregates may be utilized to achieve structural integrity. In that scenario, aggregate addition rates are usually in the range of a 50% aggregate and 50% tailings mixture. Although the Unconfined Compressive Strength of the backfill may be improved by the addition of aggregate in place of tailings, the quantity of tailings being able to be deposited underground are accordingly reduced due to the volume of the aggregate. In this case, since the primary purpose of backfilling would be to maximize the amount of disposal of tailings from surface to underground, the economic benefit of reducing the quantity of tailings reporting underground by 50% due to the aggregate addition is counterproductive. Adding aggregate to the backfill would not only reduce the amount of tailings disposed of underground from the previously documented and submitted un-economical backfill models, it would add additional costs to the various backfilling options. A TDF, which would be required in every backfilling option considered, would be even larger (when adding aggregate) than those previously modeled and described, since aggregate would replace tailings volume. Therefore, the overall costs using aggregate backfill would be even more as illustrated in Table 8-1.

Table 8-1. Reduction of TDF size with 50% Aggregate Added to Backfill

Backfill Alternative	Original TDF Size Reduction	TDF Size Reduction with 50% Aggregate Addition
Raw Tailings	26% / 84 acres	13% / 42 acres
Hydraulic Sand	0-2% / 0-6 acres	0-1% / 0-5 acres
Paste Backfill	40% / 129 acres	20% / 64.5 acres

Based on these considerations, the alternative of adding aggregate to the backfill is neither feasible nor prudent.

8.1.4 Conclusions for Alternative 1

Each of the three backfill alternatives evaluated are considered feasible in that they are technically possible, though not prudent from either safety or cost perspectives. The raw tailings backfill alternative is simply not safe. The hydraulic sand backfill alternative fails in the cost/benefit evaluation. The paste backfill alternative provides a substantial reduction in TDF footprint and related natural resource impacts; however, the excessive costs for that alternative were considered by Orvana to be potentially prohibitive to the economic viability of the Copperwood Project and therefore were also not considered prudent or practicable.

The important attributes of each of the mine backfill alternatives are summarized in Table 8-1a.

Table 8-1a. Summary of Mine Backfill Alternative Attributes

Backfill Alternative	TDF Size Reduction	Estimated Cost	Decision Factor
Raw Tailings	26% / 84 acres	$26 million	Safety
Hydraulic Sand	0-2% / 0-6 acres	$93 million	Costs/Benefits
Paste	40% / 129 acres	$63 million (min.)	Costs

8.2 Alternative 2: South On-Site TDF

Alternative 2 is the southernmost on-site alternative and would have a footprint of 267 acres. It would impact approximately 6.03 acres of wetland (not including any low permeability soil borrow area wetland impacts) and 3,706 linear feet of streams (Figure 8-3). Siting of Alternative 2 is constrained by Namebinag Creek on the west and the East Branch Gipsy Creek on the east. The maximum dike height is 181 feet (an elevation of 1120 Mean Sea Level ("MSL")). One critical attribute of Alternative 2 is that the construction of the TDF would have a negative low permeability soil balance of 9.9 million cubic yards, which would have to be obtained from another on-site or off-site location. The reason for the negative soil balance is the presence of bedrock near the surface in portions of this alternate TDF location. This bedrock limits the depth of excavation within the TDF and requires the hauling-in of additional low permeability soil from elsewhere to complete the TDF construction.

The south Alternative TDF would directly impact the North Country Trail and would have a relatively high north berm due to the siting of the TDF on a relatively steep slope that requires a high berm on the north side of the TDF (Figure 8-3). The footprint of Alternative 2 is smaller than the other on-site revised alternatives, primarily due to the location on the steep slope and the higher berms that would be required.

Alternative 2 would require the acquisition of property from the Ottawa National Forest and relocation of the North Country Trail, which Orvana concluded in 2012 is not feasible or prudent. As explained in the mine access road analysis in Section 5.0, the process to acquire the land from the Ottawa National Forest and relocate the North Country Trail would take a substantial amount of time and the result would have been very uncertain; Copperwood agrees with Orvana's conclusion that engaging in that process with the Forest Service for this purpose would not be prudent for this project.

Soil is required to construct the perimeter berms and create a configuration that would be used to contain the tailings. To construct the TDF, large amounts of soil would need to be brought to the location of the TDF from elsewhere to form the berms with increased costs to purchase and transport soil to the site. Further, a very large borrow area would be required to generate the volume of soil needed. The embankment size would not be reduced since the required TDF storage volume would be unchanged.

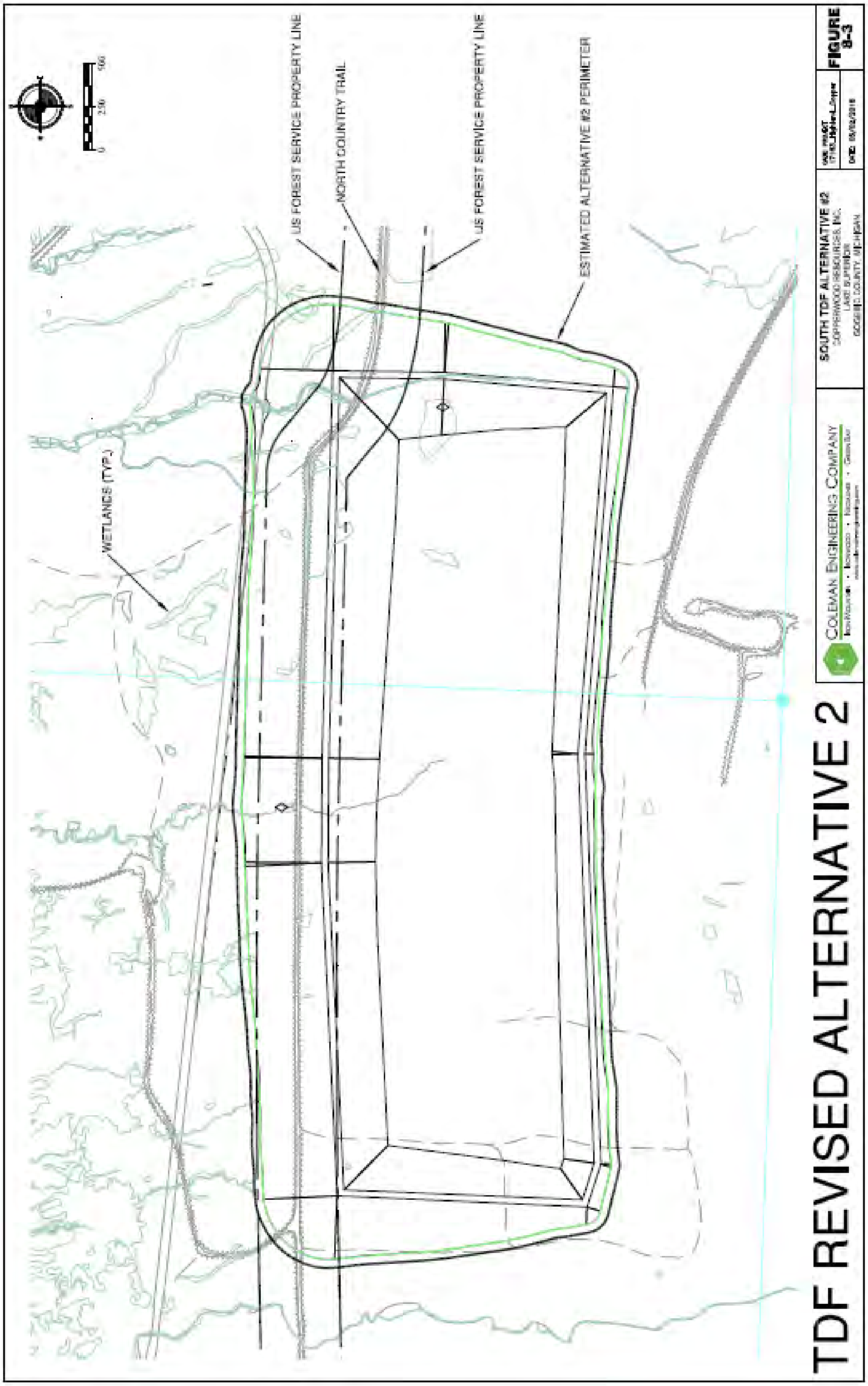
Orvana has determined that Alternative 2 is not feasible or prudent due to the following considerations:

- The high dike height (181 feet) due to the TDF being located on a steep slope is not prudent from the basis of visibility, safety, and cost;

- The TDF is located furthest from the mill, which will require longer tailings pipelines, which have cost, operational, and safety concerns;

- The procurement of an estimated 9.9 million cubic yards of low permeability soil from another location for the construction of the TDF dikes would very likely impact a substantial acreage of additional wetlands; to transport the low permeability soil to the TDF site would add considerably to the cost;

- Acquisition of property from the Ottawa National Forest and relocation of the North Country Trail is not prudent due to the extended public review process involved and the uncertainty of the outcome of this process.


TDF REVISED ALTERNATIVE 2
US FOREST SERVICE PROPERTY LINE
NORTH COUNTRY TRAIL
US FOREST SERVICE PROPERTY LINE
ESTIMATED ALTERNATIVE #2 PERIMETER
WETLANDS (TYP.)
COLEMAN ENGINEERING COMPANY
SOUTH TDF ALTERNATIVE #2
COPPERWOOD RESOURCES, INC.
LAKE SUPERIOR
GOGEBIC COUNTY, MICHIGAN
FIGURE 8-3

8.3 Alternative 3: Center On-Site TDF

Alternative 3 is the center on-site alternative and would have a footprint of 284 acres and would impact approximately 11.98 acres of wetland and 5,108 linear feet of stream (Figure 8-4). Siting of Alternative 3 is also constrained by Namebinag Creek on the west and the East Branch Gipsy Creek on the east. The maximum dike height is 175 feet (an elevation of 1080 MSL). The construction of Alternative 3 TDF would have a negative low permeability soil balance of 9.65 million cubic yards, which would have to be obtained from another on-site or off-site location. As with Alternative 2, the reason for the negative soil balance is the presence of bedrock near the surface in portions of the TDF, which limits the depth of excavation within the TDF and requires additional low permeability soil from elsewhere to complete the TDF construction.

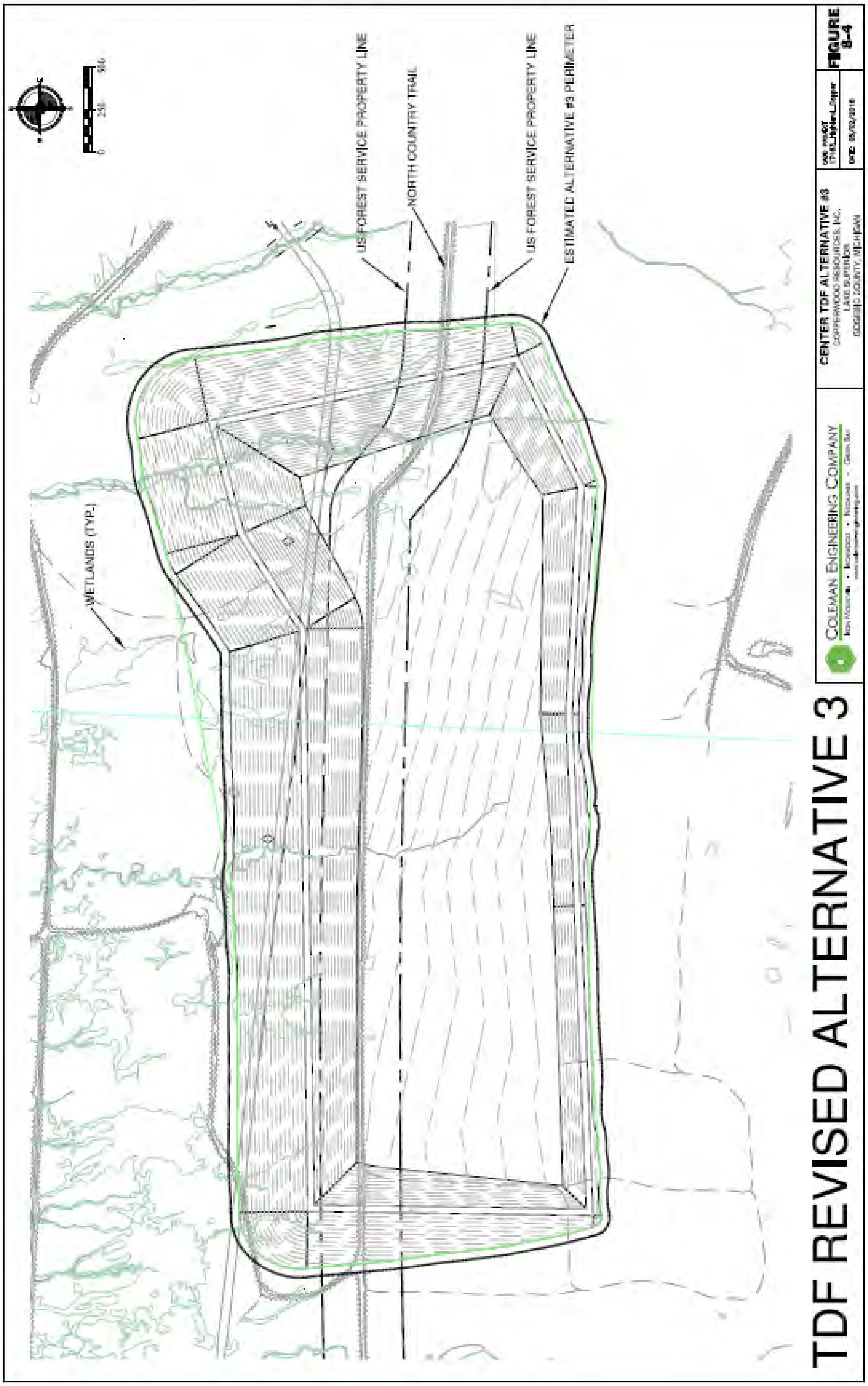
Alternative 3 would require the acquisition of property from the Ottawa National Forest and relocation of the North Country Trail, which Orvana concluded in 2012 is not feasible or prudent. As explained in the mine access road analysis in Section 5.0, the process to acquire the land from the Ottawa National Forest and relocate the North Country Trail would take a substantial amount of time and the result is very uncertain; Copperwood agrees with Orvana's conclusion that engaging in that process is not prudent for this project.

Orvana has determined that Alternative 3 is not feasible or prudent due to the following considerations:

- The high dike height (175 feet) due to the TDF being located on a steep slope is not prudent from the basis of visibility, safety, and cost;

- The procurement of an estimated 9.65 million cubic yards of low permeability soil from another location for the construction of the TDF dikes would very likely impact substantial acreage of additional wetlands and to transport the low permeability soil to the TDF site would add considerably to the cost;

- Acquisition of property from the Ottawa National Forest and relocation of the North Country Trail is not prudent due to the extended process involved and the uncertainty of the outcome of this public process.



TDF REVISED ALTERNATIVE 3
US FOREST SERVICE PROPERTY LINE
NORTH COUNTRY TRAIL
US FOREST SERVICE PROPERTY LINE
ESTIMATED ALTERNATIVE #3 PERIMETER
WETLANDS (TYP.)
COLEMAN ENGINEERING COMPANY
CENTER TDF ALTERNATIVE #3
COPPERWOOD RESOURCES, INC.
FIGURE

8.4 Alternatives 4, 4A, 4B, and 4C: North On-Site Tailings Disposal Facility

Alternatives 4, 4A, 4B, and 4C were evaluated with the goal of containing the 28.7 million tons of expected tailings from the mining operation, each having a different configuration and dike height. These TDF alternatives are entirely south of the mine workings and are underlain by non-ore containing Copper Harbor Conglomerate formation (Figure 8-1). These proposed TDF locations also have a layer of natural clay-rich till up to about 100 feet in thickness in the center of the TDF locations. Generally, a minimum thickness of 15 feet of low permeability soil will be left in the bottom of the TDF. One important factor that has been taken into consideration for Alternatives 4, 4A, 4B, and 4C is the soil balance. The additional soil needed to construct these TDF alternatives is as follows:

- Alternative 4: 0.2 million cubic yards;
- Alternative 4A: 1.7 million cubic yards;
- Alternative 4B: 0.8 million cubic yards;
- Alternative 4C: 12.6 million cubic yards.

Table 8-2 below and Figure 1-1 and the Overall Site Plan (Figure 4-2 in the MDEQ application plan set) identify the on-site locations where 0.8 million cubic yards of low-permeability soil is available to be used in the TDF Alternative 4B construction and closure.

Table 8-2. On-Site Sources and Approximate Quantities of Low-Permeability Soil to be used in TDF Construction/Closure

Location of Source of Soil	Available Quantities (cubic yards)
Stream Relocation Channels	195,000
Wetland Construction	195,000
Excess TDF Excavation	410,000
Total Low-Permeability Soil for TDF	**800,000**

All of this soil is not required until the closure of the TDF. Soil generated prior to use will be predominately stored in the area between the access road and TDF; however, some soil may be stored in one of the upland areas shown as "Construction Staging Area / TDF Cap Borrow Area" on the Overall Site Plan (Figure 4-2 in the Application for Permit as revised). All of the above sources for additional soil are *on-site* and will be borrowed and/or stored in upland areas so as to have no additional impact to aquatic resources and therefore not require further Part 301 or Part 303 permitting.

The Golder design for the TDF embankment chimney drains also requires approximately 430,000 cubic yards of sand material and the capillary break layer in the closure cap requires approximately 375,000 cubic yards of sand that is not available on-site. The nearby sand and gravel pit on CR 519 has an estimated sand supply of 120,000 cubic yards. The remaining 685,000 cubic yards will have to come from a currently unidentified off-site supply source. With no other nearby, within 5 miles, sand sources for this amount will have significant cost along with an undetermined potential wetland impact. As a result, GMining is recommending that Golder replace the sand requirement with two commercially available manufactured products that can perform the same functions (Terradrain Strip Drain for the chimney drain, and Terradrain 900 for the capillary break layer).

Alternatives 4, 4A, 4B, and 4C were located to avoid impacts to Namebinag Creek to the west and the East Branch Gipsy Creek to the east. The ground at the location between those watercourses is relatively flat, thereby generally reducing the height of the required downstream dikes. It should be noted that all of these alternatives are more than 1.5 miles, at their closest point, from Lake Superior. Geotechnical investigation results indicate that the subsurface conditions are favorable for TDF construction at this location; i.e. there is approximately 100 feet of primarily clay soil underneath these TDF sites. These alternatives would impact the upper ephemeral portions of several streams with flow originating from spring snow melt and precipitation, as these stream sections have no groundwater-fed base flow.

8.4.1 Description of TDF Alternative 4

TDF Alternative 4 is located in the most feasible location for the TDF (Figure 8-1). TDF Alternative 4 would impact 51.27 acres of wetland and will result in the removal of 16,557 linear feet of ephemeral stream channel. TDF Alternative 4 would require the excavation and placement of approximately 10 million cubic yards of earth to construct the dike and the final cover. As stated above, TDF Alternative 4 has a negative soil balance of approximately one million cubic yards, i.e. one million cubic yards of additional low permeability soil will be needed during the closure period for the TDF.

The total project construction costs are detailed in the 2012 Orvana BFS for three stages of construction and closure for TDF Alternative 4. Costs are estimated to be \$81.45 million for construction and \$20.82 million for closures, for a total estimated construction cost of \$102.27 million.

TDF Alternative 4 is feasible and prudent when compared to some of the other TDF alternatives. Although TDF Alternative 4 is not necessarily the least environmentally damaging alternative, it is a feasible and prudent alternative when all aspects of the project are considered, including cost, logistics for mine operations, alternative availability, and mine safety. However, as described in the following sections, a variation of TDF Alternative 4 (Alternative 4B) was been determined by Orvana to be a better alternative for the TDF.

8.4.2 Description of TDF Alternative 4A

In an attempt to reduce the wetland impacts for Alternate 4, in development of analysis of this alternative, a revised TDF configuration was evaluated to avoid two larger wetlands in the southwest corner of the TDF (Figure 8-1). Based on this reconfiguration of the TDF, approximately 15 acres of wetlands would be avoided. However, this reconfiguration reduces the storage volume of the TDF by about 10%. This storage volume reduction is not acceptable therefore this alternative requires the embankment height around the entire TDF to be raised by eight feet in order to provide the 10% lost tailings storage capacity. This increase in embankment height would result in the need to place an additional 2.5 million cubic yards of low-permeability soil. The estimated cost to haul and place the imported soil is estimated to be $8 per cubic yard, for a total of at least $12 million in additional soil costs.

The additional area of excavation required to generate this necessary TDF soil material would likely result in additional direct wetland impacts, depending upon where the 2.5 million yards of soil would be able to be obtained. The approximately 15 acres of wetland to be avoided on-site in this alternative would be unavoidably isolated by the remaining TDF embankment on two sides and the site access road on another, reducing the functions and values of those 15 acres of otherwise-preserved wetlands. Based on the high cost of obtaining the additional low-permeability soil needed to implement this alternative and the potential for direct and indirect wetland impacts, Alternative 4A is determined to be not prudent.

8.4.3 Description of TDF Alternative 4B

The design of TDF Alternative 4B has the west berm moved eastward approximately 100 feet (as compared to Alternative 4) to allow for a natural stream channel to be designed and constructed between the outer toe of the TDF and Namebinag Creek (Figure 8-1). This scenario requires the berm crest to be increased from elevation 946 MSL to approximately 949.5 MSL in order to incorporate the required TDF volume. The volume of borrow soils available under this scenario dropped to 8.7 million cubic yards, while the volume of soil needed increased to 0.8 million cubic yards, thus increasing the cost by approximately $3.6 million. While this configuration does not substantially reduce regulated resource impacts as compared to Alternative 4 (wetland impacts are 51.25 acres and stream impacts are estimated to be 16,557 feet), it does allow for a natural channel design of the stream diversion along the westerly edge of the TDF. Otherwise, instead of the natural channel, a ditch channel would be utilized to convey stream flow around the west end of the TDF, which does not incorporate the stream habitat provided by the natural stream channel design. For these reasons, TDF Alternative 4B has been determined to be the most feasible and prudent alternative and is therefore the preferred/proposed TDF alternative, even though it is $3.6 million higher in cost than Alternative 4.

8.4.4 Description of TDF Alternative 4C

The design of TDF Alternative 4C has the east berm moved westward in order to avoid impacting the East Branch of Gypsy Creek (Figure 8-1). This alternative design would require the berm crest of the TDF to be increased approximately 45 feet, from elevation 946 ft-amsl to 991 ft-amsl in order to incorporate most of the required TDF volume. Even with this berm height increase, the required tailings volume would not be accommodated and 300,000 cubic yards of tailings storage space would be lost. Volume calculations indicate that a further incremental increase of the berm height from 991 to 1001 ft-amsl would result in additional loss of available tailings storage volume due to the inside toe of the slope on each side of the TDF coming together, therefore limiting the raising the height of the berm above 1001 ft-amsl. In this alternative, the volume of borrow soils available dropped to 4.9 million cubic yards, increasing the volume of soil needed to 13.4 million cubic yards. This earthwork imbalance would add approximately $99 million to the project construction costs. For these reasons, TDF Alternative 4C is determined to be not prudent and was therefore given no further consideration.

8.4.5 Selection of Proposed TDF Alternative 4B as the Proposed TDF Alternative

TDF Alternative 4B has been selected as the proposed TDF alternative because it has been determined to be the most feasible and prudent alternative. Although the cost is approximately $3.6 million more and the volume of additional low-permeability soil increased by 600,000 cubic yards compared to TDF Alternative 4, providing the additional area on the west side of the proposed TDF for the natural stream channel design to be implemented for the stream relocation by moving the proposed TDF berm east about 100 feet is an overriding positive benefit. Due to the fact that all of the additional 0.8 million cubic yards of low-permeability soil needed for TDF Alternative 4B is available on the Copperwood Project site, the cost of the additional soil needed for this alternative is not a significant detrimental consideration of this alternative. Alternative 4B impacts approximately 51.25 acres of wetland and 16,557 linear feet of ephemeral streams.

8.4.6 Threatened Plant Species

Showy orchis (*Galearis spectabilis*), which is listed as a threatened plant species in the State of Michigan, has been found within the limits of the proposed TDF. During a 2011 survey, 23 plants were identified in communities #5 and #6. Prior to impacting this plant species, a permit is required from the Michigan Department of Natural Resources (MDNR) under Part 365 of NREPA (Endangered Species Protection). An application for the permit to relocate the 23 plants was submitted to MDNR on April 23, 2012; MDNR issued Threatened & Endangered Species Permit #2004 on June 19, 2012 to appropriately address the known occurrences of showy orchis. Two populations of 23 orchids were transplanted from within the proposed TDF footprint under a Part 365 Endangered Species Permit # 2004 in October of 2012 with additional monitoring of these two areas and a third community not defined in 2012 continuing since that time. If TDF construction on the proposed site is approved in a new permit, a second transplant permit will be applied for to move remaining known orchid plants to similar habitat (acidic soils in ephemeral stream drainage areas) that is fairly common on the Copperwood Project site.

8.4.7 Line-of-Sight Analysis

A line-of-sight analysis was conducted for TDF Alternative 4, which has a proposed dike height of 137 feet. The proposed TDF Alternative 4B has a dike height of 140.5 feet, so the line-of-sight analysis is also applicable for TDF Alternative 4B. The analysis determined that the proposed TDF will be at least partly visible from the Lake of the Clouds overlook at Porcupine Mountains Wilderness State Park, the crest of Copper Peak, the top of the Copper Peak scaffold, and Summit Peak Lookout (Figure 8-5). Table 8-3 provides the estimated distance and portion of the TDF visible from each of these vantage points when the TDF is fully constructed.

Table 8-3. Estimated Distance and Portion of TDF Alternatives 4 and 4B Visible from Area Vantage Points

Vantage Point	Distance from TDF	Estimated % of TDF Visible
Lake of the Clouds Overlook	14 miles	47%
Crest of Copper Peak	7 miles	2%
Top of the Copper Peak Scaffold	7 miles	52%
Summit Peak Lookout	11.5 miles	82%

Because the dikes will be vegetated there should be minimal impact to the view shed other than short-term impacts. These impacts are unavoidable but are not expected to be a major long-term concern.

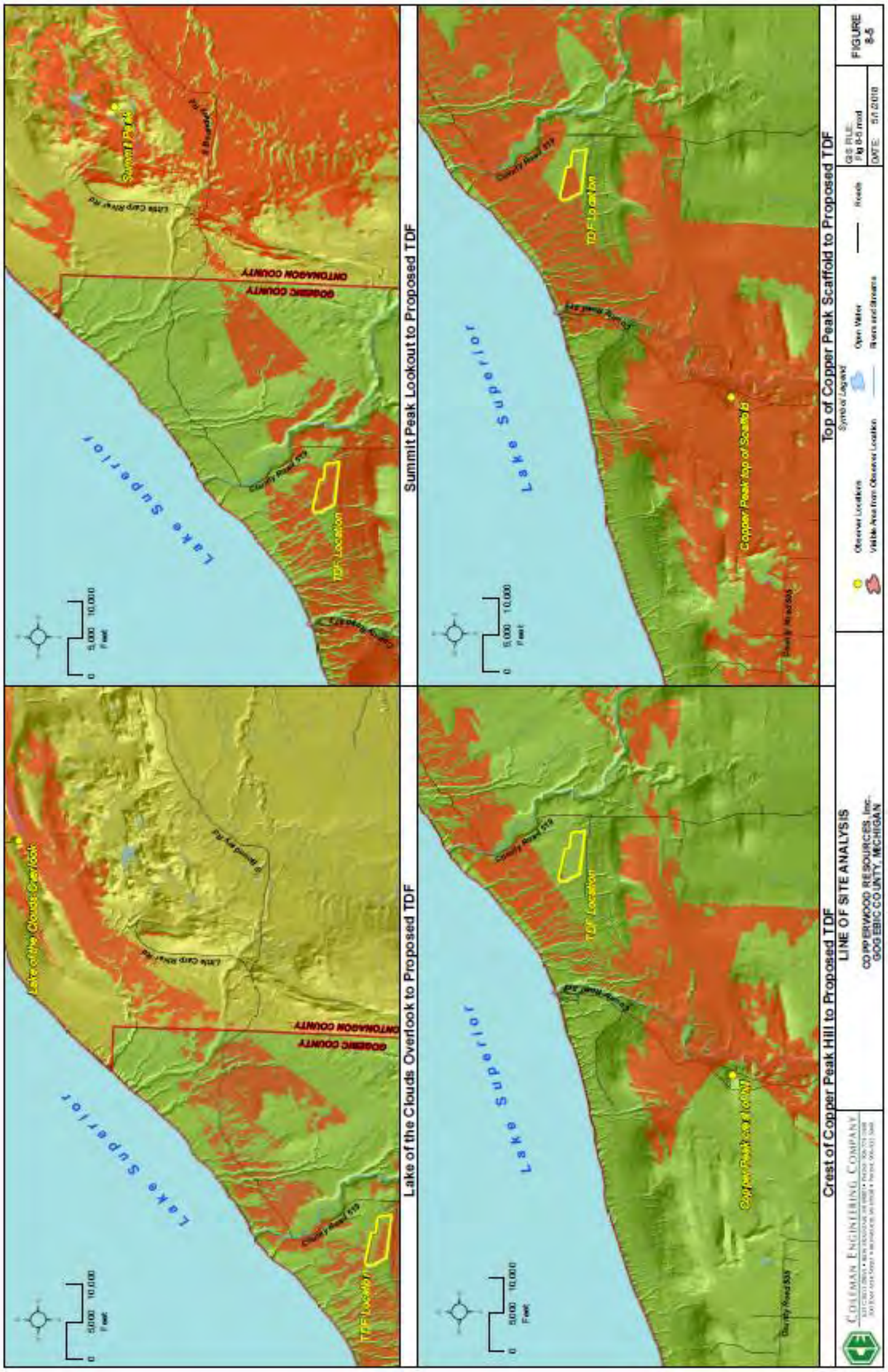
8.4.8 TDF Perimeter Road

An 11,528-foot long perimeter road will be constructed around the proposed TDF Alternative 4B.

The reasons for the perimeter road are:

- Provide for construction access as dikes are raised for each phase of the TDF. Dike construction is “downstream” construction, meaning that the outside of the dikes will have low permeability soil added to construct the lift;
- Monitoring soil erosion and stability of the TDF dikes;
- Define the outer limit of disturbance of final footprint of the TDF. Until the final lifts of the TDF dikes are constructed in Phase 3 (as these final TDF dikes are constructed, they expand outward) the perimeter road will not be at the toe of the TDF dikes, and;
- The proposed mine access road will also serve as the TDF perimeter road along the south portion of the TDF.

The proposed TDF perimeter road wetland and stream impacts are included in the totals for the proposed TDF (TDF Alternative 4B). Existing stream channels will remain down-gradient from the proposed perimeter road on the north side of the proposed TDF; stream channels will be re-routed upstream (south) of the proposed perimeter road along the south side of the proposed TDF and will be routed both easterly for Gipsy Creek and westerly for Lehigh Creek.



8.4.9 Analysis of TDF Depth and Liner Requirements

The allowable depth of the TDF basin excavation was developed by Golder using information from boreholes and monitoring wells developed during hydrogeological investigations. The allowable depth of excavation was prepared using the higher of two surfaces across the TDF. The first is a surface that provides for 15 feet or more of low-permeability soils below the TDF floor. The second is developing an artificial surface that can be designed to withstand hydrostatic uplift forces. The manner in which the thickness of that artificial surface (above the 15 feet of more of clay) is determined is described in the following paragraphs.

The TDF basin excavation is designed using cut slopes of 2.5H:1V. This slope angle for the excavation below natural grade will provide end-of-construction stability given the large depth of cut and will also facilitate subgrade preparation (compaction of upper one foot of foundation material). Based on the subsurface geology and hydrogeological conditions presented in these boreholes and monitoring well logs, Golder believes it is reasonable to assume the bedrock and weathered bedrock units act as a type of “aquifer unit”, and that the silty glacial overburden unit acts as a confining unit over the aquifer.

Under the assumed site conditions, the bedrock piezometric surface applies an upward piezometric pressure, i.e., hydrostatic uplift, at the bottom of the confining layer (the glacial overburden). The hydrostatic uplift pressure must be resisted by an equal weight of the glacial overburden to prevent soil boiling or upheaval. A factor of safety against hydrostatic uplift is commonly calculated as a ratio between the resisting weight of confining soil (glacial overburden) and the uplift pressure. For this evaluation, Golder maintained a factor of safety of 1.25 to develop the TDF allowable excavation limit. A contour map showing the hydrostatic uplift isobars for the TDF floor grades as compared to the highest potentiometric surface (i.e. the bedrock aquifer) is included in Appendix H.

For each of the monitoring wells in the TDF area, Golder selected the top of “aquifer unit” base on the corresponding borehole lithology log. Additionally, to calculate the hydrostatic uplift force, Golder selected the maximum piezometric water level recorded for each well screened in the “aquifer unit”. The hydrostatic pressure is equal to the height of the piezometric elevation above the top of the “aquifer unit” times the unit weight of water (62.4 pounds per cubic foot (“pcf”)).

The glacial overburden thickness required to resist the hydrostatic uplift was calculated by dividing the hydrostatic uplift pressure by the glacial overburden soil wet density identified during laboratory testing (139 pcf), then multiplying by the selected factor of safety (a factor of 1.25 for uplift). Golder then calculated the allowable excavation elevation by adding the required glacial overburden thickness to the top of the “aquifer unit” elevation.

The piezometric elevation and glacial overburden/”aquifer unit” contact elevation varies throughout the TDF, thus the hydrostatic uplift pressure will vary throughout the TDF. To develop a three-dimensional allowable excavation surface, Golder repeated the calculation described at each well in the TDF area. Using the allowable excavation elevations calculated at each point, Golder created a three-dimensional surface using AutoCAD Civil 3D. This allowable excavation surface was used as the basis for the TDF basin excavation grades from an uplift perspective. The depth of the TDF basin was designed to remain above the allowable excavation surface in all areas.

The analysis of the TDF liner requirements was not conducted for either Alternatives 2 or Alternative 3 due to the excessive negative soil balance (estimated at almost 10 million cubic yards) of these revised alternatives and the related costs. These costs render both of these alternatives not prudent. The estimate of negative soil balance was determined based upon the approximate depth to bedrock that was estimated to occur in each of these two revised TDF alternatives. This condition resulted in a determination that there would be a shortage of low-permeability soil, with the soil balance necessary to construct each TDF being obtained from an off-site location, requiring purchase and/or hauling costs.

8.4.10 Analysis of TDF Berm Height to Minimize or Avoid Wetland and Stream Impacts

The height of the berms around the TDF directly impacts both the amount of soil required for the construction of the TDF and the available storage volume within the TDF. While the volume of generated tailings planned for the project is known, the depth of the TDF as excavated into the ground and the corresponding berm height required to contain the requisite volume of tailings were not initially known. Subsequent information generated during the hydrogeological investigations allowed Golder to maximize the depth of the TDF and determine berm heights.

The berm height around the TDF is determined through an iterative process in which the volume of soils generated from the excavation over an area and the volume of the berm of sufficient size to generate the required disposal volume are considered. Minimizing the cut and fill and achieving a soil balance on a project is optimal. If the amount of soil needed exceeds the amount of soil generated by the TDF excavation, then the solution typically involves a deeper excavation or an increased area of impact to provide additional soil volume from excavation. If the excavation depth has been maximized, increasing the area is typically the only way to achieve a soil balance. If soil balance cannot be achieved and additional soils are not available from other locations on the site, then off-site soils would be needed and costs are substantially increased as described earlier in this document. Impacts to wetlands may also occur at off-site soil borrow locations.

The vertical component of the TDF alternatives was determined and optimized in order to minimize the wetland and stream impacts as well as providing the necessary TDF storage capacity, all within prudent cost parameters. For TDF Alternative 4, 4A, 4B, and 4C locations, because the average depth to bedrock is deeper compared to other on-site TDF alternatives, it is achievable to maximize the soil balance as the amount of fill needed for these TDF alternatives (except for TDF Alternative 4C). TDF Alternative 4B for example is within approximately 1.6 million cubic yards of the amount of cut generated (out of a total of approximately nine million cubic yards of soil to be cut and filled). If the berm heights were raised for TDF Alternative 4B in an effort to decrease the 340-acre footprint and thereby reduce wetland and stream impacts, there would be a need to obtain additional fill material from another source, either on-site or off-site. This would also require an on-site area to be utilized for stockpiles of fill material.

For example, and generally speaking, if the berm heights of TDF Alternative 4 were to be raised by approximately 25 feet from elevation 946 MSL to 971 MSL, there would be a reduction in the overall footprint of approximately 41 acres, from 321 acres to 280 acres. The amount of wetland impacts could therefore be reduced, in theory, by approximately 13 percent, depending on the orientation of the new footprint. However, in that example scenario, there would be a need to obtain approximately 5.6 million cubic yards of additional material to build the extra 25 feet in berm height and the associated issues of both obtaining that amount of material and then temporarily storing it on the site.

On the Copperwood Project, the amount of low-permeability soil required for berm construction and closure cap exceeds the amount of soil generated by excavation within the TDF. As noted in Table 8-2, additional soil for the construction of TDF Alternative 4B will be generated through one or more of the other planned site activities such as: on-site wetland mitigation excavation; stream relocation excavation; sewage lagoons and mill site runoff pond; excess material generated during roadway construction; and/or upland staging areas. The volume of soil needed for the TDF was balanced for the project overall to limit/minimize borrowing and spoiling, thereby minimizing those additional possibilities of project wetland and stream impacts.

8.4.11 Determination of Cut and Fill for TDF Construction

The amount of cut generated from the excavation of the proposed TDF alternatives was calculated using a computer software model called AutoCAD Civil 3D. Using this model, the existing topographic surface map of the TDF that was prepared by Aero-Metric, Inc. from aerial photographs taken May 4, 2009 (Horizontal State Plane North; Vertical Datum NAVD 88) was compared to a second surface which was comprised of the designed excavation grades. The difference between the two surfaces represents the soil quantity that will be generated during excavation.

The soil volume required to construct the perimeter and interior berms was also determined using the same software model by comparing two surfaces. Again, the existing topographic map was used as one surface that was compared to a second surface comprised of the designed berm configuration. The difference between the two surfaces represents the quantity of soil required for the berm construction.

Additionally, the volume of soil estimated for the construction of the final cover was based on the area of the final cover and the design thickness.

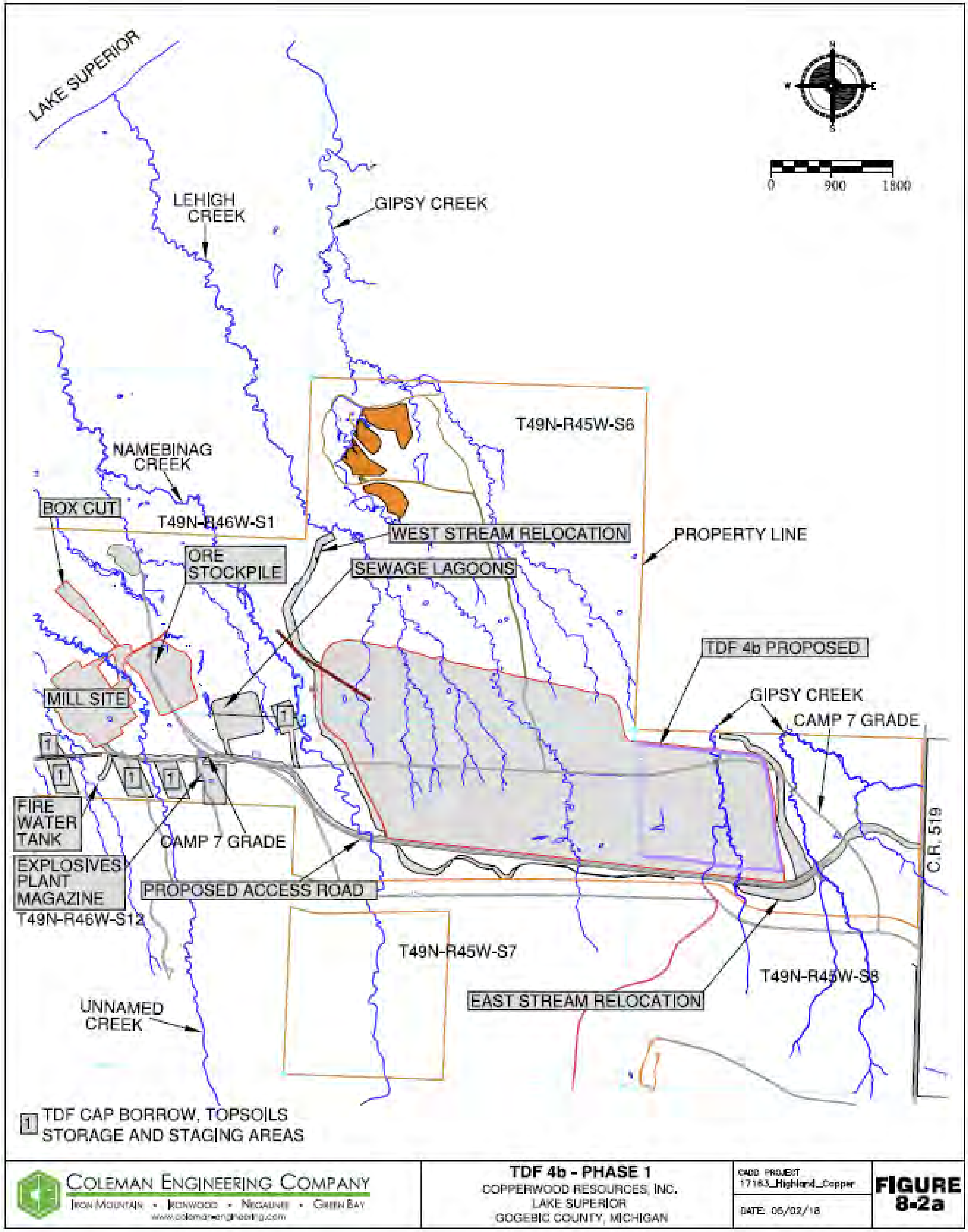
8.4.12 Summary of Phases of TDF Construction

There will be three phases of TDF construction, roughly one-third of the TDF being constructed during each phase. (The updated feasibility study plan for phase construction of the proposed Alternative 4B is to move from east to west, as compared to the 2012 plan where the stages were to progress from west to east. While the surface area of each phase is roughly the same, the western phase involves more cost due to the larger embankment size. Project economics can be enhanced if pre-production capital costs can be deferred until operating income is being generated by the project.) The current schedule for TDF construction would have the first two phases being constructed during the 5-year permit period. However, due to the necessary earthwork involved in the TDF construction and the associated on-site activities, it is anticipated that all wetlands within the proposed TDF footprint would be impacted within the initial five-year MDEQ permit period as well as other proposed wetland impact areas. In this scenario, the need for an additional MDEQ permit is not anticipated. However, in the event scheduling changes, Copperwood is aware of the need to obtain a permit revision (or another permit if any proposed/permitted stream or wetland impacts have not yet been made within five years) if changes to the site plan are necessary.

The proposed TDF will serve to store not only tailings, but also contact water during pre-production development of the underground mine. The Phase 1 area of the TDF would be used to store contact water from the mine and surface runoff and, when mill production begins, to receive tailings from the ore processing (Figure 8-2a).

Phase 2 of the TDF will be developed by constructing additional berms prior to Phase 1 reaching its capacity for tailings disposal (Figure 8-2b).

The last phase of the TDF construction is Phase 3 (Figure 8-2c). Prior to Phase 3 being utilized for tailings disposal, the topsoil stockpile located in Phase 3 will be utilized to top-dress TDF berms for permanent seeding. The outside slopes of the TDF berms, including the stormwater berms, will be vegetated by seeding, fertilizing, and mulching exposed soil. Large woody vegetation will not be allowed to grow to maturity and will be cut or removed consistent with routine operation and maintenance activities at the site.


LAKE SUPERIOR
LEHIGH CREEK
GIPSY CREEK
0 900 1800
T49N-R45W-S6
NAMEBINAG CREEK
BOX CUT
T49N-R46W-S1
WEST STREAM RELOCATION
PROPERTY LINE
ORE STOCKPILE
SEWAGE LAGOONS
TDF 4b PROPOSED
MILL SITE
GIPSY CREEK
CAMP 7 GRADE
FIRE WATER TANK
CAMP 7 GRADE
C.R. 519
EXPLOSIVES PLANT MAGAZINE
PROPOSED ACCESS ROAD
T49N-R46W-S12
T49N-R45W-S7
T49N-R45W-S8
EAST STREAM RELOCATION
UNNAMED CREEK
1 TDF CAP BORROW, TOPSOILS STORAGE AND STAGING AREAS
COLEMAN ENGINEERING COMPANY
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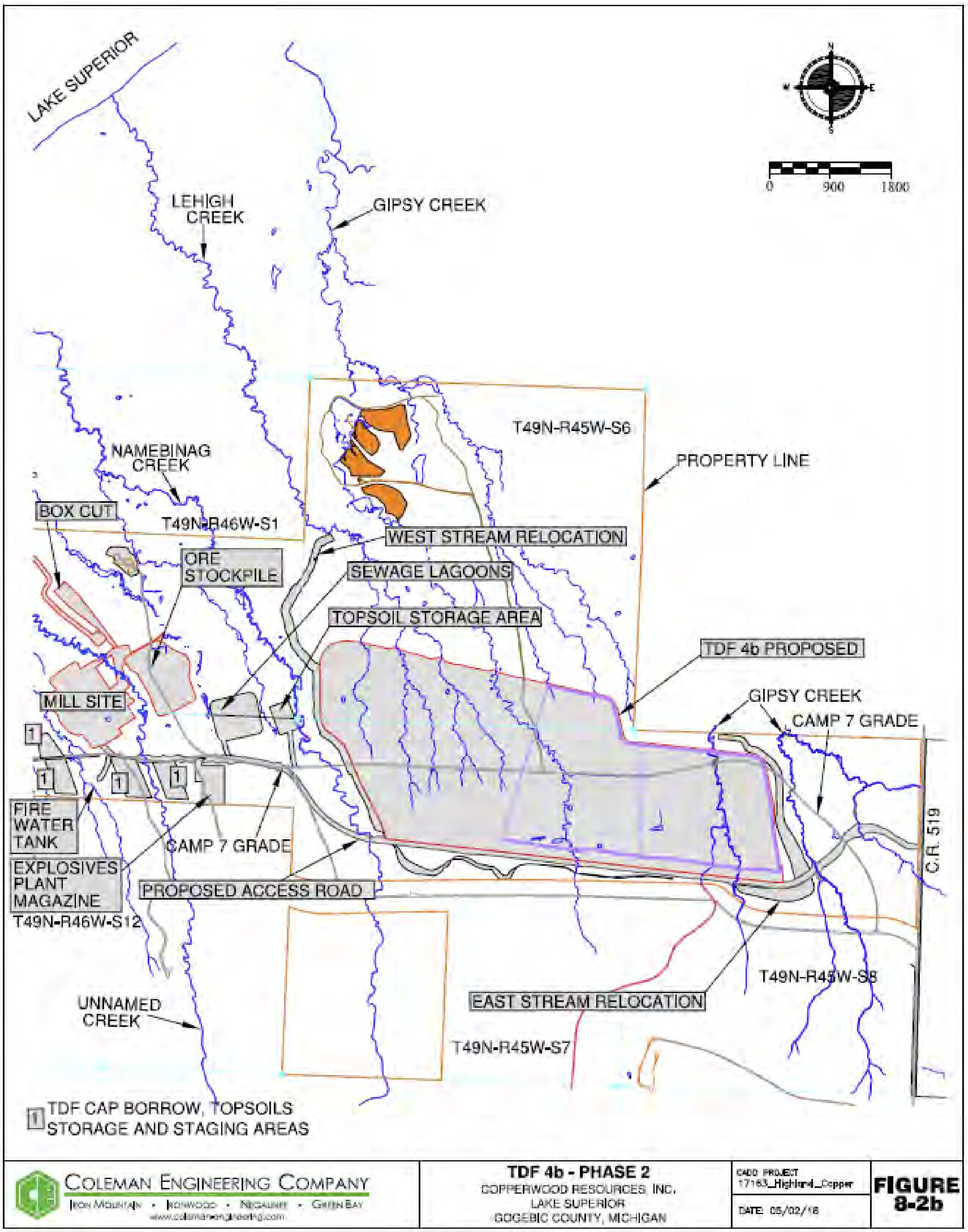
www.colemanengineering.com
TDF 4b - PHASE 1
COPPERWOOD RESOURCES, INC.
LAKE SUPERIOR
GOGEBIC COUNTY, MICHIGAN
CADD PROJECT 17183_Highland_Copper
DATE: 06/02/18
FIGURE 8-2a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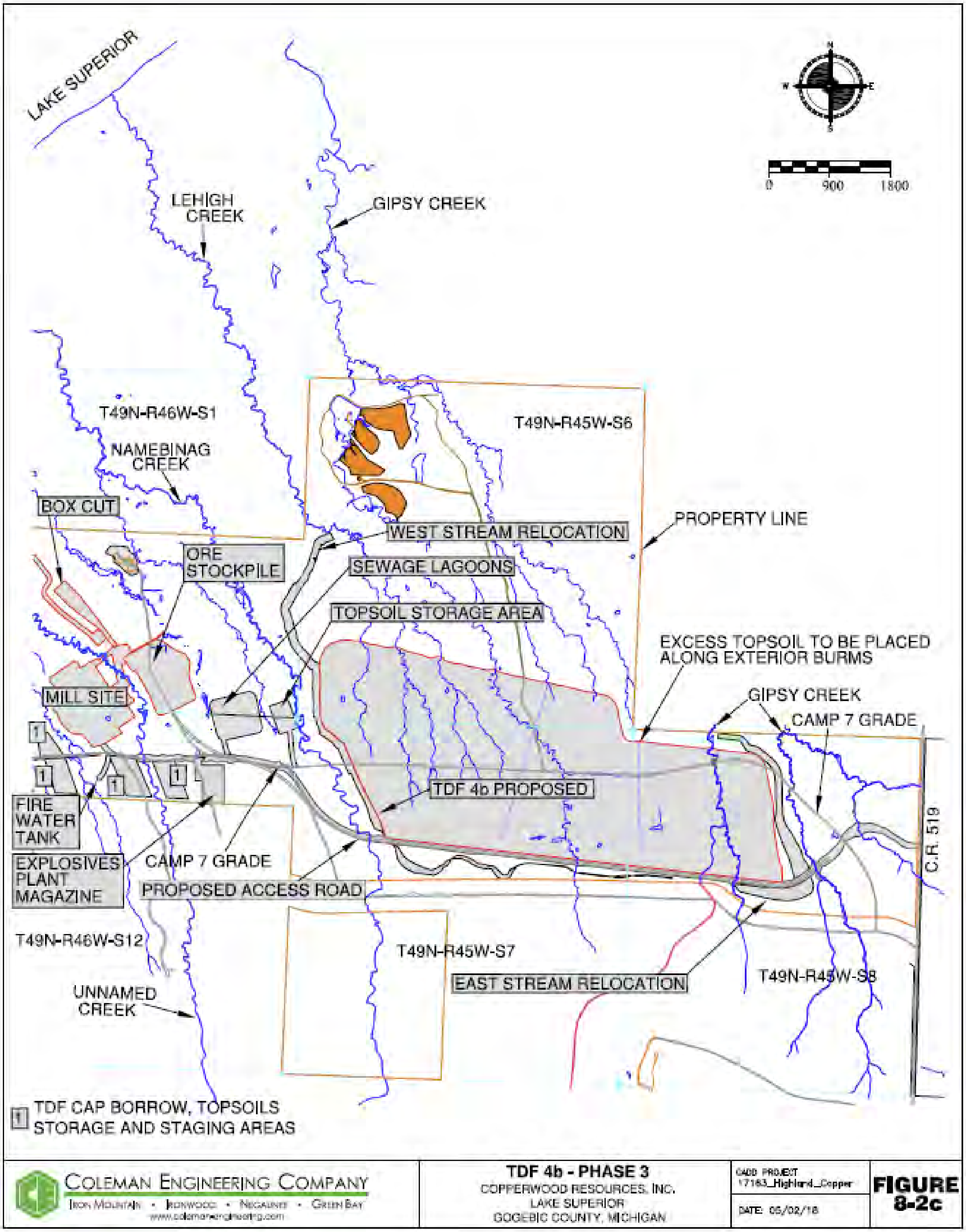
PROPERTY LINE
NAMEBINAG CREEK
BOX CUT
T49N-R46W-S1
WEST STREAM RELOCATION
ORE STOCKPILE
SEWAGE LAGOONS
TOPSOIL STORAGE AREA
TDF 4b PROPOSED
MILL SITE
GIPSY CREEK
CAMP 7 GRADE
FIRE WATER TANK
CAMP 7 GRADE
EXPLOSIVES PLANT MAGAZINE
PROPOSED ACCESS ROAD
T49N-R46W-S12
C.R. 519
T49N-R45W-S8
UNNAMED CREEK
EAST STREAM RELOCATION
T49N-R45W-S7
TDF CAP BORROW, TOPSOILS STORAGE AND STAGING AREAS
COLEMAN ENGINEERING COMPANY
IRON MOUNTAIN • IRONWOOD • NEGAUNEE • GREEN BAY
TDF 4b - PHASE 2
COPPERWOOD RESOURCES, INC.
LAKE SUPERIOR
GOGEBIC COUNTY, MICHIGAN
CADD PROJECT
DATE: 05/02/18
FIGURE 8-2b


LAKE SUPERIOR
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LEHIGH CREEK
GIPSY CREEK
T49N-R46W-S1
T49N-R45W-S6
NAMEBINAG CREEK
BOX CUT
WEST STREAM RELOCATION
PROPERTY LINE
ORE STOCKPILE
SEWAGE LAGOONS
TOPSOIL STORAGE AREA
EXCESS TOPSOIL TO BE PLACED ALONG EXTERIOR BURMS
MILL SITE
GIPSY CREEK
CAMP 7 GRADE
TDF 4b PROPOSED
FIRE WATER TANK
EXPLOSIVES PLANT MAGAZINE
CAMP 7 GRADE
PROPOSED ACCESS ROAD
C.R. 519
T49N-R46W-S12
T49N-R45W-S7
EAST STREAM RELOCATION
T49N-R45W-S8
UNNAMED CREEK
1 TDF CAP BORROW, TOPSOILS STORAGE AND STAGING AREAS
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CADD PROJECT 17183_Highland_Copper
DATE: 05/02/18
FIGURE 8-2c

8.5 Alternative 5: Off-Site TDF Search in Vicinity of Site

In preparation for submittal of the Alternatives Analysis, an additional search was conducted in an attempt to verify whether any potential off-site TDF locations exist. Certain constraints exist for off-site TDF locations that affect the feasibility of siting a TDF. The Presque Isle River is located approximately two miles east of the Copperwood Project and the Black River approximately two miles west. Both rivers have portions designated as a Scenic Rivers on lands owned by the Ottawa National Forest, thereby eliminating locating a TDF in those directions from the Copperwood Project. Potential TDF locations to the south of the project are available that could potentially accommodate a tailings facility of the required capacity; however, the watersheds of the Black River and Presque Isle River also begin approximately one mile to the south of the proposed mine site. Both of these rivers are popular recreation destinations with a network of trails, waterfalls, day use areas and campgrounds; locating the TDF in the headwater portions of these watersheds is not prudent. Clay-rich, low permeability soils are necessary for construction of the TDF; if those soils are not present on the TDF site, another site must be impacted to obtain approximately 10 million cubic yards of low permeability soil to construct the TDF. Therefore, to minimize impacts on the landscape, a TDF site should contain suitable low permeability soils. Lastly, a TDF cannot be located over mine workings due to concerns over the safety of underground miners.

In order to objectively evaluate the availability of potential off-site TDF locations, a Geographic Information System (GIS) program was implemented by Coleman Engineering to conduct the off-site search. The GIS program utilized four attributes in the GIS search for potential sites. These four attributes were utilized to eliminate lands that are not suitable for location of the TDF. The four search elimination attributes were:

- Ottawa National Forest and Porcupine Mountains Wilderness State Park ownership;
- Black River and Presque Isle River watersheds;
- Insufficient clay-bearing soils; and
- Mine workings.

Data were obtained through the U.S. Forest Service, Michigan Geographic Data Library, USDA Data Gateway and Orvana Minerals. The data included Ottawa National Forest and Porcupine Mountains Wilderness State Park boundaries, watershed boundaries, spatial and tabular soils information, Digital Orthographic Quadrangle imagery, hydrography and proposed Copperwood mine workings. The areal extent of the mine workings proposed by Copperwood remains the same as that defined by Orvana and included in the issued Part 632 MP 01 2012 permit. A distance of six miles from the Copperwood Project site was selected as the search radius due to the cost and environmental considerations for constructing pipelines and pumping costs for longer distances. Each of the previously mentioned data sources was spatially compared in an effort to determine additional potential off-site TDF locations. Criteria deemed unsuitable for TDF placement within the six-mile radius search area was removed from consideration.

The results of the GIS search are depicted in Figures 8-6 through 8-10; an explanation of each figure is provided below. Please note that the area cross-hatched on each figure is shaded within the TDF search area on the subsequent figures so that the areas previously eliminated can be seen. Figure 8-10 depicts all areas that were eliminated and illustrates the remaining suitable areas.

Figure 8-6 depicts the TDF site search radius of six miles from the Copperwood Project mill site and removes from consideration those lands that are owned by the Ottawa National Forest and Porcupine Mountains Wilderness State Park. These public lands are not reasonably available for location of a TDF for the Copperwood Project.

Figure 8-7 displays the area of the underground mine workings at the Copperwood Project. Locating the TDF over the mine workings creates unsafe conditions for miners and is not prudent.

Figure 8-8 depicts the Black River and Presque Isle River watersheds within the search area cross-hatched. Locating the TDF in these watersheds is not prudent due to portions of these rivers being designated as Scenic Rivers. Maintaining the integrity of these two watersheds is important to the long-term protection of the rivers.

Figure 8-9 indicates the areas of insufficient clay-rich soils within the search area being "removed from consideration" as a possible location of a TDF. A "clay bearing soil" is defined by the U.S. Department of Agriculture (USDA) as any soil with more than 15% clay content. The GIS program that was utilized in this exercise uses the USDA Soils Spatial and Tabular Data Soils Map to provide the soil criteria that was considered to be "clay bearing soils". Therefore, any areas that had soils with clay content of less than 15% were removed from consideration during this search exercise.

Locating the TDF on land with insufficient low permeability soil would require another large site that has suitable clay-rich soil to be excavated and the low permeability soil used to construct the TDF on more permeable soils. The borrow site would undoubtedly have wetlands that would be impacted in addition to any wetlands on the potential TDF site. Also, situating the TDF on permeable soils will introduce more groundwater interaction with the TDF and potentially create undesirable contamination of groundwater. As seen in Figure 8-7, a large portion of the search area has soils that are not suitable for siting the TDF.

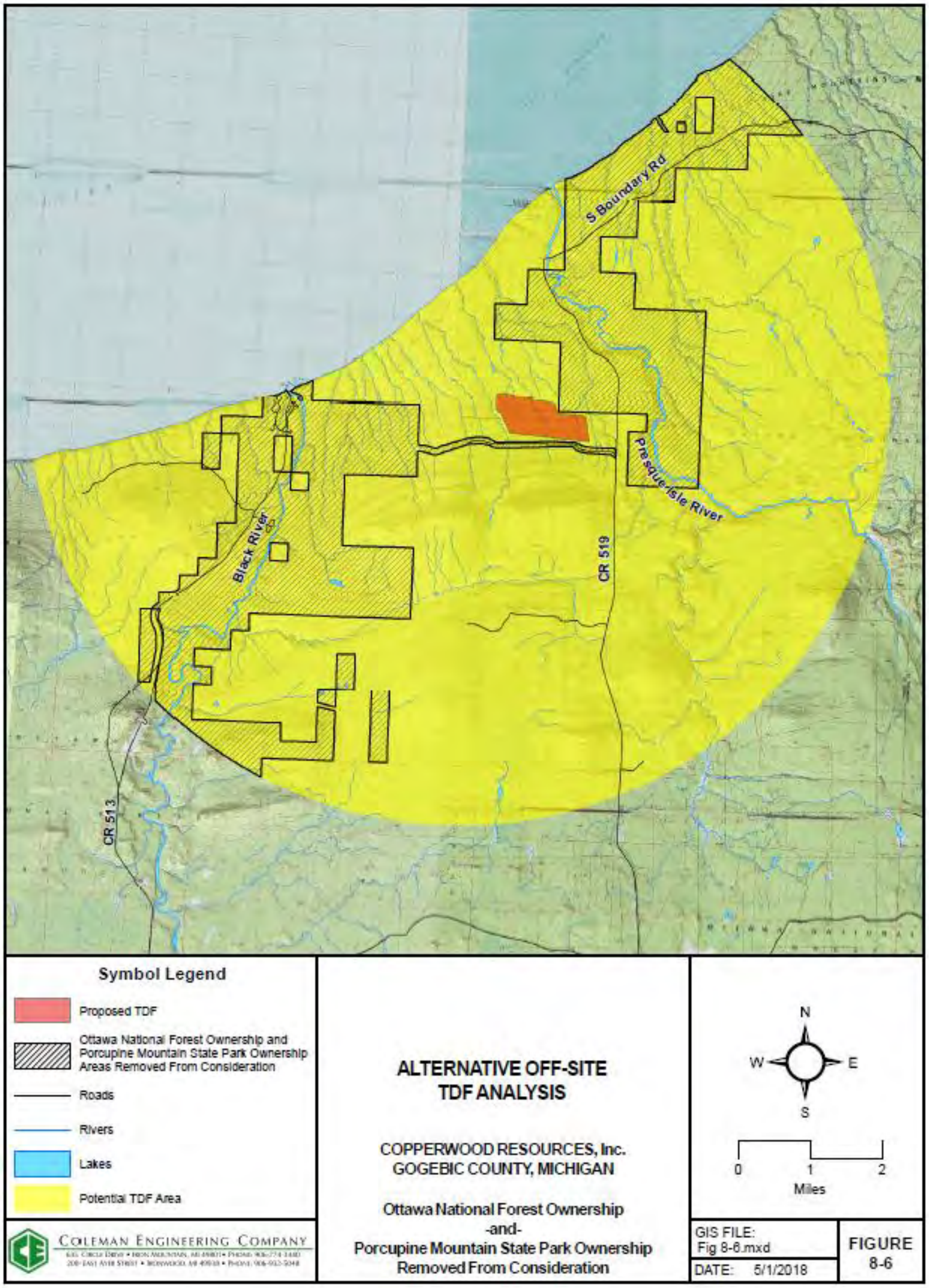
Figure 8-10 is the result of the TDF site search, with the areas shown in yellow on the figure as the only areas that made it through the screening process as potential TDF locations. In assessing the feasibility of the remaining sites from west to east, the areas shown west of the Black River are adjacent to Lake Superior and are not suitable from that parameter. Also, a pipeline to convey tailings to the TDF and a water return line from the TDF to the Copperwood Project mill site would have to be constructed through the Black River watershed and over the Black River, which is not prudent given the quality of this watershed and the designation of the Black River.

The yellow potential TDF sites located west and south of the proposed TDF for the Copperwood Project would locate the TDF closer to Lake Superior and would impact Namebinag Creek and Unnamed Creek, both of which have redside dace (State Endangered) in the lower stream reaches.

The potential sites north of the proposed TDF are each too small for the size of TDF that is required and these sites are located closer to Lake Superior and some would be over mine workings. The larger block of potential TDF sites located northeast of the Copperwood Project would require the construction of a pipeline through the Presque Isle River watershed and crossing of the Presque Isle River. The pipeline would also have to go across the Porcupine Mountains Wilderness State Park land, which is not feasible. Some of these locations are closer to Lake Superior and are adjacent to other rivers and streams that flow through the State Park.

8.5.1 Summary of Off-Site TDF Site Search

The result of the GIS search for potential TDF sites is that no feasible or prudent off-site TDF locations exist within a six-mile radius. Locating the TDF on the Copperwood Project site is the most feasible and prudent location when compared to off-site TDF locations.


S Boundary Rd
Presque Isle River
Black River
CR 519
CR 513
Symbol Legend
Proposed TDF
Ottawa National Forest Ownership and Porcupine Mountain State Park Ownership Areas Removed From Consideration
Roads
Rivers
Lakes
Potential TDF Area
ALTERNATIVE OFF-SITE
TDF ANALYSIS
COPPERWOOD RESOURCES, Inc.
GOGEBIC COUNTY, MICHIGAN
Ottawa National Forest Ownership
-and-
Porcupine Mountain State Park Ownership
Removed From Consideration
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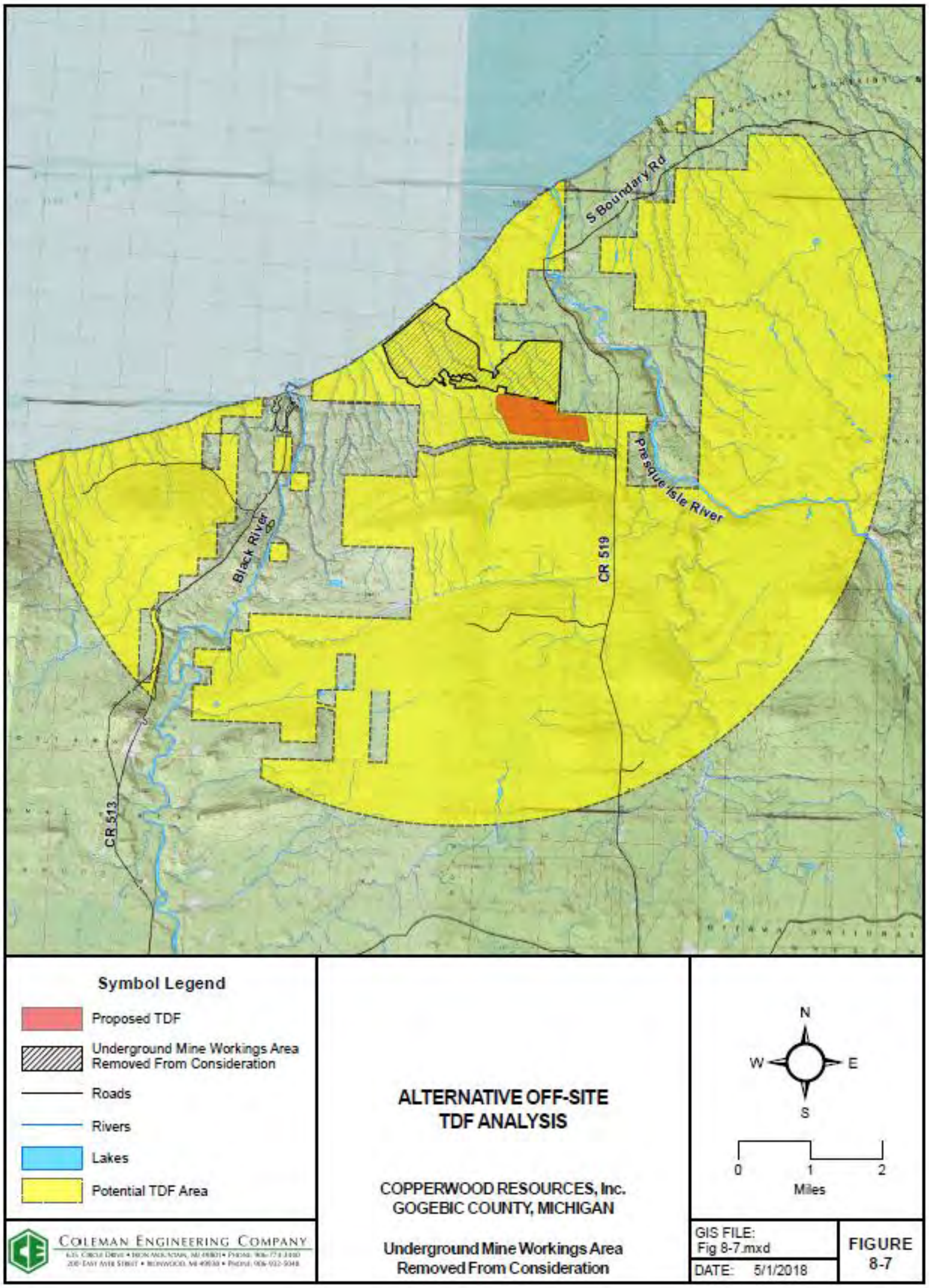
DATE: 5/1/2018
FIGURE
8-6
COLEMAN ENGINEERING COMPANY


S Boundary Rd
Presque Isle River
Black River
CR 519
CR 513
Symbol Legend
Proposed TDF
Underground Mine Workings Area Removed From Consideration
Roads
Rivers
Lakes
Potential TDF Area
ALTERNATIVE OFF-SITE TDF ANALYSIS
COPPERWOOD RESOURCES, Inc.
GOGEBIC COUNTY, MICHIGAN
Underground Mine Workings Area Removed From Consideration
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GIS FILE: Fig 8-7.mxd
DATE: 5/1/2018
FIGURE 8-7
COLEMAN ENGINEERING COMPANY

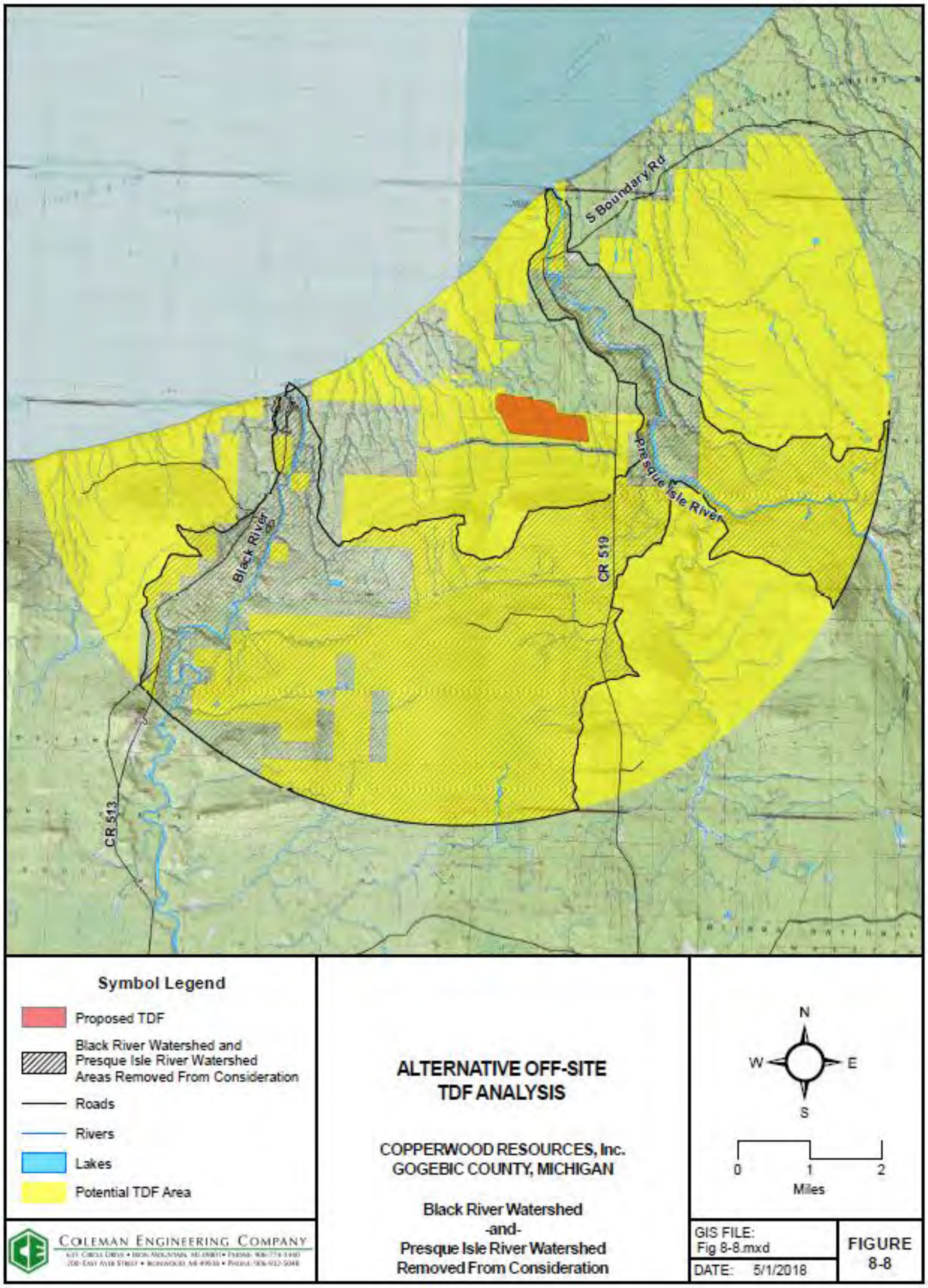
S Boundary Rd
Presque Isle River
CR 519
Black River
CR 513
Symbol Legend
Proposed TDF
Black River Watershed and Presque Isle River Watershed Areas Removed From Consideration
Roads
Rivers
Lakes
Potential TDF Area
Coleman Engineering Company
ALTERNATIVE OFF-SITE TDF ANALYSIS
COPPERWOOD RESOURCES, Inc.
GOGEBIC COUNTY, MICHIGAN
Black River Watershed
-and-
Presque Isle River Watershed
Removed From Consideration
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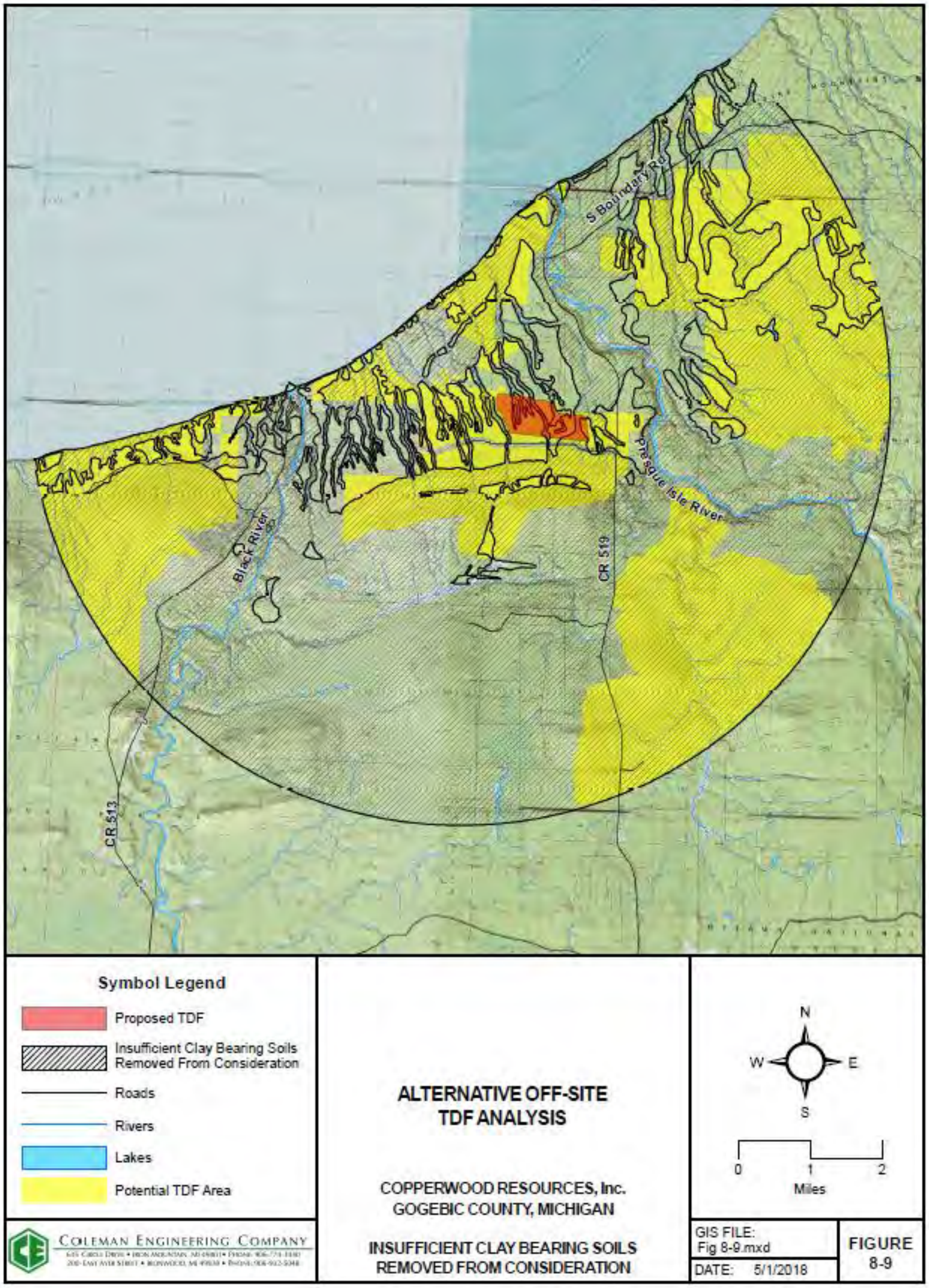
FIGURE
8-8


S Boundary Rd
Presque Isle River
Black River
CR 519
CR 513
Symbol Legend
Proposed TDF
Insufficient Clay Bearing Soils Removed From Consideration
Roads
Rivers
Lakes
Potential TDF Area
ALTERNATIVE OFF-SITE TDF ANALYSIS
COPPERWOOD RESOURCES, Inc.
GOGEBIC COUNTY, MICHIGAN
INSUFFICIENT CLAY BEARING SOILS REMOVED FROM CONSIDERATION
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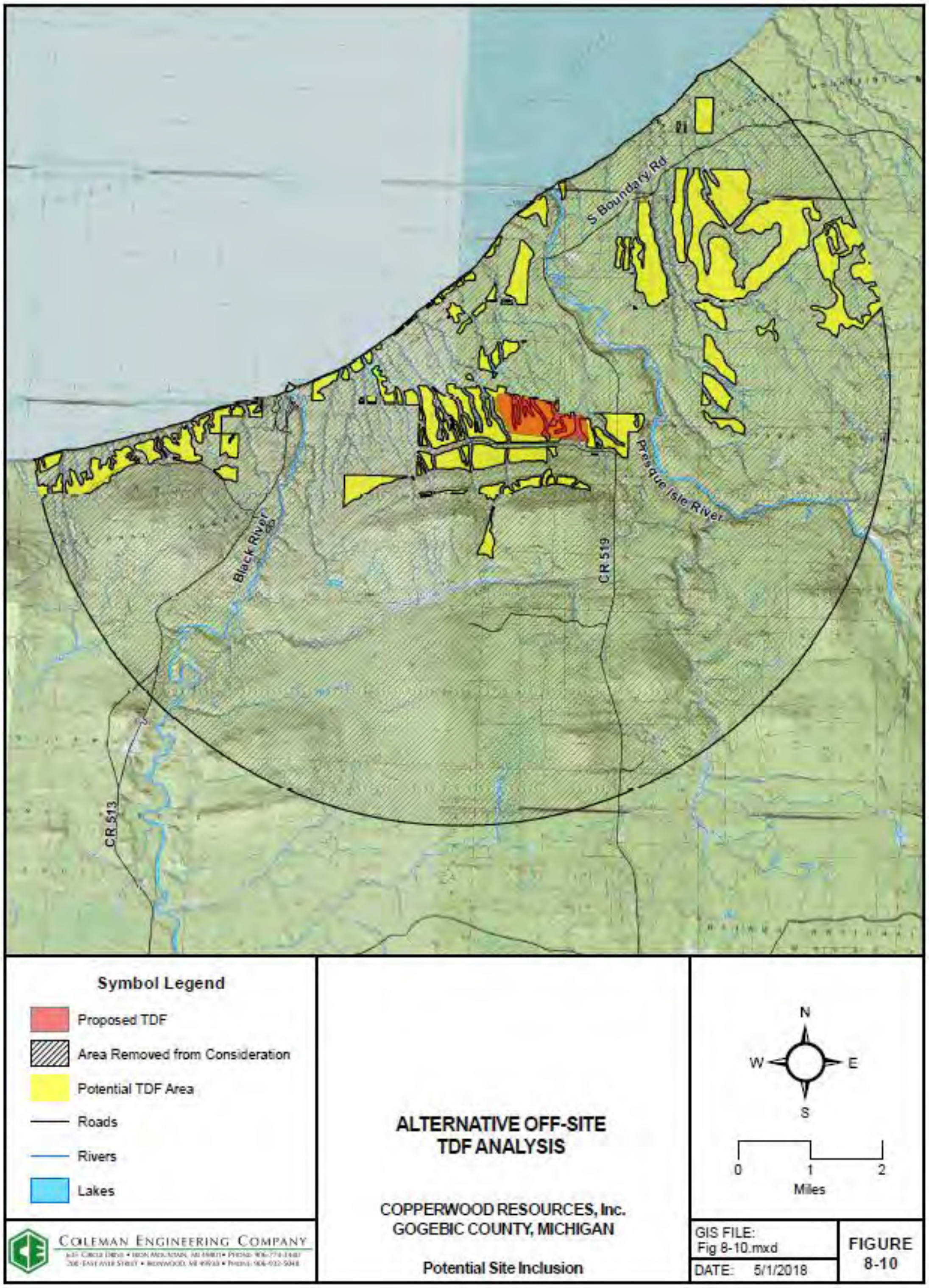
COLEMAN ENGINEERING COMPANY
GIS FILE: Fig 8-9.mxd
DATE: 5/1/2018
FIGURE 8-9



8.6 Alternative 6: On-site TDF Over Mine Workings

On-site disposal of tailings over active mine workings previously considered by Orvana in their scoping study is not prudent due to overriding mining safety concerns.

8.7 White Pine Alternatives

As discussed in Section 1.0 Introduction, Highland signed an interim agreement in May 2014 to acquire CRC's assets and environmental remediation liabilities at the former White Pine Mine Site. This acquisition, when consummated, will include the former tailings disposal facilities at White Pine. Highland's PFS study for a combined Copperwood-White Pine North Project, also discussed in Section 1.0, included completion of a draft tailings disposal tradeoff study by Golder Associates before work on the PFS was suspended in 2016 (Appendix F). The tradeoffs investigated by Golder include disposal in one basin or combinations of the three existing basins, namely the former North #1, North #2 and South Tailings Basins, as well as disposal in the underground mine, disposal in a disturbed area between the North #1 and South Basins and construction of completely new tailings disposal facilities at undisturbed locations in the vicinity of the former mine site.

Golder's tradeoff analyses for the White Pine TDF alternatives investigated requirements for the combined Copperwood (30,000,000 tonnes) and White Pine (90,000,000 tonnes) estimated tailings disposal requirements. The Golder report does, however, provide useful information to consider options for disposal of only the Copperwood tailings amount.

South Tailings Basin (referred to as the South Pond in the Golder report) and disturbed gap area between the South and North #1 Basins –
The South Basin encloses an area of 1,350 acres onto which approximately 240 acres of tailings were deposited by CRC before closing the White Pine Mine in 1997. Significant amounts of low permeability soil were removed from the existing embankment by CRC to apply a closure cap over the tailings deposit, leaving large gaps in the embankment structure. Preliminary field delineation work by Highland and aerial photo analysis identified significant wetland areas in the areas of the South Basin without tailings deposits and the disturbed gap area between the South and North #1 Basins (likely greater than 100 acres, see the wetland map included with Appendix E). For this reason, the South Basin at White Pine is not a feasible and prudent alternative for disposal of the Copperwood tailings at White Pine.

North #1 Basin –
Highland requested and received a determination from the MDEQ that wetlands occurring on the tailings surface within this, and the North #2 basins, are not considered to be regulated wetlands for permitting purposes. The North #1 basin has a surface area of 1,850 acres and Golder's analysis of Highland's processed Lidar topography data estimated an available storage space for only 11,000,000 tonnes of tailings without any embankment freeboard, making this option not feasible in its current state. Golder also concluded that adding a lift to the existing North #1 embankments to accommodate the estimated tonnage of Copperwood tailings is not technically feasible. The North #1 Basin by itself is not considered a feasible and prudent alternative for disposal of the Copperwood tailings.

North #2 Basin –

This basin has a surface area of 2,450 acres and Golder's analysis of the topography data estimated an available storage area of 22,000,000 tonnes, but again with no freeboard at the top of the embankments. The North #2 Basin was, however, not built up to its original design volume and is capable of accommodating lifts sufficient to handle all of the tailing production estimated for the Copperwood and White Pine North Projects. Golder estimated a 10-meter lift for the entire amount at a capital cost of $323,000,000 for the combined tailings. Scaling these amounts downward for only the Copperwood tailings yields an estimated 2.5 meter lift required at an estimated capital cost of $83,000,000. A disadvantage noted in the Golder report of adding additional tailings to the existing North #1 and #2 tailings basins is the anticipated need to provide a closure cap over the entire new tailings deposits in each basin, at an estimated cost of $13 to $15 per square meter. A closure cap for the area of the North #2 basin would have an estimated cost of $129,000,000 to $148,000,000. The total cost of lift construction and closure cap on North #2 would be well over $200,000,000. These costs can be compared to the estimated cost for the preferred alternative 4B for a TDF at the Copperwood mine site of $105,890,000 (see Table 8.4 below). While technically feasible and with surface wetlands inside these basins being exempt from permitting, this alternative is not considered prudent due to the much greater cost.

Underground disposal in the flooded White Pine mine workings –
Golder and Highland investigated the possibility of partial disposal of the estimated tailings production from a combined Copperwood-White Pine Project, in conjunction with surface disposal in one of the other alternatives for the volume not able to be placed in the underground mine workings. The estimate of underground void space available in the mine workings could potentially accommodate up to 37,000,000 tonnes of tailings, which is well over the Copperwood-only tailings amount. However, the Golder tradeoff report identified several significant disadvantages to this alternative: 1) Undetermined technical difficulty with physically placing the required amount of tailings in the available voids, 2) A high operating cost associated with an extensive system of boreholes into the flooded mine workings, distribution pipelines and pumps, 3) Displacement to surface, and treatment prior to discharge, of the mine water that is known to have very elevated amounts of dissolved solids and chloride, and 4) A 3rd party company is occupying a portion of the unflooded mine workings and has control over the flooded voids and existing access shafts and bore holes. For these reasons, this alternative for disposal of the Copperwood tailings at White Pine is not considered feasible or prudent.

Construction of a new surface TDF facility in the vicinity of the White Pine Mine Site –
Golder's investigation of three new surface TDF locations led them to the conclusion that one of their locations had a capital and operating cost similar to using the North #2 Basin, their other preferred White Pine alternative. They also identified significant wetland and stream filling issues based on publicly available data, i.e. no field delineations. For this reason, a new surface TDF at White Pine is not considered feasible at White Pine.

8.8 Summary of TDF Alternatives

TDF Alternative 4B was selected as the only feasible and prudent (practicable) alternative for the proposed TDF. Due to the factors discussed in the TDF analysis, the current mine plan incorporates the use of a TDF for the deposition of 100% of the tailings produced during the mine life. Table 8-4 provides a summary of the pertinent attributes of each of the TDF alternatives that were evaluated.

TDF Alternative 1, underground tailings disposal, is not feasible or prudent due to safety of miners and the excessive costs associated with disposal of only a portion of the tailings underground in mine workings. In addition, for reasons explained in this document, a TDF is still required for any of the underground tailings disposal options and the reduction in wetland or stream impacts is not commensurate with the high cost of implementing any of the underground tailings disposal options and is not prudent.

TDF Alternative 2, the south on-site TDF alternative location, is not feasible or prudent because of the high dike height (181 feet) from the basis of visibility, safety, and cost; the TDF being located furthest from the mill, which will require longer tailings pipelines resulting in cost, operational, and safety concerns; and the procurement of an estimated 9.9 million cubic yards of low permeability soil from another location for the construction of the TDF dikes would very likely impact substantial acreage of additional wetlands and to transport the low permeability soil to the TDF site would add over $100 million to the cost of the TDF compared to the proposed alternative. TDF Alternative 2 has the lowest wetland impact (6.03 acres) and the lowest stream impacts (3,706 LF) but is not prudent or practicable due to the exceedingly higher cost compared to the proposed alternative TDF.

TDF Alternative 3, the center on-site TDF alternative location, is not feasible or prudent due to the high dike height (175 feet) having visibility, safety, and cost implications; and the procurement of an estimated 9.65 million cubic yards of low permeability soil from another location for the construction of the TDF dikes would very likely impact substantial acreage of additional wetlands and to transport the low permeability soil to the TDF site would also add over $100 million to the cost of the TDF compared to the proposed alternative. Alternative 3 has the second-lowest amount of wetland impacts (11.98 acres) and the second-lowest stream impact (5,108 LF) but is not prudent or practicable due to the exceedingly higher cost compared to the proposed alternative TDF.

TDF Alternative 4, the north on-site TDF alternative location, is a feasible and prudent alternative because the stratigraphy provides adequate low permeability soil thickness to provide the low permeability soil needed for dike construction and to restrict seepage from the TDF to the groundwater; the TDF location is closest to the mill site, which provides an efficiency of operation and consolidates impacts on the landscape; the dikes would be the lowest of the on-site TDF locations, which reduces the visibility of the dikes from area vantage points and is less costly to build and maintain. TDF Alternative 4 provides a water storage facility for mine water, surface runoff contact water and process water in close proximity to mine operations. TDF Alternative 4 has estimated wetland impacts of 51.27 acres and stream impacts of 16,557 linear feet. TDF Alternative 4 has not been selected due to TDF Alternative 4B being a more desirable alternative for providing an area that accommodates construction of a natural stream channel around the west side of the TDF.

TDF Alternative 4A, a reduced-size version of TDF Alternative 4, is not prudent due to excessive costs. While it would result in less direct on-site wetland impacts (third lowest wetland impact at 41.11 acres), there would likely be indirect impacts to those wetlands that would be avoided by this alternative. Stream impacts are estimated to be approximately the same as Alternatives 4 and 4B at 16,557 linear feet. In addition, however, there are potential resource impacts associated with obtaining an additional 900,000 cubic yards (when compared to Alternative 4B) of dike material from off-site locations. For these reasons along with the higher costs associated with this alternative, Alternative 4A has been determined to not be feasible or prudent.

TDF Alternative 4B, which is essentially Alternative 4 with the west TDF berm moved east about 100 feet to provide area for stream relocation using natural stream channel design on the west side of the TDF, is the proposed TDF alternative. Although TDF Alternative 4B is second-highest in wetland impact at 51.25 acres, has the highest stream impacts (the same as Alternatives 4 and 4A), and is $2.8 million more cost than Alternative 4, the benefit of having the stream relocation on the west side of the TDF with enough room to implement natural stream channel design instead of just having a trapezoidal ditch makes TDF Alternative 4B the preferred alternative.

TDF Alternative 4C, which is essentially Alternative 4 with the east TDF berm moved west to allow for a natural stream condition to be designed and constructed between the outer toe of the TDF and the East Branch of Gypsy Creek, has the next-to-highest wetland impact (46.57 acres) and the third-lowest stream impact (9,639 linear feet). The height of the berms for this alternative is the highest of all alternatives at 182 feet, which would cause visual issues. The need for an additional 11,800,000 cubic yards (when compared to Alternative 4B) of low-permeability soil from off-site that would be needed to complete TDF construction weighs substantially against this alternative. The cost of construction for Alternative 4C is the third-highest of the on-site alternatives ($201 million). For these reasons, Alternative 4C has been determined to not be feasible or prudent.

TDF Alternative(s) 5, off-site location(s), is not feasible or prudent due to the lack of suitable locations for a TDF as demonstrated by the GIS site search conducted to assess this alternative. Therefore, an off-site TDF location is not available.

TDF Alternative 6, on-site over the mine workings is not a safe alternative.

TDF Alternative 7, White Pine has various options, but most are not prudent due to excessive costs and others are both neither prudent nor technically feasible.

8.9 Additional Description of Construction of the Proposed TDF

Although the final construction details of the TDF have not yet been completed, the BFS provides pertinent details regarding the proposed construction of the TDF (Appendix H) and its footprint will not be changed. Note that some of the components of the TDF described in the 2012 BFS have been or are being revised, e.g. the depth of excavation within the TDF. However, the discussion in the BFS may be helpful in understanding the process involved in constructing the TDF until the updated 2018 feasibility study report is available.

Table 8-4. TDF Feasibility Matrix									
	Potential Tailings Disposal Options								
Site Considerations	**Direct Shipment for Off-site Processing**	**Revised Alternative 1. Underground Disposal**	**Revised Alternative 2. On-site South TDF**	**Revised Alternative 3. On-site Central TDF**	**Revised Alternative 4. North On-site TDF**	**Alternative 4A. North On-site TDF**	**Alternative 4B. North On-site TDF**	**Alternative 4C. North On-site TDF**	**Revised Alternative 5. Off-site TDF Location**
Dike Height	Not Applicable (NA)	NA; TDF still required	181’	175’	137’	145’	140.5’	182’	No feasible or prudent locations available
TDF Size	TDF still needed at some other location	192 ac. (based upon 60% of TDF needed)	267 ac.	284 ac.	349 ac.	317 ac.	340 ac.	301 ac.	NA
Mill Proximity	Distance to ore processing mill affects transportation costs	Close to mill	Furthest on-site TDF from the mill	Second-closest to the mill of on-site TDFs	Closest on-site TDF to mill	Closest on-site TDF to mill	Closest on-site TDF to mill	Closest on-site TDF to mill	NA
Soil Conditions	NA	NA	Soils suitable; however, bedrock is much shallower which requires a substantial amount of low permeability soil from other sources.	Soils suitable; however, bedrock is much shallower which requires a substantial amount of low permeability soil from other sources.	Optimal soil conditions with up to 100’ of low permeability low permeability soil at the center of the TDF	Optimal soil conditions with up to 100’ of low permeability clay at the center of the TDF	Optimal soil conditions with up to 100’ of low permeability clay at the center of the TDF	Optimal soil conditions with up to 100’ of low permeability clay at the center of the TDF	NA
Bedrock Depth	NA	NA	See above	See above	Bedrock depths suitable for TDF construction with storage capacity needed	Bedrock depths suitable for TDF construction with storage capacity needed	Bedrock depths suitable for TDF construction with storage capacity needed	Bedrock depths suitable for TDF construction with storage capacity needed	NA
Off-site low permeability soil needed to construct	NA	NA; TDF still required	9.9 million cu.yds.	9.65 million cu.yds.	0.2 million cu.yds.	1.7 million cu.yds.	0.8 million cu. yds.	12.6 million cu. yds.	NA
Wetland Impacts	May be wetland impacts at another mill and TDF location	None underground; TDF still required	6.03 acres	11.98 acres	51.27 acres	41.11 acres	51.25 acres	46.57 acres	NA
Stream Impacts	May be stream impacts at another mill and TDF location	None underground; TDF still required	3,706 LF	5,108 LF	16,557 LF	16,557 LF	16,557 LF	9,639 LF	NA

Site Considerations	Potential Tailings Disposal Options								
	Direct Shipment for Off-site Processing	**Revised Alternative 1. Underground Disposal**	**Revised Alternative 2. On-site South TDF**	**Revised Alternative 3. On-site Central TDF**	**Revised Alternative 4. North On-site TDF**	**Alternative 4A. North On-site TDF**	**Alternative 4B. North On-site TDF**	**Alternative 4C. North On-site TDF**	**Revised Alternative 5. Off-site TDF Location**
Construction Costs (includes closure costs)	Processing costs still incurred at another location	$81 million (Includes 60% of TDF and $20 million paste plant)	$216 million	$214 million	Orvana 2012 Capex = $102.27 million, includes closure at $20.82 million	$116.2 million	GMining preliminary capex = $105.89 million includes closure cost of $28.80 million	$201 million	NA
Operation Costs	Significant cost to truck 7,500 tons of raw ore/day	$84 million (Includes paste plant O/M, binder, and WWTP costs)	$3.0 million per year[1]	$3.0 million per year[2]	$3.0 million per year[3]	$3.0 million per year[4]	$3.0 million per year[5]	$3.0 million per year[6]	NA[7]
2012 IRR [8]	4.6	7 8.5 10.5	5	5.1	11.1	10.3	10.9 (IRR for 2012 capex of $105 million)	5.3	NA[7]
Feasible or Prudent (see text for explanation)	No	No	No	No	Yes	No	Yes	No	No; alternative locations not available

Table 8-4 Footnotes

[1] Annual Opex for Alternative 2 is similar to the preferred TDF location as described in footnote 3.
[2] Annual Opex for Alternative 3 is similar to the preferred TDF location as described in footnote 3.
[3] Annual Opex for the preferred TDF location (Alternative 4) is from the BFS report as follows:
Power-Tables 21.12 & 21.13 (p. 240) = $2,100,000
Labor-Table 21.14 (p. 241) = $438,256
Flocculant-Table 21.15 (p.241) = $157,500
Golder TDF Opex-Section 21.3.4 (p.246) = $308,000
Annual TDF Opex = $3,003,756
[4] Annual Opex for the preferred TDF location (Alternative 4) is from the BFS report as follows:
Power-Tables 21.12 & 21.13 (p. 240) = $2,100,000
Labor-Table 21.14 (p. 241) = $438,256
Flocculant-Table 21.15 (p.241) = $157,500
Golder TDF Opex-Section 21.3.4 (p.246) = $308,000
Annual TDF Opex = $3,003,756
[5] Annual Opex for the preferred TDF location (Alternative 4) is from the BFS report as follows:
Power-Tables 21.12 & 21.13 (p. 240) = $2,100,000
Labor-Table 21.14 (p. 241) = $438,256
Flocculant-Table 21.15 (p.241) = $157,500
Golder TDF Opex-Section 21.3.4 (p.246) = $308,000
Annual TDF Opex = $3,003,756
[6] Annual Opex for the preferred TDF location (Alternative 4) is from the BFS report as follows:
Power-Tables 21.12 & 21.13 (p. 240) = $2,100,000
Labor-Table 21.14 (p. 241) = $438,256
Flocculant-Table 21.15 (p.241) = $157,500
Golder TDF Opex-Section 21.3.4 (p.246) = $308,000
Annual TDF Opex = $3,003,756
[7] No feasible or prudent location for an off-site TDF was found. Therefore, there is no basis to establish costs and therefore an IRR.
[8] IRR's based on 2012 BFS, updated economic analyses not available until late May 2018.

9.0 TDF SURFACE WATER DIVERSION AND STREAM RELOCATIONS

The construction of the proposed TDF requires the diversion of approximately 600 acres of existing surface water runoff and stream flows from the south as well as an additional 348 acres within the footprint of the TDF itself. The construction of the ultimate TDF footprint will occur over a period of years. When the TDF is constructed to its final footprint, two watersheds are directly affected; Lehigh Creek and Gipsy Creek. Gipsy Creek can be further divided into sub-watersheds of west, middle and east branches. Surface water draining from the south towards the TDF will be captured in channels south of the access road and diverted around the west and east sides of the TDF. Surface waters of the Lehigh Creek watershed and Middle and West Branch Gipsy Creek sub-watershed are proposed to be diverted around the west side of the TDF, while surface waters of the East Branch Gipsy Creek are proposed to be diverted around the east side of the TDF. During the active life of the TDF, approximately 280 acres will be contained within the interior of the TDF embankments. Following closure of the TDF, the TDF cap will be graded to drain to the Middle and West Gypsy Creek branches through two outlets in the north TDF embankment (Figure 9-1a). Drainage from the cap would be into two large concrete spillways/down chutes at locations similar to those in the 2013 permit. Changes in the watershed areas contributing to these four streams can be characterized as pre-TDF and post-TDF watershed areas for the portion of the watersheds upstream from the TDF and for the total watershed of each stream as shown in Table 9-1.

Table 9-1. Pre and Post-Closure Drainage Areas at the Copperwood Project

	Upstream Watershed (sq. mi.)			Total Watershed (sq. mi.)		
Stream	**Pre-TDF**	**Post-TDF**	**Change**	**Pre-TDF**	**Post-TDF**	**Change**
Lehigh Creek	0.64	0.98	+53%	0.91	1.25	+37%
W. Br. Gipsy Creek	0.35	0.22	-37%	0.60	0.47	-22%
M. Br. Gipsy Creek	0.23	0.20	-13%	0.67	0.64	-4%
E. Br. Gipsy Creek	0.35	0.26	-26%	2.90	2.81	-3%

The west and east diversion watersheds discussed in this section are shown in Figure 1 (Pre-Closure Drainage Areas) and Figure 2 (Post-Closure Drainage Areas), provided in Appendix I of this document.

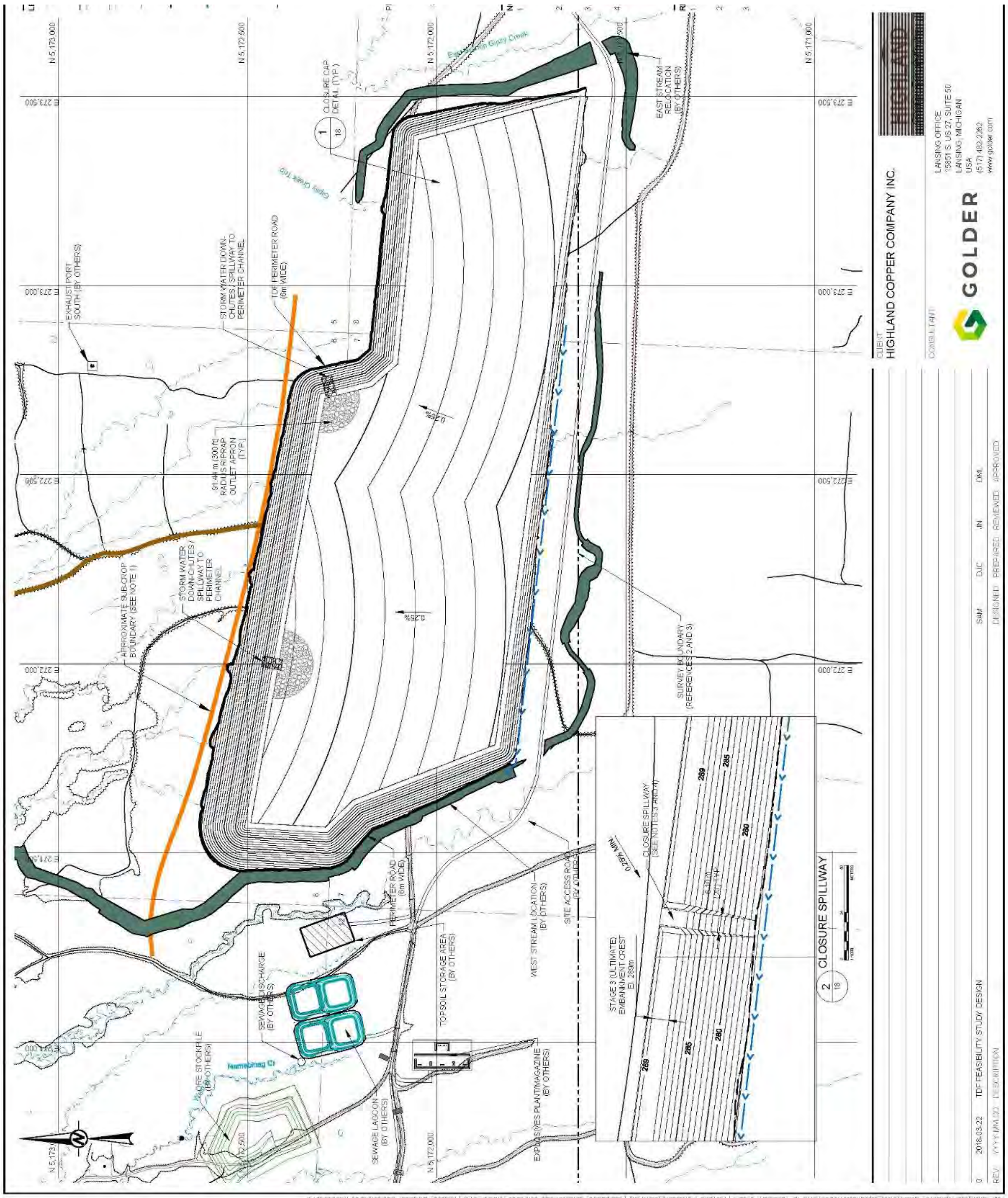


Figure 9-1a. Proposed Post-closure TDF Drainage to West Stream Diversion

The post-TDF changes in watershed areas may affect the hydraulic, geomorphology, physicochemical and biology functions of these streams. Given the lack of a regional curve for watersheds of this small size, a site-specific regional curve was prepared from geomorphic assessments of seven on-site streams and their contributing watersheds.

Based on the “Study Area Channel Dimensions Derived from Curve” table (see Stream Impact and Mitigation Summary), very little hydraulic change would be anticipated to the bankfull flows of the East and Middles Branches of Gipsy Creek, given the small changes in total watershed. The reduction in watersheds and the bankfull flows of the West Branch of Gipsy Creek will reduce flow volumes, floodplain connectivity and sediment transport; resulting in some reduction of bankfull channel dimensions in the upstream portions of these streams. Increased flows to Lehigh Creek may create an increase of the bankfull channel of approximately seven percent, resulting in increased sediment transported downstream, much of which is anticipated to be captured in existing beaver ponds.

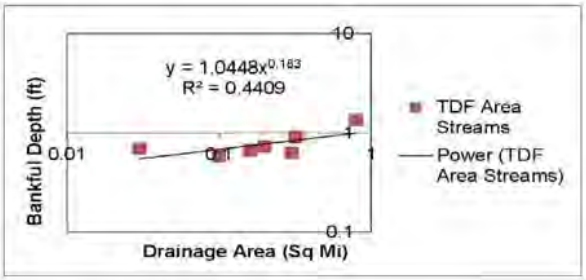
Given the stream functions present which have been assessed to be relatively low due to the ephemeral, and in their lower reaches perhaps intermittent, nature of these streams, impacts to stream functions are expected to be low, with diminishing influence as the upstream changes become an increasingly smaller portion of the downstream contributing watershed. Stream functions are assessed in further detail in the Stream Impact and Mitigation Summary.

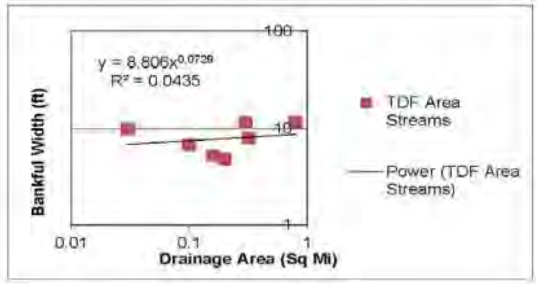
Following the determination that surface water diversion around the TDF was feasible, further evaluation was given to the potential use of natural channel design techniques for the two diversion channels on the west and east sides of the TDF. Surface water will be collected in channels on the south side of the mine access road (Figure 9-1b). The west diversion was made possible by shifting the west berm of the TDF to the east and replacing the lost storage capacity of the TDF by raising the height of the berms. The east diversion bankfull channel length is approximately 3,800 feet while the west diversion, including the channel segments south of the access road, is approximately 9,900 feet in length, for a total of 13,700 feet of new natural stream design channels (see Stream Relocation plans). Using natural stream channel design, the proposed Rosgen channel types are similar to the stream channels to be impacted by the TDF. The replacement channels will have similar hydrology and have been designed with similar physical characteristics in regard to slope, bank full channel dimensions, belt width and sinuosity based on measurements made of the on-site Rosgen E and B stream types.

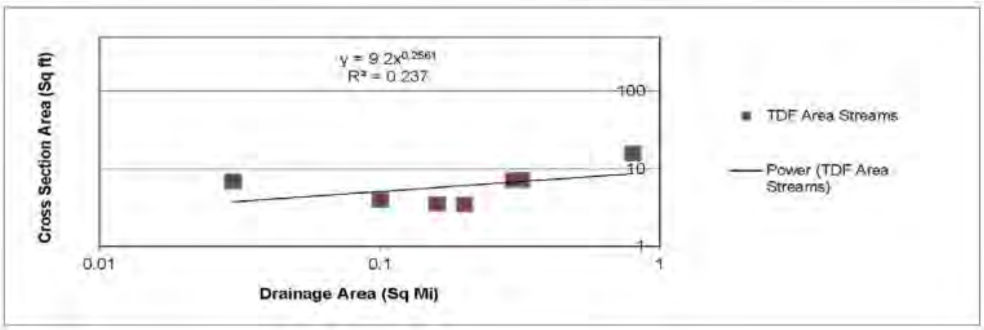
Orvana TDF Area Streams "Regional Curve"

TDF Area Stream Bankfull Channel Measurements

Stream	Drainage Area (sq. mi.)	Depth (ft.)	Width (ft.)	Cross Section (sq. ft.)
East Branch Gipsy Creek	0.32	0.91	7.91	7.23
Lehigh Creek	0.2	0.73	4.77	3.47
Namebinag	0.8	1.35	11.61	15.72
Trib B Lehigh	0.16	0.67	5.22	3.51
Trib C Lehigh	0.03	0.7	9.81	6.88
Trib D Lehigh	0.1	0.59	6.76	3.97
West Branch Gipsy	0.3	0.63	11.44	7.15

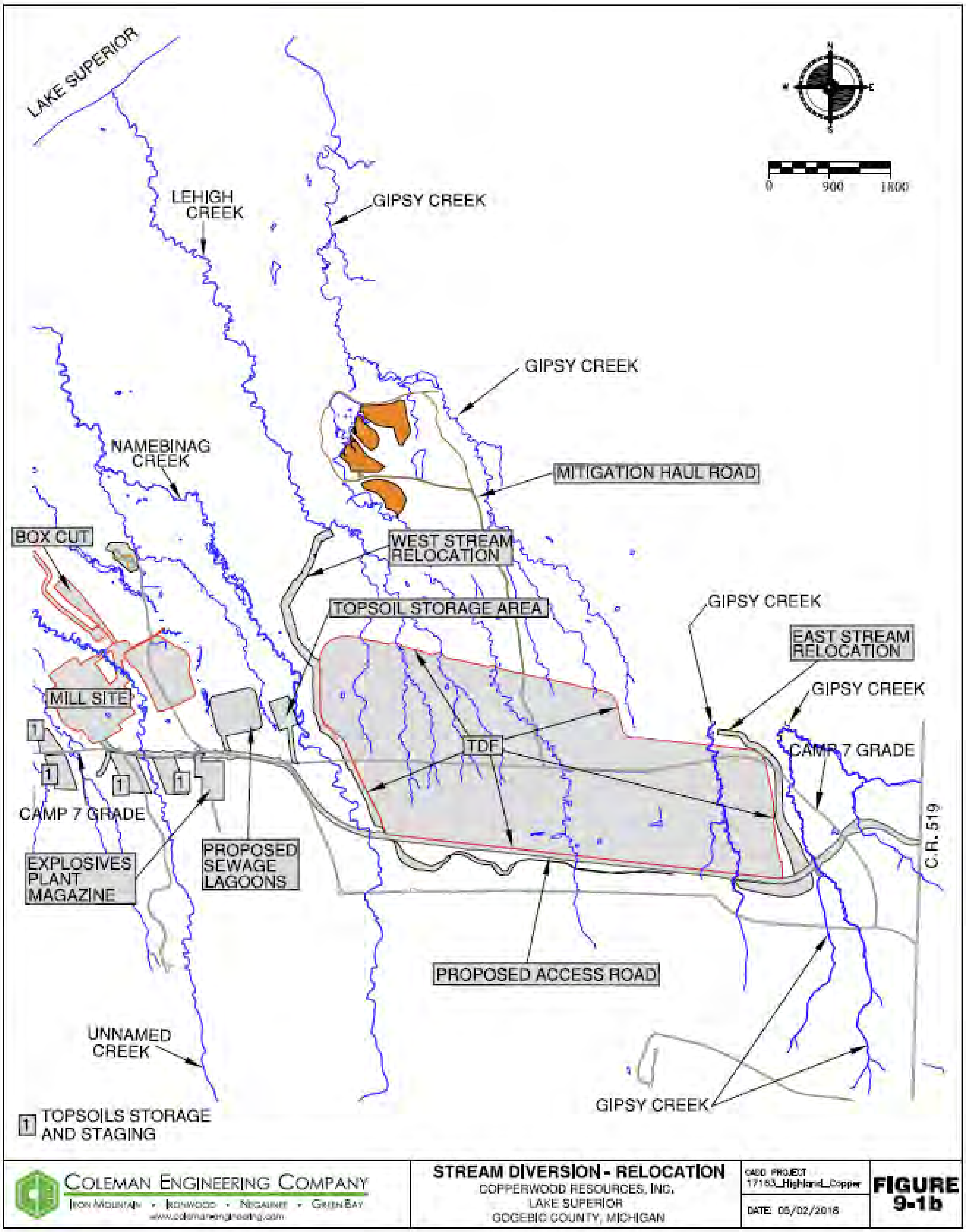






Study Area Bankfull Channel Dimensions Derived from Curve

Drainage Area (Sq Mi)	Depth (ft)	Width (ft)	Cross Section (sq ft)
0.3	0.84	8.07	6.76
0.4	0.88	8.24	7.28
0.5	0.92	8.37	7.70
0.6	0.95	8.48	8.07
0.7	0.98	8.58	8.40
0.8	1.00	8.66	8.69
0.9	1.02	8.74	8.96
1.0	1.04	8.81	9.20
1.1	1.06	8.87	9.43
1.2	1.08	8.92	9.64
1.3	1.10	8.98	9.84
1.4	1.11	9.02	10.03
1.5	1.13	9.07	10.21


LAKE SUPERIOR
LEHIGH CREEK
GIPSY CREEK
0 900 1800
GIPSY CREEK
NAMEBINAG CREEK
MITIGATION HAUL ROAD
BOX CUT
WEST STREAM RELOCATION
TOPSOIL STORAGE AREA
GIPSY CREEK
EAST STREAM RELOCATION
GIPSY CREEK
MILL SITE
TDF
CAMP 7 GRADE
CAMP 7 GRADE
C.R. 519
EXPLOSIVES PLANT MAGAZINE
PROPOSED SEWAGE LAGOONS
PROPOSED ACCESS ROAD
UNNAMED CREEK
GIPSY CREEK
1 TOPSOILS STORAGE AND STAGING
COLEMAN ENGINEERING COMPANY
IRON MOUNTAIN • IRONWOOD • NEGAUNEE • GREEN BAY
www.colemanengineering.com
STREAM DIVERSION - RELOCATION
COPPERWOOD RESOURCES, INC.
LAKE SUPERIOR
GOGEBIC COUNTY, MICHIGAN
CADD PROJECT 17163_Highland_Copper
DATE: 05/02/2018
FIGURE 9-1b

10.0 MINE VENTILATION RAISES AND ACCESS ROAD

The 2012 mine ventilation design used four portal openings at the base of the mine box cut ramp for both exhaust and fresh air intake openings. The ventilation exhaust was through twin fans at the base of the box cut into a 15-foot diameter duct and vertical stack extending to 35 feet above the ground surface. The height and diameter of the exhaust stack were determined by dispersion modeling to be the minimum dimensions required to meet air quality and stack velocity requirements.

As part of the feasibility design update, GMining reviewed the 2012 ventilation plan and determined that it could more effectively and efficiently assure adequate ventilation that meets health and safety requirements with a different design. The proposed updated design will use a remote intake raise to deliver fresh air to the mine with a remote exhaust raise and the mine portal serving as discharge points for the ventilation system. Air dispersion modeling for the proposed exhaust locations has demonstrated that a 10-foot diameter stack extending 25 feet above the ground surface will be sufficient to meet air quality requirements.

The location of the ventilation raises and access route was planned to avoid wetland and stream impacts as much as feasibly possible (Figure 1-1). However, the revised ventilation plan will unavoidably impact an additional 0.05 acres of wetland along the ventilation system access route. This new wetland impact represents a better alternative from the 2012 ventilation plan from both an operational and health and safety perspective for the Copperwood Project.

11.0 LAKE SUPERIOR INTAKE AND ACCESS ROAD

In the final 2012 permit application, Orvana was planning to obtain the required water supply for the Copperwood Project from the Gogebic Range Water Authority (GRWA) via a pipeline connecting to their system near Wakefield, a distance of about 17 miles that would have impacted an estimated 7.3 acres of wetlands and require 17 stream crossings. As part of the recent (2017/2018) feasibility update for the Copperwood Project, this alternative, and a total of ten other potential water supply sources, were evaluated by Copperwood and Coleman Engineering. The end result was a decision to propose a mine-only Lake Superior water intake in the vicinity of the mine site. An application for this proposed water intake was submitted to the U.S. Army Corps of Engineers (“Corps”) on March 26, 2018 for permitting under Section 10 of the Federal Rivers and Harbors Act of 1899, and is further included in the Part 325 aspect of this MDEQ permit application submittal.

The location of the proposed water intake, shoreline pump station and access road is shown on Figure 1-1. This proposed water supply option will unavoidably impact a total of 0.11 acres of wetlands and require a total of 7 stream crossings along the access route to the pump station. An analysis of water supply alternatives was submitted with the March 2018 Corps permit application, and is attached as Appendix J.

12.0 WETLAND AND STREAM MITIGATION

The proposed wetland mitigation plan includes both creation of new wetlands on-site and preservation of existing, critical off-site wetlands. The proposed stream mitigation plan includes on-site activities including stream relocations around the TDF utilizing natural channel design, channel restoration through removal of existing culverts and mine rock and restoring currently impeded fish passage through the off-site replacement of a culvert on Twomile Creek. (The Twomile Creek project was identified by the Ottawa National Forest as one of the highest priority watershed restoration projects within their jurisdiction.) The wetland and stream mitigation plans are provided in separate documents within this permit application package.

13.0 SUMMARY

The Copperwood Project is a very complex project that is very important to the economy of the western Upper Peninsula of Michigan. Logging and mining are heritage industries in the Upper Peninsula, especially in the central and western U.P. The development, culture and associated social and economic benefits in the region have been shaped by the mining and forest industries. However, starting in the early 1980s, the western U.P. of Michigan began to suffer economically, as national growth in the industrial sector stagnated and many manufacturing jobs were lost to overseas labor markets. The western U.P. suffered a major blow in the counties of Ontonagon and Gogebic with the temporary closing of the White Pine Copper Mine in 1983 and permanent closure in 1997. The White Pine mine closure had a region-wide effect with the loss of secondary jobs related to the unemployed miners' dislocation as well as the associated reduction of mining supply and service-related jobs. Many of the affected/unemployed workers and their families were forced to leave the region and seek employment elsewhere. Since that time, generally speaking, the western U.P. has been experiencing a five to ten percent reduction in total population per decade.

Then, in the early 2000s, manufacturing jobs were being created in Baraga and Ontonagon counties at a robust pace. In Baraga County alone, jobs increased by 48 percent. However, in the later years of that decade, three closures of manufacturing facilities occurred. Those closures reduced the number of employment opportunities for area residents substantially, as they totaled a loss of nearly 400 employees in the region.

As a result of these closures and other socio-economic factors, this region has some of the highest unemployment rates in the State of Michigan. According to the US Census Bureau 2016 data, the current per capita income for Baraga, Gogebic and Ontonagon Counties area is approximately $ 20,520.

Projected employment for the Copperwood project is estimated at approximately 400 permanent jobs (preliminary 2018 update estimates for life of mine) and approximately 450 construction jobs during the first two years. Preliminary salary estimates from the 2018 feasibility update, these 400 permanent jobs are expected to generate administrative payroll in the range of $3.8 million per year, mill workers payroll of approximately $5.6 million per year, and underground miners' payroll of approximately $20.2 million per year for a total estimate of $29.6 million annual payroll associated with the Copperwood Project. This annual payroll estimate does not take into account an estimated 450 construction jobs.

An independent study conducted by the University of Minnesota Duluth School of Business (Appendix K), concluded that an additional 213 service sector jobs will also be generated as a result of this new mine. Throughout the projected life of the Copperwood Project, the study anticipates that approximately $2.3 billion dollars will be injected into the local economy, including an estimated increase in federal and state/local tax revenue of $8.5 million and $65 million, respectively. As demonstrated by these economic factors and studies, the Copperwood Project will be a significant factor in boosting the economy of the depressed western U.P.

The landscape where the Copperwood Project site is located is conducive to construction and operation of the mine for the following reasons:

- Suitable low permeability soil soils for the required TDF;
- Ore is located relatively near to the land surface, facilitating reasonable, cost- effective access;
- Relatively isolated location, although portions of the TDF will be visible from several area vantage points.

Natural resource impacts are extremely important, and there are many associated with this project. The on-site natural resources consist of:

- Abundant wetlands that are dependent upon surface water runoff and are of similar ecological value as the adjacent uplands;
- Stream reaches that are ephemeral and have minimal value to aquatic organisms;
- Common types of wildlife habitat, as such habitats are an abundant resource in the western U.P.;
- Redside dace that are found in the lower reaches of Namebinag Creek and Unnamed Creek. Measures will be implemented to protect and monitor this fish, which is a State Endangered species.

Alternatives for each of the proposed activities associated with the Copperwood Project have been and continue to be fully evaluated to demonstrate compliance with State and Federal statutes and thereby effect MDEQ permit issuance. Copperwood is committed to providing any additional information as deemed necessary for the review of this permit application in order to effect issuance of the permit under Parts 301 and 303, a permit that will be exceptionally similar in content to previously issued MDEQ Permit Number 12-27-0050-P that has recently expired.

14.0 APPENDICES

Appendix A: Orvana's March 2012 Bankable Feasibility Study

Appendix B: MHF Services Inc. Rail Transportation and Transload Study

Appendix C: MTU Ore Slurry Pipeline Design Study

Appendix D: MTU Portal and Tunnel Optimization Design Study

Appendix E: Portal and Surface Facilities Encumbered Area & Wetland Delineation Maps

Appendix F: Golder Associates Draft Tailings Tradeoff Study, White Pine Copper Project

Appendix G: Golder Associates Technical Analysis of Underground Tailings Disposal

Appendix H: Contour map showing hydrostatic uplift isobars for the TDF floor grades

Appendix I: TDF Pre- and Post-Closure Drainage Areas

Appendix J: Water Supply USACE Alternatives Analysis

Appendix K: Economic Impact of Orvana's Copperwood Project, Upper Peninsula